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Module 1: Introduction

1.1 Introduction
The information necessary to pass this course will be presented in these notes and through other medium such as lectures and field trips. To guide the student to additional resources either out of interest or mandated by the course requirements, a list of additional resources are indicated after every major topic. The appearance of this list will be as seen below:

ADDITIONAL RESOURCES [NUMBER]
The above information is additionally supplemented by:
- Readings (Mandatory or Optional)
- Field trip (Always mandatory)
- Assignment
- Lecture

Important note: These class notes borrow heavily from the material assembled by:
- Dr. Paul Lever, Mr. William Cummings,

ADDITIONAL RESOURCES I
The above information for is additionally supplemented by:
- Module 1.1 Lecture

1.2 Geological Properties in Rock Excavation
A brief review of rock properties is required to provide students without this background or as a review.

1.2.1 Minerals and Rocks
The earth's crust consists of a variety of rocks, formed under different circumstances. Rocks consist of one or more composite minerals. A mineral is a substance formed by nature. A mineral may be an element or may consist of chemical compounds containing several elements. There are more than 3,000 different minerals.

Of 103 known elements, oxygen is by far the most common, making up about 50 per cent of the earth's crust. Silicon, about 25 per cent, aluminum, iron, calcium, sodium, potassium, magnesium and titanium, together with oxygen, account for about 99 per cent.

1 From SECOROC's Geology Drilling Theory. Document S-96174
Silicon, aluminum and oxygen occur in our most common minerals, quartz, feldspar and mica. They form the large group known as silicates, a silicate being a compound formed of silicic acid and other elements. Also included are amphiboles and pyroxenes, which contain aluminum, potassium and iron. Some of the earth’s most common rocks, granite and gneiss, are composed of silicates.

Figure 1-1: Origins of various rocktypes

Oxygen often occurs in combination with metallic elements and forms our oxidic ores (the iron ores magnetite and hematite). Sulphur readily combines with metallic elements and forms sulphide ores (galena, sphalerite, molybdenite, arsenopyrite). Halogenides (fluorite, halite), carbonates (calcite, dolomite, malachite), sulphates (barite), tungstates (scheelite) and phosphates (apatite) are other large mineral groups. Gold, silver, copper and lead are elements that can occur as native metals. Feldspar accounts for
almost 50 per cent of the mineral composition of the earth's crust. Next come the pyroxene and amphibole minerals, closely followed by quartz and mica. These minerals make up about 90 per cent of the composition. Some of the characteristics of the minerals are hardness, density, color, streak, lustre, fracture, cleavage and crystalline form.

Hardness can be graded according to the Moh's 10-point scale. (example followed by test)
1. Talc - Easily scratched with the fingernail
2. Gypsum - Just barely scratched with the fingernail
3. Calcite - Very easily scratched with a knife
4. Fluorite - Easily scratched with a knife
5. Apatite - Can be scratched with a knife
6. Orthoclase - Hard to scratch with a knife, can be scratched with quartz
7. Quartz - Scratches glass, can be scratched with a hardened file
8. Topaz - Scratches glass, can be scratched with emery
9. Corundum - Scratches glass, can be scratched with a diamond
10. Diamond - Scratches glass

Molybdenite, hardness 1.5, blackens a thumb that is rubbed against it. The density of light-colored minerals is for the most part below 3.0. Exceptions are barite or heavy spar (BaSO4), density 4.5, scheelite(CaWO4), density 6.0, and cerussite (PbCO3), density 6.5 Dark-colored minerals with some iron, silicates, have densities between 3.0 and 4.0. Ore minerals have densities over 4.0. Gold has a very high density at 19.3, and tungsten at 19.4. The highest density is shown by osmium and iridium, 22.5. Streak is the color of the mineral powder produced when the mineral is scratched against unglazed, white porcelain (e.g. an ordinary electric fuse).

Fracture is the surface produced by breaking off a piece of mineral, not following a crystallographically defined plane. Fracture is usually uneven in one way or another.

Cleavage denotes the properties of a crystal whereby it allows itself to be split along flat surfaces parallel with certain formed or otherwise crystallographically defined surfaces.

1.2.2 The properties of rocks
In order to be able to forecast the result of drilling in respect of penetration rate, hole quality, drill-steel costs, etc., we must be able to make a correct appraisal of the rock concerned. In doing so we distinguish between microscopic and macroscopic properties. A rock is composed of grains of various minerals, and among the microscopic properties are mineral composition, grain size, and the form and distribution of the grains. Taken together, these factors decide important properties of the rock, such as hardness, abrasiveness, compressive strength and density. These properties,
in their turn, determine the penetration rate that can be achieved and how heavy the tool wear will be.

The drillability of a rock depends on, among other things, the hardness of its constituent minerals and on the grain size and crystal form. Quartz is one of the commonest minerals in rocks. Since quartz is a very hard material, a high quartz content (SiO$_2$) makes the rock very hard to drill and causes heavy wear, particularly on the drill bits. We say that the rock is abrasive. Conversely, a rock with a high content of calcite is easy to drill and causes little wear on the drill bits.

As regards crystal form, minerals with high symmetry, e.g. cubic (galena) are easier to drill than minerals with low symmetry, e.g. fibrous (amphiboles and pyroxenes).

A coarse-grained structure is easier to drill in and causes less wear than a fine-grain structure. Consequently, rocks with essentially the same mineral content may be quite different as regards drillability. For example, quartzite may be fine-grained, grain size 0.5-1 mm, or dense, grain size 0,05 mm, while a granite may be coarse-grained, grain size> 5 mm, medium-grained (grain size 1-5 mm) or fine-grained (grain size 0.5-1 mm).

A rock can also be classified on the basis of its structure. If the mineral grains are mixed in a homogeneous mass, the rock is massive (e.g. granite). In mixed rocks the grains are arranged in layers. A slaty rock also has the minerals arranged in different layers, but in this case pressure and heat have compacted each layer in plates.

Among the macroscopic properties are slatiness, fissuring, contact zones, layering, veining and inclination. These factors are often of great significance in drilling. For example, cracks or inclined and layered formations can cause hole deviation and can occasionally cause drilling tools to get stuck.

Soft rocks make it difficult to achieve good hole quality, since the walls often cave in and in extreme cases the flushing air disappears into cracks in the rock without reaching the surface. The enormous variety of rocks and rock formations makes it impossible to give the subject adequate treatment in just a few pages. We must therefore content ourselves with giving a summary description of the more important rocks and referring interested readers to the literature of the subject for further study.

Rocks are classified into three main groups on the basis of their origin and the way in which they were formed:

1. Igneous or magmatic rocks (formed from solidified lava or "magma").
2. Sedimentary rocks (formed by deposition of broken material or by chemical precipitation).
3. Metamorphic rocks (formed by the transformation of igneous or sedimentary rocks, in most cases by an increase in pressure and heat).

1.2.2.1 Igneous rock

Igneous rocks are formed when a magma solidifies deep down in the earth's crust (plutonic rock), or as it rises towards the surface (dyke rock) or on the surface (volcanic rock). The most important constituents (minerals) are quartz and silicates of various composition, chiefly feldspars. Plutonic rocks solidify slowly and are therefore coarse-grained, while volcanic rocks solidify quickly and become fine grained.

Depending on whether the magma solidifies at depth, or as a dyke rock, or on the surface, the rock is given different names even if the composition is the same. This is evident from the table below, which also shows that it is customary to classify the igneous rocks by their silicon content (SiO$_2$). The greater the silicon content, the larger the amount of quartz in the rock will be.

<table>
<thead>
<tr>
<th>SiO$_2$</th>
<th>Plutonic</th>
<th>Dykes</th>
<th>Volcanic</th>
</tr>
</thead>
<tbody>
<tr>
<td>Basic</td>
<td>Gabbro</td>
<td>Diabase</td>
<td>Basalt</td>
</tr>
<tr>
<td>&lt;52% SiO$_2$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Intermediary</td>
<td>Diorite</td>
<td>Porphyrite</td>
<td>Andesite</td>
</tr>
<tr>
<td>52-65% SiO$_2$</td>
<td>Syenite</td>
<td>Syenite porphyry</td>
<td>Trachyte</td>
</tr>
<tr>
<td>Acid</td>
<td>Quartz diorite</td>
<td>Quartz porphyrite</td>
<td>Dacite</td>
</tr>
<tr>
<td>&gt;65% SiO$_2$</td>
<td>Granite</td>
<td>Granodiorite porphyry</td>
<td>Rydodacite</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Quartz porphyry</td>
<td>Rhyolite</td>
</tr>
</tbody>
</table>

1.2.2.2 Sedimentary rocks

Sedimentary rocks are formed by the deposition of material by mechanical or chemical action and a consolidation of this material under the pressure of overlying layers. It frequently occurs that the rock formation is broken down by mechanical action (weathering), carried away by running water and deposited in still water. Thus the original rock will determine the characteristics of the sedimentary rock. The weathering or erosion may proceed at different rates depending partly on climate and partly on how easily the rock breaks up.

Sedimentary rocks can also arise as a result of the chemical precipitation of minerals or by proliferation of organic organisms, as for example in coral reefs or carbon deposits. Since formation takes place by deposition, several distinct layers can often be observed in a sedimentary formation. Sedimentary rocks make up a very heterogeneous family with widely varying characteristics.
Table 1-2 Sedimentary Rocks

<table>
<thead>
<tr>
<th>Rock</th>
<th>Original material</th>
</tr>
</thead>
<tbody>
<tr>
<td>Conglomerate</td>
<td>Gravel, stones, boulders</td>
</tr>
<tr>
<td>Greywacke</td>
<td>Clay, gravel</td>
</tr>
<tr>
<td>Sanstone</td>
<td>Sand</td>
</tr>
<tr>
<td>Argillaceous schist</td>
<td>Clay</td>
</tr>
<tr>
<td>Aluminous slate</td>
<td>Clay plus organisms</td>
</tr>
<tr>
<td>Limestone</td>
<td>Calcium carbonate or various organisms</td>
</tr>
</tbody>
</table>

1.2.2.3 Metamorphic rocks

The effects of chemical action or increased pressure and/or temperature on a rock formation can sometimes be so great that it produces a transformation, which the geologist calls metamorphism. For example, pressure and temperature might increase under the influence of up-welling magma, or because the formation has sunk down deeper into the earth's crust. This results in the recrystallization of the mineral grains or the formation of new minerals. A characteristic of the metamorphic rocks is that they are formed without any complete melting. They are also frequently hard or very hard, and also compact and fine-grained, and are therefore often difficult to drill.

The earth's crust is in a constant state of flux, and the various rock formations may be subjected to very powerful forces. The result is deformation in one way or another, and in nature we may therefore observe, for example, pronounced folding, crushed zones, faults and other phenomena that can have a telling effect on drilling.

Table 1-3: Metamorphic rocks

<table>
<thead>
<tr>
<th>Rock</th>
<th>Original Rock</th>
<th>Degree of metamorphism</th>
</tr>
</thead>
<tbody>
<tr>
<td>Amphibolite</td>
<td>Basalt, diabase, gabro</td>
<td>High</td>
</tr>
<tr>
<td>Mica-schist</td>
<td>Mudstone, greywacke, etc.</td>
<td>Medium/high</td>
</tr>
<tr>
<td>Gneiss</td>
<td>Various igneous rocks</td>
<td>High</td>
</tr>
<tr>
<td>Greenschist</td>
<td>Basalt, diabase, gabbro</td>
<td>Low</td>
</tr>
<tr>
<td>Quartize</td>
<td>Sandstone</td>
<td>Medium/high</td>
</tr>
<tr>
<td>Lepitite</td>
<td>Dacite</td>
<td>Medium</td>
</tr>
<tr>
<td>Slate</td>
<td>Shale</td>
<td>Low</td>
</tr>
<tr>
<td>Veined gneiss</td>
<td>Silicic-acid-rich silicate rocks</td>
<td>high</td>
</tr>
</tbody>
</table>

1.2.3 Rock classification

Many attempts have been made to classify rocks on the basis of their drillability, and various measuring methods have been evolved with the aim
of making it possible to predict productivity and tool wear by carrying out a laboratory test before a rock job begins. The term "drillability" is used here to mean primarily the rate at which the tool penetrates into the rock, but in the wider sense it extends to the resulting hole quality, the straightness of the hole, the risk of tool jamming etc. Tool wear is often proportional to drillability, though it also depends on how abrasive the rock is.

The drillability of a given rock is determined by a number of factors. Foremost among these are the mineral composition, the grain size and the brittleness. Attempts are often made to describe drillability by stating the compressive strength or hardness; these are crude methods but they will often suffice for rough calculations. The Norwegian Technical University has developed a more sophisticated method for calculating the DRI and BWI. DRI, the drilling-rate index, describes how fast the drill steel can penetrate; it also includes on the one hand a measurement of brittleness and on the other hand drilling with a small rotating bit into a sample of the rock.

The higher the DRI, the higher penetration rate. Figure 1-2 shows how the DRI can vary from one rock to another.

Figure 1-2: Drill rate index for a variety of rocks
BWI stands for "bit-wear index", and gives an indication of how fast the bit wears down. It is determined by an abrasive test. The higher the BWI, the faster will be the wear. In most cases the BWI and the DRI are inversely proportional to each other, so that a high DRI will give a low BWI and vice-versa. However, the presence of hard minerals may produce heavy wear on the bit despite relatively good drillability. This is particularly true in the case of quartz. The quartz content has been shown to exert great influence on wear, which explains why relatively easily drilled sandstone, for example, can cause very heavy wear on the bits. In other cases, metamorphic, compact, quartz-bearing rocks may also prove to be very difficult to drill. Other examples of hard minerals that impair drillability are certain sulphides in orebodies.

1.2.4 Rock Strength

The compressive strength of many rock materials is a factor of 5 or more greater than their tensile strength. The scatter of the strength values from a series of test specimens of the same rock material is considerable. This is a result of the randomly distributed weak planes, microcracks, or flaws in the rock which greatly influence the rock strength. The flaws are often so small and the microcracks so fine that they are difficult to detect by the naked eye.

![Figure 1-3: Unconfined Compressive Strength Varying by Time.](image)

There is also an element of creep in the strength and deformation characteristics of rock, as evidenced by the ability of rock to flow and deform plastically under tectonic stress over long periods of time (hundreds of

---

thousands to millions of years). Experiments have shown that the strength of rock is time dependent, so that the compressive strength when the load is applied in 1 msec is a factor of 2 or 2.5 greater than when the load is applied in 10 sec. (Figure 1-3)

With confinement, that is when lateral expansion is restricted, rock in compression becomes stronger. This is because deformation to failure takes place as a shearing of weak planes. The action of confinement is to resist shearing, partly by creating lateral forces that resist the shear motion, and partly by increasing the friction on potential shearing surfaces by increasing the normal load thereby also increasing the rock strength.

Shear strength is composed of two parts: the friction between sliding crack surfaces characterized by a friction coefficient, \( \mu \); and the fracturing or plastic deformation of the crystal grains, which approach a limiting shear strength \( \tau_i \) when the deformation is entirely plastic.

### 1.2.4.1 Microcracks in Rock

The reasons for the strength behavior of rock and other brittle materials can be sought in the presence of microscopic cracks and flaws in the base material that is in itself strong. Most rock materials are aggregates in which separate crystal grains of different strength, different elastic and thermal moduli, and different size are cemented or grown together. Any deformation of sufficient magnitude will lead to local cracking or the development of microscopic flaws, pores, or weakened regions. Such flaws are also nearly always present in most natural rock materials because of the deformation the rock has undergone under the influence of tectonic forces and temperature changes.

In tension, microcracks grow, join, and ultimately lead to fracture at a low load. In compression, the friction on such microcracks that are stressed in shear leads to increased strength. With confinement, the crack growth is further restricted and friction is increased. This leads to a further increase in strength. In the limit, with increased hydrostatic confining pressure, we approach the real strength of the aggregate base material. It is conceivable, with a sufficiently high hydrostatic pressure, that the deformation of the weaker part of the aggregate grains will be plastic, while the hard grains still only deform elastically.

The random distribution in space, size, and direction of the micro-cracks or flaws is the reason for both the scatter of experimental strength measurement data and the dependence of strength on the size of the specimen. In a large specimen, it is more probable than in a small specimen that a sufficiently large flaw will have a direction favoring fracture in a given stress situation. Therefore, a large specimen or rock volume has a lower strength than a small specimen.
1.2.4.2 Fracture Mechanics

The most important aspect of the strength of brittle materials is their ability to break by crack propagation. Because the tensile strength of these materials is so much lower than the compressive strength, and possibly also because they already contain micro-cracks, cracks form easily and, once formed, expand because of the concentration of tensile stresses at the crack tip (Figure 1-4). We will limit this discussion to cracking under biaxial stresses, that is, stress situations where two principal stresses are equal and the third is zero. The stress concentration in front of the crack tip can be represented by the expression:

\[ \sigma(x) \approx \frac{K_I}{\sqrt{2\pi x}} \]

where the stress intensity factor \( K_I \) is a function of the crack length and the load \( \sigma_o \). The critical value of \( K_I \) when the crack just starts moving is a material constant \( K_{IC} \).

As the crack propagates, energy is absorbed by deformation work by the material at the crack tip or dissipated as elastic wave energy radiating out through the material from the crack tip. The work done per unit new crack surface is \( G_{IC} \) which is coupled to \( K_{IC} \) through the relation:

\[ G_{IC} = \frac{1 - \nu^2}{E} K_{IC}^2 \]

Figure 1-4: Crack propagation
Where $v$ is Poisson’s ratio$^3$ and $E$ is Young’s modulus.$^4$ $G_{IC}$ is called fracture toughness and is the fundamental material constant. Some authors refer to $K_{IC}$ as fracture toughness. Table 1-4 shows fracture toughness of various materials.

### Table 1-4: Fracture Toughness

<table>
<thead>
<tr>
<th>Material</th>
<th>$E$ [GPa]</th>
<th>$G_{IC}$ [J/m$^2$]</th>
<th>$K_{IC}$ [MPa m$^{1/2}$]</th>
</tr>
</thead>
<tbody>
<tr>
<td>Steel</td>
<td>210</td>
<td>10000</td>
<td>50</td>
</tr>
<tr>
<td>Aluminum</td>
<td>70</td>
<td>8000</td>
<td>25</td>
</tr>
<tr>
<td>Plexiglass</td>
<td>3</td>
<td>600</td>
<td>1.5</td>
</tr>
<tr>
<td>Granite I</td>
<td>50</td>
<td>150</td>
<td>3</td>
</tr>
<tr>
<td>Granite II</td>
<td>20</td>
<td>95</td>
<td>1.4</td>
</tr>
<tr>
<td>Marble I</td>
<td>50</td>
<td>10</td>
<td>0.7</td>
</tr>
<tr>
<td>Marble II</td>
<td>80</td>
<td>34</td>
<td>1.67</td>
</tr>
<tr>
<td>Sandstone</td>
<td>7</td>
<td>300</td>
<td>1.5</td>
</tr>
<tr>
<td>Limestone</td>
<td>15</td>
<td>65</td>
<td>1.0</td>
</tr>
<tr>
<td>Silicon Glass</td>
<td>50</td>
<td>1.2</td>
<td>0.25</td>
</tr>
</tbody>
</table>

### 1.2.4.3 Elasticity

The majority of rock minerals have an elastic-fragile behavior, which obeys the Law of Hooke, and are destroyed when the strains exceed the limit of elasticity. Depending upon the nature of deformation, as function of the stresses produced by static charges, three groups of rocks are taken into consideration:

1. The elastic-fragile or those which obey the Law of Hooke,
2. The plastic-fragile, that have plastic deformation before destruction,
3. The highly plastic or very porous, in which the elastic deformation is insignificant.

The elastic properties of rocks are characterized by the elasticity modulus '$E$' and the Poisson coefficient '$v$'. The elasticity module is the proportionality factor between the normal stress in the rock and the relative correspondent deformation, its value in most rocks varies between $0.03 \times 10^4$ and $1.7 \times 10^5$ MPa, basically depending upon the mineralogical composition, porosity, type of deformation and magnitude of the applied force. The values of the elasticity modules in the majority of sedimentary rocks are lower than those corresponding to the minerals in their composition. The texture of the rock also has influence on this parameter, as the elasticity module in the direction of the bedding or schistosity is usually larger than when perpendicular.

---

$^3$ A material stretches when pulled under tensile load and usually contracts transversely. The Poisson's ratio is the ratio of transverse strain to axial strain during axial load. For example, if a bar is pulled in the axial direction then the deformed bar (besides being longer in the axial direction) also contracts in the transverse direction by the percentage amount indicated by the Poisson's ratio.

$^4$ Young's modulus is the ratio of longitudinal stress to the resultant longitudinal strain (stress/strain). Stiffness of the material
Poisson's coefficient is the factor of proportionality between the relative longitudinal deformations and the transversal deformations. For most rocks and minerals it is between 0.2 and 0.4, and only in quartz is it abnormally low, around 0.07.

Figure 1-5: Curves of stress-deformation for different types of rocks.

1.2.4.4 Plasticity
As indicated before, in some rocks the plastic deformation precedes destruction. This begins when the stresses exceed the limit of elasticity. In the case of an ideally plastic body, that deformation is developed with an invariable stress. Real rocks are deformed and consolidated at the same time: in order to increase the plastic deformation it is necessary to increase the effort.

The plasticity depends upon the mineral composition of the rocks and diminishes with an increase in quartz content, feldspar and other hard minerals. The humid clays and some homogeneous rocks have plastic properties. The plasticity of the stony rocks (granites, schistoses, crystallines and sandstones) becomes noticeable especially at high temperatures.
1.2.4.5 Abrasiveness

Abrasiveness is the capacity of the rocks to wear away the contact surface of another body that is harder, in the rubbing or abrasive process during movement. This property has great influence upon the life of drill steel and bits. In Table 1-5, the mean amounts of quartz for different types of rock are indicated. The factors that enhance abrasive capacities of rocks are the following:

- The hardness of the grains of the rock. The rocks that contain quartz grains are highly abrasive.
- The shape of the grains. Those that are angular are more abrasive than the round ones.
- The size of the grains.
- The porosity of the rock. It gives rough contact surfaces with local stress concentrations.
- The heterogeneity. Polymetallic rocks, although these are equally hard, are more abrasive because they leave rough surfaces with hard grains as, for example, quartz grains in a granite.

Table 1-5: Relative quartz (common abrasive) content

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Quartz content (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Amphibolite</td>
<td>0-5</td>
</tr>
<tr>
<td>Anorthosite</td>
<td>0</td>
</tr>
<tr>
<td>Diabase</td>
<td>0-5</td>
</tr>
<tr>
<td>Diorite</td>
<td>10-20</td>
</tr>
<tr>
<td>Gabro</td>
<td>0</td>
</tr>
<tr>
<td>Gneiss</td>
<td>15-50</td>
</tr>
<tr>
<td>Granite</td>
<td>20-35</td>
</tr>
<tr>
<td>Greywacke</td>
<td>10-25</td>
</tr>
<tr>
<td>Limestone</td>
<td>0-5</td>
</tr>
<tr>
<td>Marble</td>
<td>0</td>
</tr>
<tr>
<td>Mica gneiss</td>
<td>0-30</td>
</tr>
<tr>
<td>Mica schist</td>
<td>15-35</td>
</tr>
<tr>
<td>Norite</td>
<td>0</td>
</tr>
<tr>
<td>Pegmatite</td>
<td>15-30</td>
</tr>
<tr>
<td>Phyllite</td>
<td>10-25</td>
</tr>
<tr>
<td>Quartzite</td>
<td>60-100</td>
</tr>
<tr>
<td>Sandstone</td>
<td>25-90</td>
</tr>
<tr>
<td>Shale</td>
<td>0-20</td>
</tr>
<tr>
<td>Slate</td>
<td>10-35</td>
</tr>
<tr>
<td>Taconite</td>
<td>0-10</td>
</tr>
</tbody>
</table>

1.2.4.6 Texture

The texture of a rock refers to the structure of the grains or minerals that constitute it. The size of the grains are an indication, as well as their shape, porosity etc. All these aspects have significant influence on drilling performance. When the grains have a lenticular shape, as in a schist, drilling is more difficult than when they are round, as in a sandstone. The type of material that makes up the rock matrix and unites the mineral grains also has an important influence. As to porosity, those rocks that have low density
and, consequently, are more porous, have low crushing strength and are easier to drill. In Table 1-6 the classification of some types of rocks is shown, with their silica content and grain size. Table 1-7 shows characteristics properties of different types of rocks according to their origins.

### Table 1-6: Common rock names and their geological definitions

<table>
<thead>
<tr>
<th>Grain size, mm</th>
<th>Texture</th>
<th>Structure</th>
<th>Sedimentary</th>
<th>Metamorphic</th>
<th>Igneous</th>
</tr>
</thead>
<tbody>
<tr>
<td>80</td>
<td>Very coarse-grained</td>
<td>Sedimentary</td>
<td>Grains are of rock fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
<tr>
<td>2</td>
<td>Coarse-grained</td>
<td>Sedimentary</td>
<td>Grains are of rock fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
<tr>
<td>0.06</td>
<td>Medium-grained</td>
<td>Sedimentary</td>
<td>Grains are many mineral fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
<tr>
<td>0.002</td>
<td>Fine-grained</td>
<td>Sedimentary</td>
<td>Grains are many mineral fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
<tr>
<td>0.0001</td>
<td>Very fine-grained</td>
<td>Sedimentary</td>
<td>Grains are many mineral fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
</tbody>
</table>

### Table 1-7: Rock Properties according to origin

<table>
<thead>
<tr>
<th>Grain size</th>
<th>Texture</th>
<th>Structure</th>
<th>Sedimentary</th>
<th>Metamorphic</th>
<th>Igneous</th>
</tr>
</thead>
<tbody>
<tr>
<td>80</td>
<td>Very coarse-grained</td>
<td>Sedimentary</td>
<td>Grains are of rock fragments</td>
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<td>Coarse-grained</td>
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<td>0.06</td>
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<td>Sedimentary</td>
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</tr>
<tr>
<td>0.002</td>
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<td>Sedimentary</td>
<td>Grains are many mineral fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
<tr>
<td>0.0001</td>
<td>Very fine-grained</td>
<td>Sedimentary</td>
<td>Grains are many mineral fragments</td>
<td>85% grains are of fine-grained igneous rock</td>
<td>Chemical-organic rocks</td>
</tr>
</tbody>
</table>
1.2.4.7 Rock Mass Strength and Structure

A rock mass, as distinguished from a rock strength test specimen, is a body of rock with its naturally occurring network of flaws and discontinuities, cracks, joints, and planes of weakness. These are important for a proper understanding of the real ability of a rock mass volume to withstand load, of how and why it fails, and of the resulting fragment size and shape. Recurring discontinuities are easily identifiable in the form of bedding planes, foliation partings, cracks, fissures, or joints. Intersecting groups are common such as parallel planes, or random, irregular structures. A description of the three-dimensional network of intersecting planes, has to include the compass bearing (called the strike) of its intersection with a reference plane (normally the horizontal), and the slope angle (dip) between the plane and the horizontal. For regular or recurring cracks there are two further descriptors, namely the average crack length and the average distance between parallel cracks.

The strength of joints is normally considerably less than that of the adjacent rock. It is described by two simple measures, the tensile or adhesive strength (often zero) at right angles to the plane, and the shear strength or friction angle along the plane. For a detailed understanding of the rock mass behavior under stress and vibration, we also need a measure of the elastic or
plastic deformability of the joint (its "spring constant") and the way the shear strength or friction angle varies with shear deformation and crack separation.

![Diagram showing classification of rock masses]

**Figure 1-6: Classification of the rock masses.**

**ADDITIONAL RESOURCES II**
The above information is additionally supplemented by:

- Lecture module 1.2. (mandatory)

### 1.3 Rock Breaking Processes

This section is concerned with the basic principles of breaking brittle hard rock. The main considerations in breaking rock are the forces required to induce fractures in the rock and the energy consumed in breaking rock. Force is important because it determines the limitation on the type of machinery that can be used to break the rock and on the materials of construction that can be used in the machinery. As the breaking mechanism of the machine changes, so would the energy required to break the rock since the strength of rock varies depending on the type of stress induced on the material.

Energy is important because it determines the rate at which rock breaking can be carried out. All machines are limited in the power that can be applied to the rock and hardness of the manufactured components of the machine. Therefore a process that demands substantial energy will result in a slow rock breaking rate.

The rock breaking process is classified into three major groups: primary, secondary, and tertiary. Each process is described below.

---

5 From Dr. Paul Lever’s 415 course notes.
1.3.1 Primary

This is the application of a force by means of a hard indenter to a free rock face much larger than the indenter. This generates chips which are of a size similar to that of the indenter at the sides of the indenter and a pulverized zone immediately below the indenter.

![Diagram of primary breakage process]

**Figure 1-7: Primary**

Primary breakage processes would include the following:

1. **Impact or hammering.** Dynamic forces are applied
2. **Percussive drilling.** Application of a hard indenter to the bottom of a hole. The force is applied from one side only and the bottom of the hole is the free face. The force applied dynamically and after each application the hard indenter is moved slightly to break out more chips on the next application
3. **Button type cutters for raise and tunnel borers.** The buttons are loaded slowly (quasi-statically) and are moved away to be re-applied elsewhere, that is, indexing occurs by rolling to the next button. Repeated applications over a large surface area maintain the flat face
4. **Disc type cutters for raise and tunnel borers.** Hard indenter indexed by rolling. Forces at a point in the rock rise very slowly.
5. **Drag –bit.** A hard indenter forced onto the rock and indexed by dragging across the surface.
6. **Diamond bits.** A very hard surface and very small indenter dragged across the surface. The real breaking is done by the force thrusting the diamonds against the rock. Diamonds produce very small fragments because they are small indenters.

1.3.1.1 Theory

The theory of the forces and energy in the primary breakage process relates to the confinement of the rock and the energy of the indenter. In the primary breakage process, the rock surrounding the area immediately under the indenter provides confinement for the rock so that stresses are very much greater than the uniaxial compressive strength (confined stresses for rock is higher than uniaxial). Figure 1-8 shows how the size of the indenter is directly related to the size of the indenter.
The stress to penetrate the rock depends on the size of the indenter (note that the nomenclature of this process is such that \( a \) is the width or diameter of the implement striking the rock):

\[
\sigma_p = \frac{\text{constant}}{\sqrt{a}}
\]

The force to penetrate is therefore equal to:

\[
F_p = \text{stress} \times \text{area of indenter}
\]

\[
F_p = \frac{\text{constant}}{\sqrt{a}} \times a^2
\]

\[
F_p = k_p a^{1.5}
\]

The energy for primary breaking is directly proportional to the stress multiplied by the strain. However, considering an elastic behavior, stress is directly proportional to the strain (Young’s modulus). Therefore the primary breaking energy is directly proportional to the square of the stress.

\[
E_p \propto \text{stress} \times \text{strain}
\]

\[
E_p \propto \sigma \varepsilon
\]

\[
E_p \propto \sigma^2
\]

so finally, \( E_p \propto \left(\frac{\text{constant}}{\sqrt{a}}\right)^2 \) therefore \( E_p = \frac{K_p}{a} \)

Note that these formulae are only valid when tools are driven to form the first chip. As penetration increases, forces and therefore required energy also increase. This may be due to the increasing confinement of the rock. Practically speaking, this effect would essentially cause the button/indenter to be buried. Indexing (moving through rotation of the bit or roller) the indenter to a new free surface would allow the indenter to penetrate with a force or stress closer to 1st chip formation.
Side note on indenter shape: From the above, it can be seen that indenter shape directly impacts the energy required to penetrate the rock. Some drill bit manufacturers sell drill button bits with ballistic or circular bits. This can be seen in Figure 1-10 which compares a ballistic shaped bit with a circular or dulled bit.

Figure 1-10: Penetration depth of various button shapes

1.3.1.2 Example 1
Determine the boring rate of a 75 kW raise boring machine using roller cutters with buttons of 1 cm diameter and boring head of 2 m diameter. Assume correct thrusting, that is, a chip forms with every pass. With buttons of 1 cm the mean chip size will also be about 1 cm.

\[ a = 10^{-2} \text{ m} \]

Fracture energy \[ E_p = \frac{K_p}{a} = \frac{1.5 \times 10^6}{10^{-2}} = 150 \text{ MJ/m}^3 \]
Power = 75 kW = 75 × 10³ J/s

Breaking rate = \frac{\text{Power}}{\text{energy/volume}} = \frac{\text{Power}}{\text{fracture energy}} \text{ m}³/\text{s}

= \frac{75 \times 10^3 \text{ J/s}}{150 \text{ MJ/m}³} = \frac{1}{2000} \text{ m}³/\text{s} \times \frac{3600 \text{ seconds}}{\text{hour}} = 1.8 \text{ m}³/\text{hr}

Boring rate = \frac{\text{breaking rate}}{\text{hole area}} = \frac{1.8 \text{ m}³/\text{hr}}{\pi r^2} = \frac{1.8 \text{ m}³/\text{hr}}{\pi (1 \text{ m})^2} = 0.57 \text{ m/hour}

1.3.1.3 Example 2

Determine the boring rate of a 75 kW raise boring machine using disc cutters on a 2m diameter head. Assume correct thrusting. The shape of the indenter is not very well defined with the disc cutter. However, a few centimeters of the disc are in contact with the rock as the disc rolls and chips are long and narrow with a width of about 4 cm. Take the width of the chip as the value for a.

\[
a = 4 \times 10^{-2} \text{ m}
\]

Fracture energy \[E_p = \frac{K_p}{a} = \frac{1.5 \times 10^6}{4 \times 10^{-2}} = 37.5 \text{ MJ/m}³\]

Breaking rate = \frac{\text{Power}}{\text{energy/volume}} = \frac{\text{Power}}{\text{fracture energy}} \text{ m}³/\text{s}

= \frac{75 \times 10^3 \text{ J/s}}{37.5 \text{ MJ/m}³} = \frac{2}{1000} \text{ m}³/\text{s} \times \frac{3600 \text{ seconds}}{\text{hour}} = 7.2 \text{ m}³/\text{hr}

Boring rate = \frac{\text{breaking rate}}{\text{hole area}} = \frac{7.2 \text{ m}³/\text{hr}}{\pi r^2} = \frac{7.2 \text{ m}³/\text{hr}}{\pi (1 \text{ m})^2} = 2.3 \text{ m/hour}

Note that from the previous 2 examples, the energy for boring with disc cutters is four times smaller and the boring rate is four times greater than with the button cutters. This is because the fragment size with the disc cutters is four times larger.

1.3.2 Secondary

This is the application of forces inside a hole near to the rock face. The forces inside the hole generate tension at the sides of the hole to which produces cracks which ultimately run to the free surface.
Free face

![Diagram of rock with cracks and wedge](image)

**Figure 1-11: Secondary breakage process**

Secondary breakage processes includes:
1. **Wedging.** Wedge driven into a hole which produces crack
2. **Blasting.** Explosive generates a pulverized zone through compression but the real breaking process is by driving tensile cracks.

Specific theoretical aspects of energy required for secondary breakage will be discussed in further blasting aspects of the course.

### 1.3.3 Tertiary

This is the application of forces from more than one side to a free surface.

![Diagram of forces on a sphere](image)

**Figure 1-12: Tertiary Breakage Process**

Tertiary breakage processes include:
1. **Breaking boulders by impact or mud blasting**
2. **Crushing**
3. **Milling**

According to theory, the tertiary breakage process is closely related to breaking the rock in tension. From Figure 1-13 is can be seen that loading of a sphere by diametrically opposed forces causes a uniform tensile stress across the diametrical plane. This causes the sphere to split in tension, that is, at a stress very much lower than the uniaxial compressive stress.
It has been found that the tertiary stress, $\sigma_t$, is also dependent on the size of the rock, but not as important as the size of the indenter for primary breakage. Larger boulders break at lower stresses, hence $\sigma_t$ can be represented as:

$$\sigma_t \approx \frac{\text{constant}}{a^{0.25}}$$

Therefore the splitting forces could be calculated as:

$$F_t = \text{stress} \times \text{area of sphere} = \sigma_t \times a = \frac{\text{constant}}{a^{0.25}} \times a^2 = k_i a^{1.75}$$

It should also be noted that $F >> F_t$

Finally, the energy for tertiary breaking is derived by:

$$E_t = \text{stress} \times \text{strain} = \text{stress squared} = \sigma_t^2 = \left(\frac{\text{constant}}{a^{0.25}}\right)^2 = \frac{K_i}{\sqrt{a}}$$

### 1.3.3.1 Example

Milling reduces rock to 70% minus 75 μm. Milling typically consumes 25 kWh/t. Compare this value with that predicted by the simple formula for tertiary rock breaking processes.

1 kWh = $1000 \times 3600 \frac{\text{min}}{\text{hr}} = 3.6 \text{ MJ}$

Density of rock = 2.7 t/m$^3$

$E_{milling} = 25 \text{ kWh/t} \times \frac{3.6 \text{ MJ}}{\text{kWh}} \times \frac{2.7 t}{m^3} = 243 \text{ MJ/m}^3$

According to the formula, tertiary breaking energy requires:

$$E_t = \frac{K_i}{\sqrt{a}} = \frac{1.5 \times 10^6}{\sqrt{a}}$$
If 70% of the rock is reduced to minus 75 $\mu$m, then the mean fragment size is about 50 $\mu$m.

$$a \approx 50 \times 10^{-6} \text{m}$$

$$E_t = \frac{1.5 \times 10^6}{\sqrt{50 \times 10^{-6}}} = 214 \text{ MJ/m}^3$$

The simplified $E_t$ formula gives a reasonably good estimate of energy consumed in milling.

### 1.3.3.2 Example

A jaw crusher is driven by a 10 kW motor and is set to produce fragments of 1 cm. Determine its ‘crushing’ capacity in tons per hour.

$$a = 1 \times 10^{-2} \text{m}$$

$$E_t = \frac{1.5 \times 10^6}{\sqrt{10^{-2}}} = 15 \text{ MJ/m}^3$$

$$\text{crushing rate} = \frac{\text{Power}}{E_t} = \frac{10,000}{15 \times 10^{-6}} \times \frac{3600 \text{s}}{\text{hr}} = 2.4 \text{ m}^3 / \text{hr}$$

$$\text{density of rock} = 2.7 \text{ t/m}^3$$

$$\text{crushing rate} = 2.4 \times 2.7 = 6.5 \text{ t/hour}$$

### 1.3.4 Miscellaneous

Several other breakage processes exist, these include:

1. **Thermal spalling.** This depends on intense heat (flame) being applied to the rock and traversed so that a high temperature gradient is produce in the rock resulting in differential expansion which produces mechanical strains and ultimately breaking of the rock. It is used in taconite and certain quarrying operations, usually in cold climates. Thermal spalling is also used for finishes on rock surfaces and where high forces must be avoided during breakage. (commonly used ancient technique)

2. **Water jets.** The water jets create high stagnation pressures against the surface it impinges on. Used for drilling in porous hard rock where water goes into pores and breaks grains out. Water jets are known to be wasteful on energy and are used only for special applications.
Table 1-8: Summarizing Theory of Forces and Energy in Breaking Processes

<table>
<thead>
<tr>
<th>Breakage process</th>
<th>Primary</th>
<th>Tertiary</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fracture Force (in Newtons)</td>
<td>$F_p = k_p a^{1.5}$</td>
<td>$k_t a^{1.75}$</td>
</tr>
<tr>
<td>Fracture Energy (J/m$^3$)</td>
<td>$E_p = \frac{K_p}{a}$</td>
<td>$E_t = \frac{K_t}{\sqrt{a}}$</td>
</tr>
<tr>
<td>Typical value $k$</td>
<td>$k_p = 10^8$</td>
<td>$k_t = 10^7$</td>
</tr>
<tr>
<td>Typical value $K$</td>
<td>$K_p = 1.5 \times 10^6$</td>
<td>$K_t = 1.5 \times 10^6$</td>
</tr>
</tbody>
</table>

Only for $a$ in meters and valid for $a << 1.0$

ADDITIONAL RESOURCES III
The above information is additionally supplemented by:
- Readings – Chapter 9.1 – Mining Engineering Handbook. (Mandatory, this covers modules 1 and 2)
- Lecture module 1.3. (mandatory)
- Assignment 1
Module 2: Drilling and Blasting Components

The following notes provide a summary of the tools and theory involved in drilling and blasting. This covers basically drills and explosives. This module is organized into four main sub-modules:

1. Drilling mechanisms
2. Drilling equipment
3. Explosives Introduction
4. Explosives Products

2.1 Drilling Mechanisms

Drilling is used in several industries and purposes. These various sectors are briefly listed:

- Mining: drilling and blasting of rock

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6 These notes were assembled directly from the following references:

- From Dr. Paul Lever’s 415 course notes Hartman, Howard L. Ed. SME Mining Engineering Handbook. 2nd ed. 1992
- Stefanko, Robert. Coal Mining Technology Theory and Practice. Littleton CO.: Society of Mining Engineers. 1983
- Mining-Technology.com, search: continuous mining
- Caterpillar Performance Handbook, 28th Ed.
- 2001 Class notes, from Bob Cummings
- The History of Explosives. http://sis.bris.ac.uk/~dj9006/explosives/history.html
• Exploration: diamond core drilling, percussive, churn, hammer, rotary (collect chips as samples)
• Quarrying: including specialized, small diameter (to create slabs)
• Construction: Intermediate drill sizes (typically to reduce excess damage, to maintain slope)

In focusing on mining applications, several features can be remarked:
• May need different drills for ore and waste, due to the variances in hardness and for grade control
• May need different drills depending on the location of drilling (air pressures)
• Underground typically uses smaller holes
• Surface mines typically use very large diameter holes
• Drill selection can depend on bench height (surface) or direction of drilling (underground)

The four main components in drilling include:
• Energy source (drill and compressed air) – where the drill converts energy (such as the potential energy in compressed air) into mechanical energy.
• Energy transmitter – (steel) – the steel is the primary mover of the bit
• Energy applicator – (bit) – the bit attacks the rock mechanically
• Fluid – cleans hole, lubricates bit & hole wall, controls dust, cools bit, stabilizes hole. The fluid can be water, air, or mud.

There are several losses of energy once the drill has converted the original energy into mechanical energy:
• Compression of steel
• Bending
• Elastic strain @ couplings
• Internal friction in the drill
• Heat
• Side friction of cuttings
The two primary rock drilling functions are:
- Penetration – where the fracturing or breaking of the material from in-situ
- Circulation – where the debris is ejected.

Figure 2-1 shows the drilling process. Three components of the drilling process should exist at all times. If any one of these components is ineffective, the drilling process will deteriorate rapidly and the information obtained from the hole will be invalid.

![Figure 2-1: The drilling process](image)

2.1.1 Percussive Drilling

Percussive drill bits break rock by indentation. The peak stress applied to the bit by the drilling machine causes the bit to strike the rock at a stress calculated by:

\[ \sigma_p = \rho cv \]

Where:
- \( \nu \) - bit velocity at the rock face
- \( \rho \) - density of the impacting body, steel = 7850 kg/m3
- \( c \) - compression wave velocity of the impacting body (~5000 m/s)

Hence, \( \sigma_p = 39.25 \nu \) MPa for units of m/s

Figure 2-2 shows the mechanical components and the nomenclature of a typical percussion drill. Either compressed air or water pushes the piston in the drill to strike the steel in a repetitive manner. The energy to break the rock is imparted through the drill bit by the compressive stress wave imparted by the piston to the steel.
Issues of drill efficiency and energy losses associated to coupling, the mechanics of energy along the drill steel are presented by the following discussion. The process of imparting energy into the rock begins with the piston being forced at high speed into the drill steel as seen diagrammatically in Figure 2-3.

When the piston strikes the steel the kinetic energy is transformed into a compressive wave imparted to both the steel and piston as seen in Figure 2-4.

These waves travel at a velocity \( c \), to the ends of the piston and drill steel where they are reflected. The nature of the reflected wave depends on the boundary condition at the ends. For a bar with a free end (piston), the wave is reflected as a tensile wave, whereas at a fixed end it is reflected as a compressive wave. The piston remains in contact with the steel until the tensile wave returns to the steel-piston interface and causes the two to separate. This ends the pulse upon which the hammer imparts energy onto the steel. Calculating the duration time of the pulse has been used to calculate the yield strength for the drill before the machine would break. The reflection and separation effect is shown diagrammatically in Figure 2-5. Since the drill steel is in contact with the rock (not directly fixed), some of the energy is imparted into the rock to break out chips and the remaining energy returns as a compressive or tensile wave. The type of wave depends on the rock type, and contact between the drill steel and the rock.
Consider the case where the piston has a length $l_1$, and the drill steel has a length $l_2$, and where the diameters of the piston and drill steel are the same. The length of the compressive wave set up in the steel would therefore be equal to $2l_1$ divided by the speed of the piston.

$$t_s = \frac{2l_1}{c}$$

Some have observed that the yield strength of steel used for the piston and drill steel limits the stresses that can be applied to the rock by using the piston-type devices. The maximum stress in the piston $\sigma_h$ and the drill steel $\sigma_t$ are given by:

$$\sigma_h = \rho cv \frac{A_t}{(A_t + A_h)}$$
$$\sigma_t = \rho cv \frac{A_h}{(A_t + A_h)}$$

Where:
- $v$ - bit velocity at the rock face
- $\rho$ - density of the impacting body, steel = 7850 kg/m$^3$
- $c$ - compression wave velocity of the impacting body $\sim$5000 m/s
- $A_t$ is the cross-sectional area of the drill steel
- $A_h$ is the cross-sectional area of the piston

Therefore, where $A_t$ and $A_h$ are equal, $\sigma_h = \sigma_t = \rho cv/2 = 19.63v$ MPa. If it is generally desirable to limit the stresses in these steel components to about 212 MPa, the maximum piston velocity is equal to about 10.8 m/s. However, these velocity limits are only for hydraulic drills as pneumatic drills cannot generate such high velocities.

Not all the energy in the fluid in drilling is expended in breaking the rock. Sources of energy loss include:
- Friction at coupling and other contact points,
- Bit wear
- Noise

Figure 2-5: Reflections and wave transmission.
• Vibration.
• Flushing (positive work)
• Rotation of steel (positive work)

Machine mounted drills are known as drifters but most commonly as jumbo drill rigs. Most of the drills in service are pneumatic, meaning compressed air provides the energy to move the piston. Hydraulic, using water as the energy transmission method, drills are also used but are technically more sophisticated and are typically found on jumbo drill rigs. The advantages and disadvantages of the hydraulic and pneumatic drills are listed below.

**Table 2-1: Comparison of Hydraulic and Pneumatic drills**

<table>
<thead>
<tr>
<th></th>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Pneumatic</strong></td>
<td>• Low cost</td>
<td>• Very low efficiency (less than 5% of the input power to the compressor is delivered to the rock)</td>
</tr>
<tr>
<td></td>
<td>• Traditional, well established technology</td>
<td>• Poor environment for the machine operator (machine is noisy and lubrication causes a fog)</td>
</tr>
<tr>
<td></td>
<td>• Simple mechanical components</td>
<td>• Components more complex</td>
</tr>
<tr>
<td></td>
<td>• Most mines have compressed air lines through the drifts</td>
<td>• More capital outlay</td>
</tr>
<tr>
<td><strong>Hydraulic</strong></td>
<td>• More efficient (25-30% of energy input power delivered to rock)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>• 8-10 decibels less than pneumatic drills (quieter)</td>
<td></td>
</tr>
</tbody>
</table>

**Table 2-2: Comparative table**

<table>
<thead>
<tr>
<th>Drill type</th>
<th>Typical Hole Sizes</th>
<th>Typical Hole Lengths</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Inches  mm</td>
<td>Ft m</td>
</tr>
<tr>
<td>Jacklegs / stopers</td>
<td>0.75 – 1.25 19 - 32</td>
<td>1 – 4 0.3 – 1.2</td>
</tr>
<tr>
<td>Drifter / Jumbo</td>
<td>1.50 – 2.25 38 – 57</td>
<td>4 – 100 1.2 – 30.5</td>
</tr>
</tbody>
</table>

**ADDITIONAL RESOURCES IV**
The above information is additionally supplemented by:
• Lecture module 3.1 (mandatory)
2.1.2 Rotary Drilling

In rotary drilling, the bit attacks the rock with energy supplied to it by a rotating drill stem. The drill stem is rotated while a thrust is applied to it by a pulldown mechanism using up to 65% of the weight of the machine. The bit breaks and removes the rock by either a ploughing-scraping action in soft rock, or a crushing and chipping action in hard rock, or a combination of the two. Compressed air or water is supplied to the bit via the drill stem. The air cools the bit and provides a flushing medium for the cuttings. The drills typically operate in the vertical position although many types can drill up to 25 to 30 degrees horizontal. Figure 2-6 provides the nomenclature and dynamic movement for rotary surface blasthole drills. Figure 2-7 provides the deck plan for a P&H 100 rotary drill.

![Dynamics of surface blasthole drill](image)

**Figure 2-6: Dynamics of surface blasthole drill**
Rotary drilling is one of the most popular drilling techniques for large surface mines where large diameter holes are used in blasting. Figure 2-8 shows how rotary tri-cone bits are the ideal application in hardrock surface mines.

**Figure 2-7: Floor plan for rotary drill P&H 100XP**

**Figure 2-8: Selection of Drilling method**
2.1.2.1 Rotary drill string components

The drill string consists of three main systems:

- The suspension and drive that connect the string to the rig and power system
- The drill pipe that transmits power, fluid, and cuttings
- The drill tools and bit that advance and shape the hole and provide samples.

Figure 2-9: Components in rotary drilling

2.1.2.1.1 Stabilizers

The efficiency of rotary cutting is lost when the drill pipe is allowed to bounce around or when it fails to run smoothly in the center of the hole. The bit is steadied and hole maintained on grade by the use of stabilizers. There are two primary types of stabilizers, blade and roller (Figure 2-10).

Blade stabilizers are designed to center the drill collars in the hole. They are run between or above the drill collars. Blade stabilizers come in many forms, some have spiraled blade arrangements, other run blades vertically along the stabilizer length.

Roller stabilizers have throwaway or replaceable rollers. They have the advantage over blade stabilizers by providing better stabilization, longer life, and lower torque requirements than blade types.
The advantages of using stabilizers should be considered when designing or selecting a drilling process. The primary reason is costs. These costs include reduced cost per ton of rock removed, the more efficient use of expendable items such as rock bits and drill steel can be achieved. Rock Bits are designed to rotate about their own center. Stabilization assures that the bit will do this and thus cause the energies and forces exerted on it to be most efficiently utilized in an axial direction. Lateral movement or stumbling is restricted and hole is produced in the direction intended. More footage per bit and an increased penetration rate is achieved by proper utilization of the forces applied to the bit. Dull conditions of bits give evidence of this effective stabilization. Gage wear is uniform and less severe. Shirt-tail wear problems are reduced. The inner row cutting structure is more uniformly worn and inner and outer flank wear is reduced.

Stabilizers also affect drill stem life. Without stabilization, rough spiral bores, ledges and other unconformities are obtained. The possibilities of crooked hole are enhanced. Drill steel rotating in these rough and crooked bores scrubs and scrapes against the bore wall and thereby abrades.

Stabilizers also affect drill availability. The smooth bore produced with adequate stabilization permits faster rock bit and drill steel retraction from the bore hole. Furthermore, the smooth bore sloughs less than a rough bore. This means that less rock particles fall to the bottom of the hole. Re-drill time required is normally eliminated or drastically reduced.
In the selection of a stabilizer, one must take into account the adequate stabilization. Unless the guiding elements of a stabilizer are very near the bore wall, adequate stabilization of the bit is not achieved. Theoretically, the guiding elements should have the same diameter as the bit. Unfortunately this is not practical because of the normal attrition of rock bit gage wear surfaces. The stabilizer should therefore be held at the largest diameter practicable. Concentricity (the quality of having the same center - as circles inside one another) of guiding elements with the axis of the bit and steel is also quite important to proper stabilization. Eccentricities (different centers) of these elements tend to void any hope of reducing drilling costs with a stabilizer.

The stabilizer which maintains guiding elements close to hole wall is most efficient. The conventional welded rib or cast rib stabilizer (also known as blade) does an adequate job when new and unworn, provided the ribs are concentric. The problems with this type of stabilizer are wear rate and maintenance cost. Rib-type stabilizers, due to their construction, necessarily drag and scrape against the bore wall and abrade rapidly. Due to the cost in maintaining close-to-hole wall contact these stabilizers are left in the drill string long after adequate stabilization has been lost. The most efficient stabilizer is one with true-rolling centralizers that are in rolling contact with the bore wall. Scraping and gouging of the guiding elements are eliminated and co-centricities are maintained. Roller-stabilizers provide adequate stabilization without imposing additional torque.

2.1.2.1.2 Substitutes (Subs)
Substitutes (known as subs) are used as adapters where threads of one size or type must be coupled with threads of another size or type. They are also used at points of heavy wear to provide a readily replaceable thread.

Shock absorber subs are run above hammers or above the bit or rod in rough drilling situations, thus reducing drill string vibration and increasing bit life. A swivel mount shock sub is also used to dampen vibration effects on the rig structure thereby reducing maintenance on the rig and mechanisms for pulldown and rotation.

![Figure 2-11: Swivel mount shock sub](image)
Shock and vibratory energy transferred between drill and bit are absorbed through a series of rubber bonded segments. The absorbed energy is released to the atmosphere in the form of heat.

A. The pulldown or drilling weight to the bit is transferred through the segment rubber placed in shear.

B. The rotational torque (or torsional loading) to the bit is transferred through the segment rubber placed in compressive loading.

C. The accelerated changes in axial and torsional loadings are minimized by the segment rubber.

Figure 2-12: Vibration reducing effects of using shock subs

The shock absorber is most beneficial when drilling in fractured formations, intermittent hard and soft layers or hard formations. The benefits include:

- Reduces drilling machine maintenance by dampening torsional and axial shock loads.
- Increases drilling rates by keeping bit in more uniform contact with formation. Allows use of more weight and higher rotary speeds in rough drilling areas.
- Increases bit life by dampening cyclical shock loading normally transmitted to the bit bearings and cutting structure.
- Decreases operator noise level by eliminating the metal to metal contact between rotary drive and drill pipe.

Normally, no modification of the drill is required and installation can be completed in a short time. On multi-pass drills where length is not critical, a saver sub is recommended for use between rotary shouldered connection and drill pipe.

2.1.2.1.3 Rotary Drill Stems

Factors involved in selecting drill stems (aside from bailing requirements) includes:

- Fabricated (welded) or integral (machined from single steel bar) drill steel;
- Thread size and type
- Wall thickness
- Types of connections
2.1.2.2 Rock drillability / Penetration rate

Rock drillability is defined as the penetration rate of a drill bit into the rock. It is a function of several rock properties such as:
- Mineral composition;
- Texture;
- Grain size;
- Degree of weathering.

Figure 2-13: Blasthole components with respect to drill rig
As is obviously important to costs and productivity, penetration rate is one of the most important factors in drilling. Some empirical equations have been developed from extensive rotary tests in iron ore. The Bauer and Calder method states that penetration rate, \( P \), can be calculated using the following equation:

\[
P = (61 - 28\log_{10} S_c) \frac{W}{\phi} \cdot \frac{rpm}{300}
\]

where
- \( p \) = penetration rate in ft/hr
- \( S_c \) = uni-axial compressive strength, in thousands of psi
- \( W/\phi \) = weight per inch of bit diameter, in thousands of pounds
- \( Rpm \) = revolutions of drill per minute

The adjustable factors in the above equations are the variables controlled by the operators. Rock compressive strength is not but can be estimated using the graph in Figure 2-14.

![Figure 2-14: Penetration rate vs. Rock strength](image)

Several other indices exist for rock drillability which include:
- Drilling Rate Index;
- Classification of rock types based on the drillability of Barre granite
- Mohr's test
- Indices (sensor-measured), e.g. AQUILA's drill performance monitoring
2.1.2.3 Rotary Drill Pulldown Weight

A portion of the machine weight is applied by the pulldown motor via the pulldown chain or chains, rotary head and drill stems to the drill bit. Figure 2-15 illustrates recommended bit loadings for different bit sizes. As the bit diameter increases, the bearing size increases thus allowing an increase in the tolerable load. Overloading the bit results in severe loss of bit life as illustrated in Figure 2-16.

![Figure 2-15: Recommended pulldown weights per inch of bit diameter.](image)

![Figure 2-16: Bit life vs. pulldown weight for 9 1/4 inch diameter rotary bits in hard formation](image)

There are several method of rotary drive. The bit may be turned by:

1. rotating a rotary table which turns the pipe as it slides through.
2. rotating the pipe directly by a drive unit which moves down with the pipe.
3. rotating the bit, using a down hole turbine drive.

Figure 2-17: Rotary drive types

Figure 2-18: Pulldown mechanisms

Figure 2-19: Rack and pinion pulldown
General parameters for drill penetration can be seen in Table 2-3.

**Table 2-3: Average Drill Bit Footages (from large Iron Ore mine in Canada)**

<table>
<thead>
<tr>
<th>Drill Bit Diameter</th>
<th>Average Feet per Bit</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>ore</td>
</tr>
<tr>
<td>(in.)</td>
<td>(ft)</td>
</tr>
<tr>
<td>$9\frac{3}{4}$</td>
<td>250</td>
</tr>
<tr>
<td>12</td>
<td>311</td>
</tr>
<tr>
<td>15</td>
<td>381</td>
</tr>
<tr>
<td>17</td>
<td>445</td>
</tr>
</tbody>
</table>

2.1.2.4 **Flushing medium**

Air is used to bail the drill cuttings from the hole as well as cool the bit bearings and, when used, roller stabilizer bearings. Approximately 20% of the air is forced through the roller cones for cooling purposes by adjusting the air pressure across the bit using the bit nozzles. The air volume is the primary requirement for bailing cuttings from the hole. Air velocity up the hole is dependent on the air volume per minute as well as the hole annulus (ring-shape where hole and stem meet). The velocity of the drill cuttings in this air is dependent on the chip size, density, and shape. Experimentally, the balancing air velocity in feet per minute is given by:

$$U_m = 264p^{\frac{1}{2}}d^{\frac{1}{2}}$$

Where: 
- $d = \text{diameter of the chip in inches}$
- $p = \text{density of chip in Ib/ft}^3$

At air velocities above this balancing value, the chips begin to move, their velocity being approximately one half the excess air velocity above the balancing value. A bailing velocity of 1800 mpm/s (6000 fpm) is usually adequate to bail 13 mm (1/2 in.) chips. Figure 2-20 illustrates a typical air requirements chart.
Factors involved with choosing the air velocity are that higher velocities:
1. give higher bailing velocities;
2. will bail larger chips;
3. tend to give higher bit life;
4. will help cater for hole cavities, etc.;
5. will help cater for drill stem wear;
6. may give higher penetration rates and possibly lower cost per ft; and
7. reduce the volume of cuttings in the hole for a given penetration rate;

The drawbacks of increased bailing velocities include:
1. will give increased stabilizer and pipe wear;
2. increased dust deflector and deck bushing wear
3. may damage borehole walls in soft drilling

**Figure 2-20: Air Bailing Chart**
2.1.2.5 Operational tips
The following are some operational tips on bit and rig for top performance when using air circulation bits.

1. Use straight drill steel with properly maintained threaded connections.
2. Use a good grade of thread grease and maintain connections properly.
3. Use care attaching and removing a bit from the drill pipe.
4. Open air valves before starting to drill with a bit and keep the air on until the bit is out of the hole.
5. Break in a new bit by drilling at reduced down pressure and rotational speed for a short period.
6. To collar or start new hole, reduce down pressure and rotation.
7. Re-establish bottom hole pattern with reduced down pressure and rotation when drilling is interrupted.
8. Never finish an old hole with a new bit. This can pinch the cones, damaging the bearings and gauge teeth.
9. Always maintain drilling air pressure at appropriate levels.
10. Rotary speed should be decreased as down pressure increases.
11. Do not use more water than is necessary to control dust and maintain hole wall.
12. Maintain rotation while tripping into or out of a hole.
13. In wet holes, maintain as high an air pressure and volume as is possible.
14. Guard against dropping the bit and drill steel to the bottom of the hole at high speed. High speed impact is a common cause of damage.
15. Periodically inspect the bit for damage or impending failure. Non-uniform cone temperatures may indicate obstructed air courses and potential bearing failure.
16. Before an idle period, clean the bit by passing air or water through it while rotating the cutters by hand. After an idle period and before re-using a bit, make sure all cones turn freely by hand.

2.1.2.6 Safety Tips
The following safety tips should be taken into account around large surface blasthole drills:

2.1.2.6.1 Moving the Drill
As with all heavy equipment, care must be taken when blasthole drills are moved around the job site. Watch for people, overhead electrical power lines, other equipment, low or narrow clearances, ground-bearing limits, and steep hills or uneven terrain. Use a signal man when traveling and use an audible warning signal to alert people in the area before traveling. Know the height, width and weight of the machine. Always secure on-board equipment before traveling. Do not travel with people riding on the outside of the machine or inside the machinery house.

2.1.2.6.2 High Voltage dangers
Hazardous voltage from the machine contacting power lines can cause electrocution and burns to anyone near the blasthole drill. MSHA and OSHA regulations require at least ten (10) feet of clearance from overhead lines carrying 50,000 volts or less. Greater clearances are required for lines with higher voltages. Some local regulations require greater distances than OSHA or MSHA.

2.1.2.6.3 **Trailing Cables**

Blasthole drills with electric tail cables can break the cable connection or crush the cable when traveling. Maintain cable slack while operating and traveling. Use signalmen during traveling to prevent damage to the cable or cable connection.

Rotational Dangers
During drill operation, keep all people off the drilling platform and drill mast, and away from drill stems. Moving components or rotating drill stems can entangle clothing and can crush, pinch or strangle personnel.

2.1.2.6.4 **Tipping Dangers**

Exceeding the slope or grade limitations given for your specific machine and machine configuration can cause machine tip-over. Prior to propelling, always determine the slope limitations for your specific machine and never travel or operate on site slope conditions which exceed these limitations. When propelling with the mast elevated, always position the machine with the mast on the uphill side of the machine. When the mast is down, propel the machine with the operator's cab on the downhill side of the machine. Sudden tip-over can occur when raising, lowering, or leveling the machine with jacks. Always level the machine at the lowest practical height that will unload the crawler belts. Inspect the ground for lifting support and add cribbing or support mats before lowering jacks. When lowering the machine to the ground, always lower the machine slowly and in stages, maintaining a level condition until the crawler belts contact the ground and are supporting the machine. Use a signal person to assist in watching the jacks, crawlers and machine during the process.

2.1.2.6.5 **Crushing Danger from Lowering Mast**

Before lowering the mast, notify personnel to evacuate the roof and drill deck and inspect mast storage area for obstructions. After storing the mast, check for secure mast attachment on the front jack caps.

2.1.2.6.6 **Chips and Dust**

Drilling produces flying debris and dust which can cause serious respiratory disease. Always lower dust curtains before drilling and inspect the curtains to
make sure all dust curtains are installed and in the lowered position. Curtains should not be lifted to remove cuttings while the drill is drilling in the hole. Keep personnel off drilling platform and away from the drill hole while drilling is in process. Avoid dust contamination from work cloths, eating or drinking. Follow mine procedures for air monitoring, exposure limitations, and protection methods for crystalline silica exposure.

Specific design/plan-related safety issues to consider
- Where and when the blasting is scheduled for the day
- The other equipment, powerlines, or structures which must be moved or avoided during movement of the drill.
- The ground surface strength in terms of having adequate weight bearing ability for the drill mass
- The grade of the slope which the drill will be working on (too sloped will cause a tip-over)
- Steps necessary to keep unnecessary people and equipment at a safe distance from drill area.

2.1.2.7 Chip Sampling
Penetrating always involves" chip making". Usually penetration efficiency is improved when large chips are produced and cleared quickly. The best chips are large chips; large chips often make better samples. Large chips need:
- bits with sharp cutting edges.
- large amounts of energy per chip.
- rapid clearing and transport up the hole to prevent regrinding of the chip.

Although larger amounts of energy are required for each large chip, fewer chips are produced and the total energy used per metre of hole drilled is less (less surface area produced.

Many rigs produce good chips at the bottom of the hole but fail to clear them or lift them out of the hole before they are broken up. For best chips, the operator should use the drill as follows:
- Use sharp blade bits or long toothed roller bits with bottom clearing nozzles.
- Use high thrust or feed.
- Keep hydrostatic head to a minimum.
- Use high flushing flow of low viscosity, low solids fluid (must be balanced against erosion of hole wall).
- Use reverse circulation techniques for broken, fractured, cavernous or other formations prone to lost circulation problems.

2.1.2.8 Single pass
Most drills in surface mines use single pass drilling, either by blast design or through purchasing equipment with taller masts. Single pass drilling is
where the entire length of the hole is drilled without having to add more drill stem. The advantage to single pass drilling includes:

- Eliminates adding stems
- Reduces associated thread damage
- Reduces machine downtime for rod changing equipment and tool racks
- Facilitates the cleaning of boreholes
- Permits a continuous air flow through the bit at all times.

Disadvantages of single pass are:

- High masts make the drill more instable
- Extra care is needed in moving drill long distances
- Pulldown chains become long and may require special attention

2.1.2.9 Capacities.

Key operating capacities for a drill rig includes:

- Maximum mast loading capacity
- Maximum hoist or hook load capacity
- Head torque capacity.

2.1.2.10 Productivity Estimate

Figure 2-21 shows a generalized productivity estimate for various materials. Productivity calculations would be site-specific and should take into account:

- set-up time,
- movement time,
- idle periods,
- bit and stem changes
- penetration rate
- Burden and spacing
- bit diameter (alters the penetration rate)
- bit life (changes the frequency of bit and stem changes)
- hole depth (changes the frequency of the movement time)
- number of holes (changes the frequency of set-up time)
Figure 2-21: Tons drilled per operating hour for rotary drills for various hole diameters.

Figure 2-22 shows an estimate of the cost per meter. The drilling costs are made up of two different parts, the drill consumables per unit length of drilled hole and the balance of the drill cost. The balance of the drill cost is converted to the drill cost per unit length of hole using the drill penetration rate as the drill cost would be time dependent. The reason for this split is that the drill consumables cost, predominantly the drill bit cost, is independent from the drill penetration rate, assuming correct operating methods (poor operation can cause increased bit wear life), whereas all other costs are penetration rate dependent. Drilling records or statistics are therefore important to record so that accurate budgeting can be undertaken both in terms of costs that are incurred along with the output from the drill.

Determining the drilling costs must take into account, the following factors: Ownership costs: include amortization and depreciation, interest on borrowed money (if not purchased out of cash-flow, and taxes and insurance. Operating costs: includes power, maintenance, direct and indirect labor, warehousing, and consumables. These consumables are dependant on penetration rate and wear rate and would include items such as stabilizers, drill stems, and drill bits (at times one of the most significant cost).
Drilling technology

Most advanced modern drilling technology focuses around the navigation and positioning of the drill. GPS based navigation systems are available from companies such as AQUILA (Caterpillar owned). The company has a both a drill and shovel product line that uses GPS and PLCs to enhance drilling productivity and provide substantial information from which better mine planning can be undertaken. AQUILA DM systems are available in two ways: as factory installs on new drills, or can be installed as retrofits. There are five main options for the drilling systems:

DM-1: Production Monitoring System
DM-2: Material Recognition System
DM-3: Drill Control System
DM-4: Guidance system for vertical drilling
DM-6: Guidance system for inclined drilling
2.1.2.11.1 DM-1 Production Monitoring System

DM-1 uses AQUILA’s Advanced Monitoring Platform (AMP) with Graphical User Interface (GUI) software to give the operator immediate feedback on drilling productivity and performance. The DM-1 is designed to minimize the amount of operator effort, which means faster work with fewer errors. For example, the start of drilling can be automatically detected, so the operator doesn't have to zero the bit depth. Steel changes are also automatically sensed, eliminating errors in determining hole depth.

Figure 2-24: DM-1 production Monitoring System

2.1.2.11.2 DM-2 Material Recognition System

DM-2 uses a vibration sensor and pattern recognition software to automatically process and analyze drill variables, and determine hole geology while drilling. DM-2 pinpoints the location of ore and waste interfaces, then delivers real-time information via a color LCD screen. The result is immediate, accurate-to-the-centimeter information for both the operator and planning engineers in the mine office.

Figure 2-25: DM-2 Material Recognition System
DM-1 and DM-2 functioning in tandem would enable the following:
- Detailed, real-time geological information on a location-by-location basis.
- Improved explosives usage, better fragmentation and reduced ore dilution.
- Wireless connectivity to and from the mine office.
- Comprehensive production reporting.

2.1.2.11.3 DM-3 Drill Control System
DM-3 delivers consistent drill operation and performance in all kinds of conditions, for operators of all skill levels. By interfacing the DM-1 or DM-2 to Programmable Logic Controller (PLC) modules and electrohydraulic actuators, DM-3 makes automatic hole drilling possible - from collar to design depth. It regulates pulldown pressure and rotary speed within the most productive limits for torque and vibration. This means more consistent performance, with optimized penetration rates for changing downhole conditions. And the ultimate benefits of longer drill bit life and more productivity.

![Figure 2-26: DM-3 Drill Control System](image)

2.1.2.11.4 DM-3 provides:
- Reduced maintenance and consumables costs.
- Optimized operation for overall productivity.
- More consistent hole depths and more even benches.

2.1.2.11.5 DM-5 Guidance System
The DM-5 Guidance System for Vertical Drilling combines high-resolution, Real-Time Kinematic (RTK) GPS receivers and an AMP platform. This allows the operator to place a blasthole within centimeters of his target, without the need for traditional surveying or staking. Once the drill is leveled and the hole started, the DM-5 automatically determines collar elevation, calculates the required drilling depth and displays the information on an easy-to-read LCD screen. And all the information on blasthole positions is stored by the DM-5 and transmitted to the mine office for use in blast design and updating the geological model. This technology allows the operator to navigate to each
blasthole via a navigation screen. The figure below shows the view provided to the operator during the navigation phase of drill positioning and the second picture shows the operator’s view when within 1 meter of the designed hole location (the view automatically zooms-in).

**Figure 2-27: Operator’s view when navigating**

The location where the drill finally punched the hole, along with sensor data (that can be interpreted as geological information) can be recorded and sent back to the engineering office. The positioning information is provided by differential GPS, from two receivers on the head of the mast of the drill.

**Figure 2-28: GPS receivers**

### 2.1.2.11.6 DM-6 Guidance System for Inclined Drilling
DM-6 has the same basic features as the DM-5, but is specifically designed for use on machines that routinely drill inclined blastholes with angles between five and 30 degrees from the vertical. For inclined drilling, the DM-6 uses a servo-controlled Automatic Leveling System (ALS) that automatically maintains the GPS antennas level with the horizon. The DM-6 also uses software designed to facilitate drill setup and improve alignment accuracy when drilling inclined holes. And like the DM-5, this system also stores all blasthole positions, which are transmitted to the mine office for updating the design file.

Using both DM-5 and DM-6 allow:
- Greater hole location and depth accuracy for both inclined and vertical drilling.
- Time and money savings from no-stake marking and better blast results.
- The elimination of pre- and post-blast pickups.

### 2.1.2.11.7 Case Study: Highland Valley Copper Mine

A post-audit was undertaken in 1998 of the implementation of AQUILA technology on three B&E 49Rs at the Highland Valley Copper mines. The benefits achieved after only 1 year of utilization are listed below. It should be noted that further development or utilization of the the data provided by the drill were used in other initiatives.

1. By designing the blast pattern on an office computer (as opposed to on-site by the surveyors), engineering considerations can more readily be taken into account.
2. HVC was able to place up to five surveyors (who formerly would be marking blast patterns) on other revenue generating projects within engineering.
3. The creation of a pattern no longer requires a large amount of survey consumables.
4. Adverse climatic conditions no longer impede the blast design and implementation.
5. There is consistency in blast design (only one technician does all of the designs).
6. Design time has been reduced (no surveying is required).
7. Much less skilled drill operators can operate the drill, since positioning the drill is made easy through the ‘video game’ type navigation interface. This has also tended to increase the productivity of the drill through a reduction in tramming and positioning time.
8. Supervision was increased as the productivity and delays are recorded and delivered in the log file.
9. Any need to re-survey the drilled pattern was eliminated (according to mine regulation, the position of each hole must be known so that bootlegs are avoided.)
10. The most important change incurred by the implementation of the technology was the increase in collaring accuracy (previously, errors of
up to a meter in an X & Y direction were common, the current average error is 10 cm in X & Y). The misalignment of the blastholes induced poor fragmentation. Engineering staff consider that such oversize caused increased wear on shovels, trucks, and crushers. It was also observed that poor fragmentation would adversely affect the mill. Throughput would fluctuate when processing such ore, resulting in decreased recovery.

ADDITIONAL RESOURCES V
The above information is additionally supplemented by:
- Lecture module 3.1 (mandatory)
- Assignment 4: Drill selection

2.2 Drilling equipment
This sub-module will familiarize students with the equipment that, as engineers, they will manage as part of a wider fleet to ensure system efficiency.

ADDITIONAL RESOURCES VI
The primary component of this sub-module is provided by:
- Lecture module 3.2 (mandatory)
- PowerPoint presentations: material developed for 2002’s 415 Rock Excavation course. Available upon request

2.3 Explosives Introduction
The physics and chemistry behind explosives and their detonation are introduced in this first sub-module on explosives. The second sub-module will cover the properties of explosives products.

2.3.1 Detonation Theory
Blasting theory is one of the most controversial topics in the rock excavation industry. No single concept has been developed and accepted that fully explains the mechanisms of rock breakage in every situation, yet a vast amount of research work has contributed valuable information and insight into blasting theories. Some of the theories and findings are discussed in this section.

An explosion is a self-propagating, exothermic reaction. The stable end products are gases that are compressed, under elevated temperature and very high pressures. It is the sudden rise in temperature and pressure from
ambient conditions that results in a shock wave, or a detonation traveling through the un-reacted explosive. The velocity of detonations (VOD) lies in the approximate range of 5000 to 30,000 fps (1500 to 9000 m/s), well above the speed of sound in the explosive material. Deflagration is the chemical burning of explosive ingredients at a rate well below the sonic velocity. It is associated with heat only and carries no shock due to its much slower reaction rate. Deflagration occurs when less than ideal hole-loading conditions or explosive formulation are involved.

The maximum energy release upon detonation occurs when the explosive mix is formulated for oxygen balance. An oxygen-balanced mixture is one in which there is no-excess or deficiency in oxygen, such that the gaseous products formed are chiefly H$_2$O (water vapor), CO$_2$ (carbon dioxide), and N$_2$ (nitrogen). In actual blasting practice, small amounts of noxious gases such as NO (nitric oxide), CO (carbon monoxide), NH$_4$ (ammonia), CH$_4$ (methane), and solid carbon, are formed resulting in nonideal detonations and somewhat less than ideal pressures and energies. Commercial explosive formulation attempts to achieve an oxygen-balanced mixture. The work done by chemical explosives in the fragmentation and displacement of rock depends on the shock energy as well as the energy of the expanding gases.

The self-sustained shock-wave produced by a chemical reaction in a gaseous medium was described by D.L. Chapman and E. Jouquet as a space. This space of negligible thickness is bounded by two infinite planes - on one side of the wave is the un-reacted explosive and on the other, the exploded gases, as seen in Figure 2-29.

![Figure 2-29: Detonation Process of explosive cartridge](image)

There are three distinct zones: (a) the undisturbed medium ahead of the shock wave; (b) a rapid pressure rise at Y leading to a zone in which chemical reaction is generated by the shock, and proceeds until complete at X; and (c) a steady state wave where pressure and temperature are maintained. This condition for stability exists at hypothetical X, which is commonly referred to as the Chapman-Jouquet (C-J) plane. Between the
two planes X and Y there is a conservation of mass, momentum, and energy. A simplified and approximate velocity of detonation (VOD) can be obtained from the following empirical relation:

\[ C_d = \sqrt{J}(1 + 1.3\rho) \]

Where:  
- \( C_d \) is the VOD in m/s  
- \( J \) is heat of reaction in MJ/kg  
- \( \rho \) is the specific gravity

The detonation pressure \( P_d \) in N/m\(^2\) which exists at the C-J plane can be estimated with comparable accuracy using the equation below:

\[ P_d = \frac{\rho C_d^2}{4} \]

Explosion pressure, \( p_a \), is the pressure when the product gases have the same density as the unreacted explosive. The explosion pressure is approximately half the detonation pressure.

### 2.3.1.1 C-J Plane

A deeper exploration of the individual zones in the C-J theory should be undertaken. Note Figure 2-30 as being the idealized detonation wave traveling through a cylindrical explosive shape, producing an increase in pressure. The steady-state chemical reaction takes place behind the shock front within the reaction zone. At the end of this zone, a nonsteady-state region exists. It is created by a flow of expanding gases in a direction opposite to that of the traveling wave front. Once again, the C-J plane is seen as the boundary between the steady and non-steady state, where the reactions are considered complete. This is also the plane where all the thermodynamic properties are calculated. These are:

- \( p \) – pressure
- \( V \) – velocity
- \( T \) – temperature
- \( E \) – internal energy or
- \( Q \) – heat of formation and
- \( \rho \) – density
Figure 2-30: Detonation Process for cylindrical explosive

The maximum pressure and duration of a wave pulse is directly proportional to the shock energy and gas pressure of the explosive, respectively. High explosives such as military explosives or highly sensitive commercial explosives are characterized by an intense shattering effect upon detonation (known as brisance). They liberate gaseous products very quickly. The distance between the shock front and the C-J plane is very short and results in a pressure pulse of high amplitude and short duration. The pressure pulse for less-sensitive commercial explosives shows a decreased pressure amplitude and a longer pulse length. In this case, the reaction is slower and the gas volume is greater, as seen in Figure 2-31.

Figure 2-31: Pressure shape for A-high explosive and B-commercial explosive containing high gas volume
2.3.1.2 Quick note on Shock Wave Propagation

As the stress wave front generated from a blast travels outwards, it has a tendency to compress the material at the wave front through a volume change. There is a tangential or hoop stress at right angles to the compression wave front. If this tangential stress is strong, then radial failure from the explosive source is evident. When the compression wave front travels from one material to another, there could be three possibilities at the interface as seen in Figure 2-32. These possibilities depend on the material’s acoustic impedance $I_r$ which is defined as the product of the material’s density $\rho$ and sonic velocity $V$. As seen in the figure, the material in which the energy wave originates is labeled A, and the material into which the wave travels, B. When the ratio of acoustic impedance of material A to material B is less than 1, some of the energy is transferred to material B as compression waves, whereas the rest is reflected back as compression waves as well. When the acoustic impedance ratio is 1, all the energy is transmitted into material B as compression waves. When the impedance ratio is greater than 1, some of the energy travels through the interface as compression waves and the rest is reflected back as tension waves. When a compression wave traveling through a rock medium encounters an interface such as a free face, nearly all of the energy will be reflected back as a tension wave. If the distance between the free face and the explosive charge is relatively small, most of the energy will be spent in peeling off layers of rock at the free face.

Figure 2-32: Shock wave propagation
2.3.2 Comparative Explosive Properties

Explosives and blasting agents are characterized by various properties that indicate how they will perform under field conditions. These properties include fume class, density, water resistance, temperature effects, detonation velocity, detonation pressure, borehole pressure, sensitivity, and strength. Each of these properties will be covered.

2.3.2.1 Fume Class

Class-Fumes are noxious gases that are produced from the detonation of explosives. The production of these gases is most critical in underground and other confined workings. Many factors affect the volume of poisonous gas produced including oxygen balance and adverse loading of explosives. The fume class is a measure of the toxic gases in cubic feet per 0.44 lb (200 g) of un-reacted explosive. Institute of Makers (IME) has developed a fume class classification scheme as seen in Table 2-4. The now-disbanded US Bureau of Mines (USBM) limits the volume of poisonous gases produced by permissible explosives (those used in underground coal and other gaseous mines) to 2.5 lb (1.14 kg).

<table>
<thead>
<tr>
<th>Class</th>
<th>Volume of poisonous gas per 200g of explosive, in ft$^3$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.16</td>
</tr>
<tr>
<td>2</td>
<td>0.16-0.33</td>
</tr>
<tr>
<td>3</td>
<td>0.33-0.67</td>
</tr>
</tbody>
</table>

2.3.2.2 Density

The density of an explosive is defined as the weight per unit volume or the specific gravity. Commercial explosives range in density from 0.5 to 1.7. Explosives with a density less than 1 will float in water. Therefore, in water-filled holes, an explosive with a density greater than 1 is required. For certain granular explosives such as dynamite, density correlates to the energy released in a given borehole volume. However, for water-based explosives, this is not the case, and often the reverse is true. Density is most useful in determining the loading density or the weight of explosives one can load per unit length of borehole (in pound per foot or kilogram per meter). Note that knowledge of loading density is required for blast-design calculations, and is calculated in English units as:

$$LD = 0.3405 \rho D^2$$

Where: $ho$ is density
D is explosive column diameter in inches.
2.3.2.3 Water Resistance
The ability of an explosive to withstand exposure to water for long periods of time without loss of strength or ability to detonate defines the water resistance. A numerical rating is used based on the results of tests performed on the explosive. However, explosive manufacturers individually rate products based on a relative basis as good, fair, or poor rating. The presence of moisture in amounts greater than 5% dissolves chemical components in dry blasting agents and alters the composition of gases produced, contributing to the formation of noxious fumes and lower energy output. Gelled granular products have good water resistance, and certain water-based mixtures have an excellent rating.

2.3.2.4 Temperature Effects
Extreme low temperatures affect the stability as well as the performance of explosives. The sensitivity and detonation velocity are hampered for certain water-based explosives at low temperatures while dynamites can become dangerously unstable below freezing temperatures. Explosives manufacturers recommend the appropriate range of temperature for storage and use.

2.3.2.5 Detonation Velocity
The detonation velocity is the speed at which the detonation front moves through a column of explosives. For high explosives such as dynamite, the strength of an explosive increases with detonation rate. For dry blasting agents and water-based explosives, field loading conditions greatly affect detonation velocity. Such conditions include (not exclusive list):
- borehole diameter
- density
- confinement within the borehole
- the presence of water

The speed of detonation is important when blasting in hard, competent rock where a brisance effect is desired for good fragmentation. For most explosives, there is a minimum diameter $D_{\text{min}}$ below which detonation velocity increases nonlinearly with increasing borehole diameter as can be seen in Figure 2-33. Above $D_{\text{min}}$ the explosive has reached its steady-state velocity. At this point, all thermodynamic properties are at a maximum as the reaction front approaches a plane shock front. At diameters less than $D_{\text{min}}$, complete reactions do not take place, and less than ideal energy and pressure evolve from the slower detonation rates. This represents a loss in terms of dollars spent on explosive energy.
2.3.2.6 Detonation Pressure

The detonation pressure is the maximum theoretical pressure achieved within the reaction zone and measured at the C-J plane in a column of explosives. The actual pressure achieved is somewhat less than this maximum due to non-ideal loading conditions always present in practice and due to certain explosive formulation. Most commercial explosives achieve pressures in the range of $0.29$ to $3.48 \times 10^6$ psi (2 to 24 GPa). Although detonation pressure is related to the temperature of the reaction, a number of simplifying formulas are available for estimating detonation pressure for granular explosives based on detonation velocity and density, for example (in English units):

$$P = 0.00337 \rho V^2$$

where
- $P$ is detonation pressure in psi,
- $\rho$ is density XXX
- $V$ is detonation velocity in fps.

2.3.2.7 Borehole Pressure

Borehole pressure is the maximum pressure exerted within the borehole upon completion of the explosive reaction measured behind the C-J plane. Such measurements cannot be made directly and are done during underwater tests performed for energy and strength determinations. With the use of hydrodynamic computer models, theoretical calculations of borehole pressures are made. There is little agreement in the literature regarding specific estimates of actual borehole pressures. In general, pressures after detonation within the borehole are estimated to be less than 30% of the theoretical detonation pressure.
2.3.2.8 Sensitivity

The definition of explosive sensitivity is two-fold. It includes sensitivity against accidental detonations in addition to the ease by which explosives can be intentionally detonated. From the standpoint of safety and accidental detonations, the sensitivity of an explosive to shock, impact, friction, and heat determine its storage and handling characteristics. Standardized tests for high explosives have been adapted for commercial explosives that include the friction (pendulum), impact (fallhammer), and projectile tests, among others.

The term properly used to define the propagating ability of an explosive is sensitiveness. In this respect, tests such as the No.8 strength blasting cap test, air-gap test, and the minimum critical diameter test are used. The cap sensitivity test measures the minimum energy required for initiation and is used to classify explosives (e.g., cap sensitive vs. noncap sensitive products) or the ability to initiate an explosive directly with a standard cap.

The No.8 cap is an industry standard cap of specific dimensions and charge characteristics. The air-gap test measures the distance between the ends of adjacent cartridge explosives for which reliable initiation can be propagated from one cartridge to another. The critical diameter of an explosive is the smallest diameter at which an explosive will maintain a steady-state detonation. Below this critical diameter, explosives may deflagrate or "dead press." Dead pressing occurs when an explosive is densified to a point that no free oxygen is available to ensure the start or progression of detonation.

2.3.2.9 Strength

The strength of an explosive is a measure of its ability to break rock. The terms "weight strength" and "bulk strength" were useful many years ago when explosives were primarily comprised of nitroglycerin cartridges, packaged in 50 lb (23-kg) boxes. In recent years, with the development of bulk blasting agents and less sensitive ingredients, new testing methods have been established to determine relative energies for all commercial products regardless of ingredients or packaging. The performance potential of an explosive is a function of the detonation velocity and density, as well as the volume of liberated gases and the heat of the reaction. A number of methods are used to establish this energy including the use of theoretical computer models and tests such as crater, ballistic mortar, and underwater tests. Of these methods, underwater tests give the best correlation to rock-breakage performance. Underwater tests were developed to measure both the shock energy and the gas (bubble) energy released during the detonation of standard test samples. These energy values have been useful in predicting the rock-breaking capabilities of explosives for comparative purposes.
Other terminology widely used by manufacturers is based on the theoretical heat of reaction determined by explosive formulation. Absolute bulk strength (ABS) in calories per cubic centimeter and absolute weight strength (AWS) in calories per gram are computed from the heat liberated during the detonation and formation of gaseous end products. Note ABS and AWS can be computed from one another if density is known, and it is the volumetric basis of reaction heat which correlates with energy. Most manufacturers of explosives will include either value with technical product literature.

A mixture of ammonium nitrate and fuel oil (ANFO) is by far the most widely used commercial blasting product. Depending on the proportions of the mix, the heat of reaction is approximately 850 cal/g. As a dry, free-running blasting agent, ANFO is capable of being loaded or packaged at varying densities. For a typical density of 0.85 and an AWS of 850 cal/g, the ABS = (850 cal/g) (0.85) = 723 cal/cm³. Other common strength terms are the relative weight strength (RWS) and relative bulk strength (RBS) in which the relative measure of energy available per unit weight or volume of an explosive is compared to an equal weight or volume of the standard commercial explosive ANFO. The RWS and RBS are computed as a percentage of that available from ANFO.

**Example 1**
Determine the relative strengths of explosives A (ABS 645 cal/cm³ and density of 0.8) and B (ABS 980 cal/cm³ and density 1.25)

Relative Bulk Strength

\[
RBS_A = \frac{ABS_A}{ABS_{ANFO}} = \frac{645\text{cal/cm}^3}{723\text{cal/cm}^3} = 89.2\%
\]

\[
RBS_B = \frac{ABS_B}{ABS_{ANFO}} = \frac{980\text{cal/cm}^3}{723\text{cal/cm}^3} = 135.5\%
\]

Relative Weight Strength

\[
RWS_A = \frac{AWS_A}{AWS_{ANFO}} = \frac{645\text{cal/cm}^3}{(0.8)850\text{cal/cm}^3} = 94.9\%
\]

\[
RWS_B = \frac{AWS_B}{AWS_{ANFO}} = \frac{980\text{cal/cm}^3}{(1.25)850\text{cal/cm}^3} = 92.2\%
\]

Therefore on a volume basis, explosive A is less powerful than ANFO while explosive B is more powerful.
2.3.3 Thermochemistry of Explosives and the Detonation Reaction.

In order to achieve maximum rock-breaking efficiency from an explosive, an oxygen-balanced mixture is formulated to ensure the formation of nonnoxious gases upon detonation. With an oxygen-balanced formula, it is assumed that optimum values of thermodynamic properties such as energy, temperature, and pressure are achieved, and that these values are not affected by changes in the reactants (explosive mixture). This is the case for ideal reactions. Unfortunately, commercial explosives are non-ideal materials. Changes in the physical nature rather than the chemistry of the explosive mixture, such as particle size and borehole diameter, vary the rate of detonation and hence affect thermodynamic variables. Furthermore, many granular explosives, such as ANFO and nitroglycerin, do not completely react, whereas water-based blasting agents react more efficiently, releasing optimum energy as predicted by formulation.

It is possible, however, to estimate the thermodynamic properties of an explosive reaction, assuming an oxygen balance. The following calculations are made to illustrate the methods used to estimate explosive properties of interest. Although there are a number of methods available, the procedures selected herein are the least difficult to apply. Thermodynamic data used for these calculations are found in Table 2-5.

### Table 2-5: Thermodynamic data for some explosive components and gases.

<table>
<thead>
<tr>
<th>Explosive</th>
<th>Formula</th>
<th>C</th>
<th>H</th>
<th>O</th>
<th>N</th>
<th>Enthalpy, kcal kg⁻¹</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nitroglycerin</td>
<td>C₃H₅N₉O₉</td>
<td>13.2</td>
<td>22.02</td>
<td>38.62</td>
<td>13.2</td>
<td>-362.0</td>
</tr>
<tr>
<td>Ammonium nitrate</td>
<td>C₁₂N₃O₁₃</td>
<td>40.05</td>
<td>34.40</td>
<td>24.99</td>
<td>55.0</td>
<td>-1091.0</td>
</tr>
<tr>
<td>TNT</td>
<td>C₇N₃O₅</td>
<td>23.0</td>
<td>26.40</td>
<td>23.0</td>
<td>30.0</td>
<td>-62.5</td>
</tr>
<tr>
<td>Diesel fuel</td>
<td>C₁₄H₂₀O₁₂</td>
<td>46.0</td>
<td>30.0</td>
<td>20.0</td>
<td>10.0</td>
<td>990.0</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Gases</th>
<th>Formula</th>
<th>a</th>
<th>b</th>
<th>c</th>
<th>Enthalpy, kcal mole⁻¹</th>
</tr>
</thead>
<tbody>
<tr>
<td>Carbon dioxide</td>
<td>CO₂</td>
<td>6.95×10⁻³</td>
<td>8.53</td>
<td>-2.48</td>
<td>-94.05</td>
</tr>
<tr>
<td>Water (vapor)</td>
<td>H₂O</td>
<td>6.89</td>
<td>3.32</td>
<td>-0.34</td>
<td>-57.50</td>
</tr>
<tr>
<td>Nitrogen</td>
<td>N₂</td>
<td>6.30</td>
<td>1.02</td>
<td>-0.05</td>
<td>0</td>
</tr>
<tr>
<td>Oxygen</td>
<td>O₂</td>
<td>6.13</td>
<td>2.99</td>
<td>-0.81</td>
<td>0</td>
</tr>
</tbody>
</table>


2.3.3.1 Oxygen Balance

As previously explained, an oxygen-balanced explosive formula is one in which the amount of oxygen O₂ is sufficient to form the desired detonation gases CO₂, HO₂, and N₂ but not noxious fumes such as CO, NO, and CH₄ (if the mix is oxygen deficient), or free oxygen (if the mix has an excess of oxygen). Oxygen balance for an explosive component or a mixture of components is usually reported in percentages or as moles of monatomic oxygen. It is computed using the mass balance relationship for a reaction of 1 kg of explosive, assuming the formation of ideal gaseous products.

Example 2
Determine the oxygen balance for nitroglycerin, C₃H₅O₉N₃.
The mass-balance equation is:
\[ C_{13.21}H_{22.02}O_{39.62}N_{13.21} \rightarrow 13.21 \text{CO}_2 + 22.02/2 \text{H}_2\text{O} + 13.21/2 \text{N}_2 + (39.62/2 - 13.21 - 11.01/2) \text{O}_2 \]

from the above equation the \( \text{O}_2 \) is \((19.81 - 13.21 - 5.505) = 1.095\), therefore the reaction has an unbalance total of 2.19 moles of oxygen. Hence, the oxygen balance (OB) can be calculated by:
\[ \text{OB} = \frac{\text{O excess}}{\text{O available}} = \frac{2.19}{39.62} = 0.0553 = 5.53\% \]
Note that the OB for an oxygen deficient component is:
\[ \text{OB} = \frac{\text{O deficient}}{\text{O deficient} - \text{O required}} \]

Note that some authors (Persson) consider that oxygen balance is given as the mass of oxygen which needs to be removed or added to the composition to achieve oxygen balance. In this case, it is expressed as a fraction or percentage of the explosive formula mass, for example grams \( \text{O}_2/100 \) grams of explosives. A simpler method of calculating oxygen balance for explosives that contain only carbon, oxygen, nitrogen, and oxygen, is given by:
\[
\text{oxygen balance} = -\frac{32\left(\frac{x+y}{4} - \frac{w}{2}\right) \times 100}{\text{explosive molecular weight}}
\]
Where \( C_xH_yN_zO_w \)

Therefore, consider the previous example where nitroglycerin = \( C_3H_5O_9N_3 \).
\[
\text{oxygen balance} = -\frac{32\left(\frac{3+5}{4} - \frac{9}{2}\right) \times 100}{227.09} = 3.5228\% \text{ or } +0.0352 \text{ gO/g of } C_3H_5O_9N_3.
\]

### 2.3.3.2 Explosive Energy
Other parameters of explosive thermodynamics are best described in the readings assigned with this lecture.

**ADDITIONAL RESOURCES VII**
The material in the above submodule is further supplemented by:
- Lecture Module 3.3: Introduction to Explosives.
2.4 Explosive Products

Note that section 2.4.1 (below) is optional. Those interested in the history of explosives are welcome to become more informed. The remainder of this sub-section is mandatory readings.

2.4.1 History of Explosives

The history of explosives is interesting from its profound influence on world development. The key developments are summarized and described below:

Black Powder
- Saltpeter or “Nitre”
- Probably originated with the Chinese around the 10th Century (mostly for fireworks, rockets, etc.)
- Roger Bacon published a formula for Black Powder around 1242
  - In the 13th century Roger Bacon, a European, was interested in the new knowledge from far east.
  - He studied fireworks it and tested it over and over again. After many months he found the perfect ratio of saltpeter, sulfur, and a new ingredient, charcoal.
  - After he found out the perfect ratio he wrote the ingredients and the amounts in code in his diary.
  - Roger Bacon had made, and recorded, the first black powder (the early form of gunpowder).
  - Bacon did not get credit for the making of black powder because he didn't use his invention. Berthold Schwarts saw this and exploited it.
- Berthold Schwartz invented the gun around 1300 which resulted in further refinement of black powder.
  - Schwarts used the black powder to launch a pebble at high speed out of a metal tube.
  - Gunpowder also sped up the very slow process of digging up stones.
  - With gunpowder they could blow the stones out of the ground.
  - Now even some poor people could have a house of stone. This was a great technological step.
- Blasting with powder replaced “fire setting” for loosening rock around the beginning of the 17th Century.

Mercury Fulminate
- Discovered by Howard in 1800
- Later used as detonator for dynamite by Alfred Nobel.

Nitroglycerin
- Nitroglycerin (NG) and Nitrocellulose (NC) discovered by Ascanio Sobrero in 1846 in Switzerland
- He was afraid of it, destroyed his notes and warned against its use

**Dynamite**
- Alfred Nobel and his father built a NG small factory in 1861 in Sweden
- Loading and transporting nitroglycerin was dangerous (liquid poured in to holes and ignited with various types of black powder igniters.
- NG proved to be very dangerous and resulted in the death of many people including his brother Emil.
- Nobel discovers dynamites by accident (Dynamite is derived from the word Dynamis, meaning power):
  - When ‘blasting oil’, NG spilled into kieselguhr (NG was packed in it), Nobels saw that the kieselguhr absorbed about 3 times its weight of NG.
  - Nobel began to sell the 75/25 NG/kies., the first of the dynamites. Eventually went to wood pulps which increased the energy output of the NG.
  - This development allowed the relatively safe transport transportation of NG.
  - Patented in 1867
- Nobel knew the amount of destruction his invention would cause and he did not want to be associated with thousands of deaths, so he left a large amount of money to the awarding of prizes in science, literature, politics, etc... every year.
- Pros: more powerful than black powder, higher detonation velocity and more effective in breaking rock (not only moved, but broke them apart)
- Cons: Would freeze, nitro fumes and nitro headaches

**Safety Fuse**
- William Bickford of England devised the safety fuse, originally a textile-wrapped cord with a black powder core, which for the first time enabled safe, accurately timed detonations.
- In 1865 Nobel invented the blasting cap, providing the first safe and dependable means for detonating nitroglycerin and thereby considerably expanding its use for industrial purposes.
- Electrical firing, first used successfully in the late 19th century, allows greater control over timing.

**Ammonium Nitrate**
- First synthesized in 1659 by J. R. Glauber by combining ammonium carbonate and nitric acid.
- Two major uses: Fertilizer and Blasting
- Initially used to replace a portion of nitroglycerin in dynamite
- Dupont introduced NITRAMON in 1935
- Dupont called his products “Blasting Agents” due to their safe handling, low cost and non-headaches
Two disastrous shipload explosions resulted in development of ANFO (Ammonium Nitrate & Fuel Oil) Texas City, Brest France.

In 1955, Ammonium nitrate/fuel oil (ANFO) was discovered to greatly increase the energy output of AN prills.

Watergel slurries invented simultaneously by Dupont USA and CIL Canada.

Further sensitized by powered aluminum.

ANFO eventually replaced dynamite in dry holes.

Pros: easier to load, cheaper and safer than dynamite.

Cons: desensitized by water.

What can be concluded about the history of explosives are the key aspects in the design of commercial explosives:

- Explosive power
- Safety in transportation and handling
- Cost
- Controllable and predictable detonation.

2.4.2 Energetic material

Energetic materials are all materials that can undergo exothermal chemical reaction releasing a considerable amount of thermal energy. An explosion is basically any rapid expansion of matter into a volume much larger than the original. An explosive is a material that can undergo an exothermal chemical reaction resulting in a rapid expansion of the reaction products into a volume larger than the original. Figure 2-34 shows a simplified version of an explosives family tree.

![Explosives Family Tree](image)

**Figure 2-34: Explosives Family Tree**
2.4.3 Explosives and Propellants
The difference between an explosive and a propellant is often functional rather than fundamental. Explosives are intended to function by detonation following shock initiation by a detonator or a booster charge. Propellants are intended to burn steadily at a rate determined by the design pressure of the rock or gun breech, and they are ignited to burning by a flame that provides a spray of hot burning particles.

2.4.4 Single Molecule and Composite Explosives
There are two fundamentally different kinds of explosive materials, namely single explosive substances and composite explosive mixtures. Single explosives are chemical substances that contain in one well-defined molecule all that is needed for an explosion. The molecule decomposes into mainly gaseous reaction products, such as CO₂, N₂, and H₂O. The solid explosive trinitrotoluene, TNT (C₇H₅(NO₂)₃), and nitroglycerine, are examples of single explosive substances.

A composite explosive can be a mixture of two single explosive substances, a mixture of a fuel and oxidizer, or an intermediate mixture containing one or more single explosive substances together with a/or oxidizer ingredients.
Table 2-6: Single chemical explosive substances.

<table>
<thead>
<tr>
<th>Common name, Composition</th>
<th>Symbol</th>
<th>Mol. weight g/cm³</th>
<th>( \rho_{\text{max}} ) g/cm³</th>
<th>( D_i ) (km/sec)</th>
<th>( p_i ) kbar</th>
<th>( Q_d ) kJ/g</th>
<th>( V_{\text{gas}} ) cm³/g</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Primary Explosives:</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cupric azide, ( Cu(N_3)_2 )</td>
<td></td>
<td>147.6</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hydrazoic acid, ( HN_3 )</td>
<td></td>
<td>43.0</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mercury fulminate, ( HgC_2N_2O_2 )</td>
<td></td>
<td>284.7</td>
<td>4.42</td>
<td>5.05</td>
<td>5.4</td>
<td>1.79</td>
<td>315</td>
</tr>
<tr>
<td>Lead stypnate, ( PbC_2H_2N_3O_9 )</td>
<td></td>
<td>468.3</td>
<td>3.10</td>
<td>5.2</td>
<td>1.54</td>
<td>407</td>
<td></td>
</tr>
<tr>
<td>Lead azide, ( PbN_6 )</td>
<td></td>
<td>291.3</td>
<td>4.71</td>
<td>5.1</td>
<td>1.53</td>
<td>308</td>
<td></td>
</tr>
<tr>
<td>Silver azide, ( AgN_3 )</td>
<td></td>
<td>149.9</td>
<td>5.1</td>
<td>5.90</td>
<td></td>
<td></td>
<td>224</td>
</tr>
<tr>
<td>Tetrazene, ( C_2H_8N_10O )</td>
<td>DDNP</td>
<td>188.2</td>
<td>1.7</td>
<td></td>
<td></td>
<td>2.75</td>
<td>1190</td>
</tr>
<tr>
<td>(guanylnitrosoaminoguanyltetrazene)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Diazodinitrophenol, ( C_6H_2N_2O(NO_2)_2 )</td>
<td></td>
<td>210.1</td>
<td>1.63</td>
<td>7.10</td>
<td>3.43</td>
<td>856</td>
<td></td>
</tr>
<tr>
<td><strong>Secondary Explosives, Liquids:</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Methyl nitrate, ( CH_3NO_3 )</td>
<td></td>
<td>77.0</td>
<td>1.22</td>
<td>6.30</td>
<td>6.17</td>
<td>909</td>
<td></td>
</tr>
<tr>
<td>Nitroglycerin</td>
<td>NG</td>
<td>227.1</td>
<td>1.60</td>
<td>7.58</td>
<td>220</td>
<td>6.30</td>
<td>715</td>
</tr>
<tr>
<td>(glyceroltrinitrate), ( C_3H_5(NO_3)_3 )</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ethyleneglycoldinitrate, ( C_5H_4(NO_3)_2 )</td>
<td></td>
<td>152.1</td>
<td>1.48</td>
<td>7.30</td>
<td>6.83</td>
<td>737</td>
<td></td>
</tr>
<tr>
<td>Tetranitromethane, ( C(NO_2)_4 )</td>
<td>TNM</td>
<td>196.0</td>
<td>1.65</td>
<td>6.36</td>
<td>159</td>
<td>2.29</td>
<td>686</td>
</tr>
<tr>
<td>Nitroform (trinitromethane), ( CH(NO_2)_3 )</td>
<td></td>
<td>151.0</td>
<td>1.60</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dinitromethane, ( CH_2(NO_2)_2 )</td>
<td></td>
<td>106.1</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Nitromethane, ( CH_3NO_2 )</td>
<td>NM</td>
<td>61.0</td>
<td>1.13</td>
<td>6.29</td>
<td>141</td>
<td>6.4</td>
<td>723</td>
</tr>
<tr>
<td>Ethylenedinitramine, ( C_2H_6N_2(NO_2)_2 )</td>
<td></td>
<td>150.1</td>
<td>1.5</td>
<td>7.61</td>
<td>4.42</td>
<td>908</td>
<td></td>
</tr>
<tr>
<td>Isopropyl nitrate, ( C_3H_7NO_3 )</td>
<td>IPN</td>
<td>105.1</td>
<td>1.04</td>
<td>5.4</td>
<td>85</td>
<td>2.36</td>
<td></td>
</tr>
</tbody>
</table>

*Detonation velocity at initial density of 4.113 g/cm³.*
### Table 2-7: More Single Chemical Explosives

<table>
<thead>
<tr>
<th>Common name, Composition</th>
<th>Symbol</th>
<th>Mol. weight</th>
<th>$\rho_{max}$</th>
<th>$D_i$ (km/sec)</th>
<th>$\rho_{max}$</th>
<th>$\gamma$</th>
<th>$Q_d$</th>
<th>$V_{ge}$</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Secondary Explosives, Solids:</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mannitohexanitrate, (Nitromannit), $C_6H_8(NO_3)_6$</td>
<td>HN</td>
<td>452.2</td>
<td>1.73</td>
<td>8.26</td>
<td>6.02</td>
<td>755</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Trinitroxadiobenzene, $C_6H_2N_3(NO_2)_3$</td>
<td>NC</td>
<td>254.1</td>
<td>1.75</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Nitrocellulose, $C_{24}H_{10}O_{20-24}(NO_3)_x$</td>
<td>PETN</td>
<td>316.2</td>
<td>1.77</td>
<td>5.55</td>
<td>7.98</td>
<td>300</td>
<td>6.12</td>
<td>780</td>
</tr>
<tr>
<td>Pentaoxythiophetetranitrate, $C_5H_6(NO_2)_4$</td>
<td>DiTeU</td>
<td>386.2</td>
<td>1.79</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dinitroethylyurea, $C_6H_2N_2O(NO_2)_2$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Trinitrophenylmethylnitramine (Tetryl), $C_7H_5N(NO_2)_4$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Nitrostarch (penta), $C_{12}H_5O_5(NO_3)_3$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hydrazine nitrate, $N_2H_4\cdot HNO_3$</td>
<td>HNIW</td>
<td>549.2</td>
<td>1.60</td>
<td>8.69</td>
<td>3.83</td>
<td>1001</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hexanitrohexaazaisowurzitan, $C_6H_6N_2(NO_2)_6$</td>
<td>RDX</td>
<td>434.2</td>
<td>2.1</td>
<td>10.2</td>
<td>450 (calculated)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cyclotrimethylenetrinitramine (Hexogen), $C_5H_6N_2(NO_2)_3$</td>
<td>HMX</td>
<td>222.1</td>
<td>1.80</td>
<td>6.08</td>
<td>8.75</td>
<td>347</td>
<td>5.46</td>
<td>908</td>
</tr>
<tr>
<td>Cyclotetramethylenetetranitramine (Octogen), $C_6H_4N_4(NO_2)_4$</td>
<td>DATB</td>
<td>296.2</td>
<td>1.90</td>
<td>9.10</td>
<td>393</td>
<td>5.46</td>
<td>908</td>
<td></td>
</tr>
<tr>
<td>Diamonitrotrinitrobenzene, $C_6H_6N_2(NO_2)_3$</td>
<td>HNB</td>
<td>243.2</td>
<td>1.79</td>
<td>7.52</td>
<td>259</td>
<td>4.67</td>
<td>625</td>
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<tr>
<td>Hexanitrobenezene (hexyl), $C_6N_6O_6$</td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>Hexanitrothilene, $C_6H_6(NO_2)_6$</td>
<td>HNS</td>
<td>252.1</td>
<td>1.97</td>
<td>9.30</td>
<td>355</td>
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<td></td>
</tr>
<tr>
<td>Trisazidotrinitrobenzene, $C_6N_6(NO_2)_3$</td>
<td>TNTAB</td>
<td>450.2</td>
<td>1.79</td>
<td>7.14</td>
<td>221</td>
<td>4.12</td>
<td>700</td>
<td></td>
</tr>
<tr>
<td>Picric acid, (trinitrophenol), $C_6H_3O_3(NO_2)_3$</td>
<td>PYX</td>
<td>336.2</td>
<td>1.74</td>
<td>8.58</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Picrylaminodinitroprpyridine, $C_{17}H_2N_2(NO_2)_3$</td>
<td></td>
<td></td>
<td></td>
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<td></td>
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<td></td>
</tr>
<tr>
<td>Ammonium picrate, $C_6H_6N_2(NO_2)_3$</td>
<td></td>
<td></td>
<td></td>
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<td></td>
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<td></td>
</tr>
<tr>
<td>Nitrotriazolone, $C_2H_2N_2O_2(NO_2)_3$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Trinitroacetide, $C_6H_6N_4O_4$</td>
<td>NTO</td>
<td>130.1</td>
<td>1.93</td>
<td>8.42</td>
<td>316</td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>Nitroguanidine, $C_6H_3N_2O_3$</td>
<td>TNZ</td>
<td>192.1</td>
<td>1.84</td>
<td>8.8$^b$</td>
<td>380$^b$</td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>Trinitrobenzene, $C_6H_3(NO_2)_3$</td>
<td>NQ</td>
<td>105</td>
<td>1.70</td>
<td>5.46</td>
<td>8.20</td>
<td>2.88</td>
<td>1075</td>
<td></td>
</tr>
<tr>
<td>Ethylenedinitramine, $C_2H_2N_2(NO_2)_2$</td>
<td>TNR</td>
<td>213</td>
<td>1.76</td>
<td>7.30</td>
<td>4.81</td>
<td>678</td>
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<td></td>
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<tr>
<td>Monomethyamine nitrate, $C_2H_2N_2(NO_2)_2$</td>
<td>EDN</td>
<td>150.1</td>
<td>1.5</td>
<td>7.57</td>
<td>4.42</td>
<td>908</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ethylenediamininedinitrate, $C_2H_2N_2(NO_2)_2$</td>
<td>MMAN</td>
<td>94.1</td>
<td>1.42</td>
<td>7.15</td>
<td>214</td>
<td>2.88</td>
<td>800</td>
<td></td>
</tr>
<tr>
<td>Trinitrotoluene (Trotlyl), $C_7H_4(NO_2)_2$</td>
<td>TNT</td>
<td>227</td>
<td>1.64</td>
<td>5.01</td>
<td>6.95</td>
<td>190</td>
<td>4.10</td>
<td>690</td>
</tr>
<tr>
<td>Triaminitrotrinitrobenzene, $C_6H_2N_2(NO_2)_3$</td>
<td>TATB</td>
<td>258.2</td>
<td>1.90</td>
<td>7.98</td>
<td>315</td>
<td>3.12</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dinitrotoluene, $C_7H_4(NO_2)_2$</td>
<td>DNT</td>
<td>182.1</td>
<td>1.54</td>
<td>3.27</td>
<td>800</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

- $^a$$e = 12$. $^b$Calculated using $\gamma = 2.76$. $^c$Because of their low detonation temperature, AN and AP are unlikely to sustain detonation at crystal density even at very large charge diameters.
2.4.5 Primary, Secondary, and Tertiary explosives.

Single molecule explosives range with respect to the strength of the stimulus required for initiation or a self-supporting chemical decomposition reaction: from primary explosives, such as lead azide (PbN₆), which are used as igniting charges in detonators; through the secondary explosives, of which NG, NM, and TNT are examples; to tertiary explosives, of which ammonium nitrate, AN (NH₄NO₃), is an example.

The primary explosives are able to transit from surface burning to detonation within very small distances. A 0.2 mm thick grain of lead azide when ignited will transit from burning to detonation within a distance less than the grain thickness. This is because the lead azide molecule is very simple, decomposing in a very simple two-step reaction, and also because the reaction products have a high molecular weight. Reaction products are generated at the surface faster than they can expand away from the surface, which results in a quick build-up of pressure at the burning surface.

The secondary explosives, too, can burn to detonation, but only in relatively large quantities. For example, a stick of dynamite can burn as a candle, slowly, if ignited with a flame (it is strongly advised against performing a demonstration of this without due precautions against the chance event of a detonation), whereas a truckload of dynamite may burn to detonation.

Under normal conditions, tertiary explosives are extremely difficult to explode and are in fact officially classed as non-explosives provided that certain conditions are observed (such as that an oxidizer not be mixed with fuels or sensitizers, and that the grain size exceeds a certain minimum size.) They are nonetheless explosive, as demonstrated by some of the largest accidental explosions in history, such as the (April 16, 1947) Texas City explosion of ammonium nitrate. A more detailed classification scheme for explosive material can be seen in Figure 2-35 which shows primary, secondary, and tertiary explosives and the differences between single and composite explosives.

2.4.6 Commercial Explosives

Figure 2-34 shows the breakdown of commercial explosive categories. This classification scheme shows how military explosives are often segregated from military applications. Examples of military explosives include TNT, PETN, RDX (cyclonite), Tetryl, and compositions such as A3, B, B4, C4, which are mixtures of RDX, TNT, and additives to provide a moldable consistency. These explosives are chiefly used in the weapons industry. However, small amounts of HE are added to commercial explosives to increase strength and sensitiveness. Many commercial or industrial explosives are classified as HE because they contain critical amounts of military explosives or nitroglycerin, and usually they are cap sensitive. Others, such as dry blasting agents, are not classified as HE, and require boosters or primers of HE for initiation.
Industrial explosives are classified as one of the following:
- nitroglycerin-based,
- dry blasting agents,
- water gels,
- emulsions,
- permissibles, (for underground coal and will not be discussed)
- primers, and boosters.

Two-component explosives, a common category, actually contain mixtures or characteristics that fall in other classifications. Often the difference among these products is formulation; however, product packaging and consistency can also change a classification.

Explosive components are referred to as:
- Oxidizers: Oxidizers contribute oxygen for oxygen balance, and include nitrated salts such as ammonium nitrate (AN), sodium nitrate (SN), and calcium nitrate (CN).
- Fuels: include fuel oil, carbon, granular aluminum, TNT, black powder, or any carbonaceous material that produces heat. Many of these components are also referred to as sensitizers and can also act as absorbents.
- absorbents, are products, such as wood pulp, sawdust, cotton, and cellulose, that incorporate liquid explosive components such as nitroglycerin.
- Stabilizers: include flame retardants, gelatins, densifiers, water, gum, emulsifying agents, and thickeners.

2.4.7 Nitroglycerin-based Explosives
Dynamite is a trade name introduced by Alfred Nobel. It is comprised primarily of a stable yet powerful mix of nitroglycerin (nitrostarch). Since its invention, a number of nitroglycerin (NG)-based products have been developed of three basic types: granular, gelatin, and semigelatin, which are all considered HE.

Gelatins and semi gelatins contain nitrocotton that combines with NG to form a gel structure whose consistency is controlled by the percentage of nitrocellulose. Dynamites are packaged in cylindrical cartridges from 7/16 in. (22 mm) in diameter and 8 to 24 in. (203 to 610 mm) in length. The quality of the waxed paper wrapping is important for:
- water resistance,
- fume production,
- ease and safety of loading.

Straight dynamite derives its energy source from NG, SN, and AN, including absorbants such as wood pulp and flour that also act as combustibles.
Ammonia dynamite (or "extra" dynamite) is a granular mix that contains a smaller quantity of NG mixed with AN and SN.

Gelatin dynamites are either straight gelatin or ammonium (extra) gelatin. Each has similar mixtures as straight and ammonia (extra) dynamites with the addition of nitrocellulose for a gel consistency.

Semigelatins are ammonia gelatin with a small amount of nitrocellulose and a 65% weight strength. They are also used as primers and boosters.

2.4.8 Dry Blasting Agents

Dry blasting agents are one form of a general category of blasting agents. A blasting agent is, by definition, a mixture of fuel and oxidizer. It is not classified as an explosive, and cannot be detonated with a No.8 blasting cap. A dry agent is a granular, free-running mix of a solid oxidizer (usually AN), prilled into porous pellets onto which a liquid fuel oil or propellant is absorbed. ANFO is the most widely used blasting product, with approximately 94.5% industrial-grade ammonia nitrate and 5.5% fuel oil. Figure 2-36 shows the varying effect of the addition of fuel oil. No.2 grade diesel fuel oil is used for a nearly oxygen-balanced mix.

![Figure 2-36: Energy output vs. percent fuel oil added to ammonium nitrate](image)

Typical values of specific gravity range from 0.75 to 0.95. The properties of dry blasting agents vary significantly with borehole diameter, density, confinement, particle size, water conditions, and size of primer used for initiation. Figure 2-37 shows the varying effect of diameter confined on various explosives. The steady state detonation velocity of ANFO is over 15,000 fps (4500 m/s) and is achieved in borehole diameters greater than 15 in. (381 mm). The critical diameter of ANFO is between 2 and 4 in. (51 and 102 mm) and is a topic of controversy among blasters and, in particular, those who blast underground using small-diameter holes. The exact values of
the critical diameter depends on the loading conditions; however, ANFO does not detonate reliably within the range cited above.

Aluminum in granular form can be added to ANFO to increase the heat or energy output. For increasing percentages of aluminum by weight up to 6%, a measurable increase in fragmentation energy is noted. The cost of additional aluminum beyond 6% does not result in proportionally increased work output and, therefore, is not cost effective.

### 2.4.9 Wet Blasting Agents

Blasting agents that contain more than 5% water by weight are referred to as wet blasting agents. Within this category are:

- water gels or slurries,
- emulsions, and
- heavy ANFO.

Heavy ANFO is a combination of prilled ANFO and emulsion. The development of wet blasting agents, led by slurries in the 1950s, came about in response to the disadvantages of ANFO in certain applications. These were:

- lack of water resistance
- low bulk strength due to low density.
Table 2-8: Typical Compositions of selected slurries and Emulsions

<table>
<thead>
<tr>
<th>Aluminum Sensitized Slurry</th>
<th>Water Gel Slurry</th>
</tr>
</thead>
<tbody>
<tr>
<td>10% Aluminum</td>
<td>13% Amine nitrate</td>
</tr>
<tr>
<td>15% Water</td>
<td>15% Water</td>
</tr>
<tr>
<td>5% Ethylene glycol</td>
<td>5% Sodium nitrate</td>
</tr>
<tr>
<td>44% Ammonium nitrate</td>
<td>3% Ammonium perchlorate</td>
</tr>
<tr>
<td>25% Calcium nitrate</td>
<td>63% Ammonium nitrate</td>
</tr>
<tr>
<td>1% Guar gum</td>
<td>1% Guar gum</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Explosives Sensitized Slurry</th>
<th>Emulsion</th>
</tr>
</thead>
<tbody>
<tr>
<td>25% TNT, smokeless powder or</td>
<td>6% Wax/oil</td>
</tr>
<tr>
<td>nitrostarch</td>
<td>2% Emulsifier</td>
</tr>
<tr>
<td>25% Water</td>
<td>14% Water</td>
</tr>
<tr>
<td>15% Sodium nitrate</td>
<td>76% Ammonium nitrate</td>
</tr>
<tr>
<td>44% Ammonium nitrate</td>
<td>2% Hollow microballoons</td>
</tr>
<tr>
<td>1% Guar gum</td>
<td></td>
</tr>
</tbody>
</table>

Table 2-8 shows some typical compositions of water gels and emulsions. The critical diameter of wet blasting agents is often less than 1 in. (25 mm). Three varieties of wet blasting agents are in common use in the mining industry: slurries, emulsions, and Heavy ANFO.

2.4.9.1 Slurries

Slurries, or water gels, are a colloidal suspension of solid AN particles suspended in a liquid AN solution that is gelled, using cross-linking agents. The gels (guar gum) effectively surround the solid AN, rendering the oxidizer water resistant while thickening the explosive mix. Fuels and sensitizers such as TNT, nitrostarch, Composition B, ethyl alcohol, fuel oil, and glass bubbles (microspheres) are dissolved or added to the liquid phase. Granular aluminum, added as a sensitizer, increases weight and bulk strength. Up to 18% aluminum by weight has been found to provide increased energy output. In general, 20% water is used. Certain mixtures, containing high-explosive sensitizers, are cap sensitive and hence should not be classified as a blasting agent, but rather as a slurry explosive.

Slurries are characterized by:
- excellent water resistance,
- high density and bulk strength,
- good oxygen balance, confinement, and coupling within the borehole.

2.4.9.2 Emulsions

Definition of emulsion: A suspension of small globules of one liquid in a second liquid with which the first will not mix. Emulsion explosives and
blasting agents are the latest development away from ingredients that are in themselves explosive substances. Emulsions are a two-liquid phase containing microscopic droplets of aqueous nitrates of salts (chiefly AN) dispersed in fuel oil, wax, or paraffin using an emulsifying agent. The water-in-oil structure depends on entrapped air or microspheres for sensitivity, thereby eliminating the need for expensive explosive compounds. Microspheres, microscopic glass, or plastic air-filled bubbles and the AN droplets form the oxidizer, while the fuel oil exists as the oil phase. In the resulting margarine- or Vaseline-like, smooth mixture, the AN solution stays as a super cooled liquid without crystallizing even upon cooling to sub-zero temperatures. By distributing in it finely dispersed voids in the form of glass microballoons or gas bubbles, those that can act as hot spots to initiate the chemical reaction upon shock compression, a variety of emulsion explosives or blasting agents of different sensitivity levels can be produced. As they contain no ingredient that is an explosive in itself, and also because of the desensitizing effect of the water content, all such emulsions have a high degree of inherent safety.

In contrast to ANFO which cannot be used in water-filled drillholes because of the high water solubility of AN, emulsion explosives have excellent water resistance since each AN/water droplet is surrounded by a thin film of oil which repels water. The extremely small droplet size, and the sub micron thickness of the oil film gives a very large contact area between the fuel and the oxidizer solution; the intimacy of mixing of the fuel and oxidizer approaches that of a solution. Over time, the oil phase migrates, and droplets combine to form larger particles sizes whose bulk surface area is reduced. Less fuel is in contact with the oxidizer, and less than ideal explosive properties are achieved.

Emulsions provide high detonation pressures of 1.45 to 1.74 x 10^6 psi (10 to 12 GPa). Densities range from 1.15 to 1.45. Emulsions have excellent water-resistant properties regardless of packaging. The cost of emulsion products is within the range for slurries.

Emulsions can be mixed on site and pumped from bulk trucks. Premixed emulsions are available in plastic tubes in a variety of diameters and lengths. Depending on product diameter and sensitziers used, emulsions can be cap sensitive

2.4.9.3 Heavy ANFO
Heavy ANFO is a product comprised of up to 45 to 50% ammonium nitrate emulsion mixed with prilled ANFO. It was developed in an attempt to increase the bulk density of ANFO. The only fuel component is in the ANFO (or a liquid fuel), while the emulsion contains no solid fuel, making the mixture a "repumpable" consistency. The final product has improved strength and provides good water resistance in comparison to ANFO, with a price
range between that of ANFO and emulsions. Research has shown, however, that the ability of emulsion to prevent ANFO from being dissolved in the presence of water, thereby reducing blasting efficiency, is questionable.

2.4.10 Primers and Boosters-
A primer charge is an explosive ignited by an initiator, which, in turn, initiates a non-cap-sensitive explosive or blasting agent. A primer contains cap-sensitive high explosive ingredients. Often cartridges of dynamites, highly sensitized slurries, or emulsions are used with blasting caps or detonating cord. Other primers are cast into specific shapes and weights, using TNT and PETN, designed with wells for initiator acceptance.

Boosters are highly sensitized explosives or blasting agents, used either in bulk form or in packages of weights greater than those used for primers. Boosters are placed within the explosive column where additional breaking energy is required. Often-times, cartridge or plastic-bagged dynamites or sensitized wet blasting agents are used as primers as well as boosters. Boosters are often used near the bottom of the blasthole at the toe level as an additional charge for excessive toe burden distances. They are also placed within the explosive column adjacent to geological zones that are difficult to break or intermittently within the main explosive charge to ensure continuous detonation.

2.4.11 Initiators and Initiation Systems
Initiators are devices containing high explosives that, upon receiving an appropriate mechanical, or electrical impulse, produce a detonation or burning action. Initiators are used as components within a system of explosives and other devices to start the detonation of all other components. Initiation systems are either electric or nonelectric, and include blasting caps, safety fuse, detonating cord, or non-electric shock tubes.

2.4.12 Electric Caps
Electric blasting caps are a commonly used method of initiation. Electrical energy (ac or dc) is sent through copper or iron legwires to heat an internal-connecting bridgewire. This heat, in turn, starts a chain reaction of explosives burning within the metal cap shell, through a powder delay train. This process detonates a high-explosive base charge, igniting a cap-sensitive explosive. They are manufactured with an instantaneous (no delay train) time of initiation, or time delays (in milliseconds) used in delayed blasting practices. Time delays with intervals of 25, 50, 100, 500, and 1000 ms are available for short- (ms) or long-period (LP) delays. Table 2-9 summarizes typical delay time intervals available for electric and non-electric initiating systems. Short delays are used in surface blasting operations, while longer delays are used underground where blasting conditions are more confined.
The use of time delays in blasting enhances fragmentation and the control of ground vibrations. In recent years, improvements have been made in the manufacturing of blasting caps that increase the accuracy in detonation time. The next generation of high-precision detonators will contain an electronic circuit instead of pyrotechnical delay elements. The integrated circuits will permit microsecond rather than millisecond timing accuracy and allow programmability for onsite selection of each cap detonation timing. Ac power lines and capacitor-discharge dc power sources approved for blasting are used to energize caps. Precise calculations are needed to determine the entire blasting circuit resistance, including all accessory connecting wires. This is to ensure that the power source supplies the correct current to each cap in the circuit. Safe blasting practices dictate that precautions are used to avoid blasting in the vicinity of extraneous electricity such as stray current, static electricity, electrical storms, and radio frequency energy when using electric caps.

**Table 2-9: Typical Delay times for donators**

<table>
<thead>
<tr>
<th>Short-Period Delays</th>
<th>Long-Period Delays</th>
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<tbody>
<tr>
<td><strong>Electric</strong></td>
<td><strong>Nonelectric</strong></td>
</tr>
<tr>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>1</td>
<td>25</td>
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<tr>
<td>19</td>
<td>900</td>
</tr>
<tr>
<td><strong>1000</strong></td>
<td></td>
</tr>
</tbody>
</table>

2.4.13 Non-electric caps

Non-electric initiation systems include a cap similar to that of an electric cap, but they are connected to plastic tubing or a transmission line that carries an initiation (shock and heat) to initiate the cap. The energy source in the tubing is either a gas mixture or an internal coating of special explosive. Non-electric tubing is not used in underground coal or gassy mines as it carries an open flame. The plastic tube itself does not detonate; therefore, the only noise source is the cap itself. Caps and tubes of varying lengths are connected with special connectors between holes to configure unique blast pattern arrays. Surface delay elements, when used in conjunction with in-
hole delays, provide nearly infinite numbers of delays in blasting patterns. Delays are available in short and long periods as well as in-hole and surface delays. The advantage of non-electric systems over electric systems is the ability to design blasts with a greater number of holes than traditional electric blasting. In addition, concerns about the effects of accidental detonations of electric caps due to stray currents are eliminated with the use of non-electric systems.

Figure 2-38: Generic Design detonators

2.4.14 Electronic Detonators
Electronic detonators have an electronic counter on a microchip in place of the pyrotechnic delay charge, and a capacitor to supply the discharge energy for ignition. Advantages compared to regular electric or nonel detonators:

- Higher timing precision (10 μs instead of 1-10 ms delay scatter)
- Same high timing precision at long delay times (10 μs at 5 second delays)
- Increased control over time delay
- Greater safety against accidental ignition (coded firing signal)

Current disadvantages include:

- Higher price because of chip and capacitor cost
- Back to electric wiring – risk of ground faults or poor contacts

2.4.15 Detonating cord
Detonating cord is a flexible but strong continuous detonator that can be several hundred meters long. A detonator is required to initiate a length of
detonating cord which cannot be normally initiated by fire. Detonating cord has two functions:

- to provide simultaneous detonation of several interconnected blasthole charges, thus avoiding the need for multiple electric or plain detonators
- to provide continuous initiation of the full length of an explosive column in a blasthole, as distinct from point initiation with individual detonators.

Detonating cord (sometimes known as Cordtex) consists of a core of PETN enclosed in a tape wrapping that is further bound by counter-laced textile yarns. The cord is either reinforced or completely enclosed by strong waterproof plastic. Detonating cords are available with a variety of charge weights, tensile strengths and protective coatings, depending on the application. Their energy release depends on the amount of PETN in the core, which generally varies from 1.5 g/m to 70 g/m. However, 10 g/m is the PETN weight of standard detonating cord whose VOD is about 7000 m/s.

**ADDITIONAL RESOURCES VIII**
The material for this sub-module is additionally supplemented by:

- Lecture module 3.4: Explosive products.
Module 3: Blast Design

The section will cover the basic aspects of blast design in terms of geometry, explosives, geology, sequencing, and initiation. Note that these are the key theory, tools and techniques used for most blasting applications. More unique applications of blasting will be covered in the next module, along with underground blast design.

Blast Design Basics

Blast design is a semi-empirical systematic method that involves balancing numeric and qualitative assessments of rock properties, explosives, and desired products.

System Approach

The first step in the process of design is to determine the goal or purpose of the intended design. From a historical perspective, it has been seen that

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7 These notes were assembled directly from the following references:
- From Dr. Paul Lever’s 415 course notes Hartman, Howard L. Ed. SME Mining Engineering Handbook. 2nd ed. 1992
- Stefanko, Robert. Coal Mining Technology Theory and Practice. Littleton CO.: Society of Mining Engineers. 1983
- Mining-Technology.com, search: continuous mining
- Caterpillar Performance Handbook, 28th Ed.
- 2001 Class notes, from Bob Cummings
- The History of Explosives. http://sis.bris.ac.uk/~dj9006/explosives/history.html
drilling and blasting has been a process by which the mine has undertaken full responsibility. Its key goal was to deliver to the mill the least expensive product. This often limited the blast design to enable the efficient loading and hauling of the material from the stope to the crushing circuit. However, the more recent ‘systems design’ approach whereby the scope changes from simply the mine to the overall mine-mill system, an optimum design for simply the mine will most likely change when the mill is included.

**Figure 3-1: Systems Approach**

In visualizing the system, the variables upon which an engineer is to optimize based on lowest overall cost can be seen in Figure 3-2. For example, as can be seen, the internal and external environments specify a minimization of wall damage, flyrock, noise, and other detrimental aspects of working with explosives. These aspects of the design are facilitated in that they are often specified by law. The internal environment is also less limiting as rock damage to walls that are not permanent are also not to dominate the design. Other constraints not shown are the limitations posed by pit design, limited operating room, bench sizes, equipment, etc... The proper fragmentation to optimize the loading& hauling, crushing, and grinding processes are to be taken into account. All these factors must be balanced with cost.
Figure 3-2: Cascading constraints and goals

Figure 3-3 shows the numerous controllable variables that can be manipulated to achieve the optimum design. The uncontrollable variables and output requirements should be taken into account in the design. This focus of design assumes that the optimization is to be taken at the blasting stage, however, it should be noted that changes in downstream processes may reduce some of the cost upstream where an expensive aspect of the design can be removed.
Figure 3-3: Controllable and uncontrollable input variables and output goals.

Idealized fragmentation curves

Some 30 years ago, MacKenzie presented his now classic conceptual curves showing the cost dependence of the different mining unit operations on the degree of fragmentation. They are presented in their original form in Error! Reference source not found. to Figure 3-8. As can be seen some of the costs decrease with increasing fragmentation while others increase. By adding the curves together one obtains the overall cost versus degree-of-fragmentation curve presented in Figure 3-8. It has the form of a saddle indicating that there is a certain degree of fragmentation for which the overall cost is a minimum. In the particular case shown, the base of the
saddle is quite broad suggesting that the overall costs change little over a wide fragmentation range. Before discussing the development and application of these curves it is important to understand the logic behind them. Beginning with the loading, hauling and crushing curves the logic, as presented by MacKenzie, is as follows:

**Figure 3-4: Loading cost curve**

**Loading**

An increase in the degree of fragmentation will give the shovel a higher rate of productivity. At standard operating costs per hour (for all practical purposes independent of the production rate) this will result in lower costs per ton or cubic yard moved. The effect of wear and tear will also decrease, giving lower operating cost per hour.

**Figure 3-5: Hauling Cost Curve**

**Hauling**

Under similar conditions of haul, lift, size and type of truck and haul road conditions, truck production per hour will increase with greater degree of fragmentation due to faster shovel loading rates and a decrease in bridging (and hence waiting time) at the crusher. There will be a consequent decrease in cycle time. At a standard operating cost per hour, this increase in truck speed or productivity will result in lower unit operating costs.

**Figure 3-6: Crushing Cost Curve**

**Crushing**

An increase in the degree of fragmentation gives lower crushing costs as more material passes through as undersize. Liner costs, repair and maintenance, and bridging time will decrease and the crushing rate per hour will increase. As indicated decreased bridging time also cuts down on truck delay time at the crusher which in turn gives higher truck and shovel...
productivity. Any increase in degree of fragmentation means less work for the crusher. The % bridging time is one indicator, along with shovel loading rate of this degree of fragmentation.

These have been the easiest to explain since the unit costs always decrease with increasing fragmentation. The same is not true for the drilling and blasting costs. There are many possible combinations which can occur depending upon the particular design.

![Figure 3-7: Drilling and Blasting Cost Curves](image)

**Drilling & Blasting**

For a given rock type, geologic structure, and firing sequence, an increase in the degree of fragmentation may be achieved by:

1. increasing the consumed quantity of a given explosive
2. changing to an explosive having greater energy content per unit hole volume (higher energy content! density)
3. combinations of both.

For blasting case (1.) the associated drilling cost would increase if the explosive quantity were to be increased by simply drilling the same diameter drill holes but on a tighter pattern. Thus there would be more drill holes required to blast a given volume. If larger diameter drill holes were substituted and the increased hole volume (explosive quantity) achieved in this way then the rate of increase or decrease would depend upon the comparative drilling cost per foot of hole. For case (2.), presuming that the same hole diameter and pattern is used, the drilling costs would remain constant independent of the fragmentation. For case (3.) the drilling cost could: remain constant, increase, or decrease depending upon the situation. If the same fragmentation is desired and a more energetic explosive is substituted for the one currently in use, then the unit drilling cost could decrease due to the possibility of increasing the hole spacing (spreading the pattern).

In his original presentation MacKenzie has explained the drilling dependence as follows:
Generally speaking, for a given type of drilling and of explosive, the cost per cubic yard or ton will remain constant or increase with the degree of fragmentation. If higher energy explosives are substituted, the drilling cost per yard will decrease. The rate of increase or decrease will be dependent upon the drilling cost per foot.

Therefore summing all cost curves together, the overall optimum fragmentation can be assumed to be the lower section of the saddle.

Figure 3-8: Overall cost curve.

**Preliminary guidelines for blast layout**

The preliminary guidelines for blast layout were taken directly from the Hustrulid (1999) which in turn were taken directly from Ash (1967). These preliminary guidelines provide the reasoning (proofs) for the five key relationships for blast design. The readings prescribed in the additional readings provide additional theory and equations. Figure 3-9 shows the basic nomenclature for the key variables discussed here. Note that the basic guidelines shown here apply to surface mines. However, the same key variables are also applicable to underground mine blasts, as will be shown in future modules.

Figure 3-9: Isometric view showing nomenclature

The key variables for blast design include:
drilled burden (B) - is defined as the distance between the individual rows of holes. It is also used to describe the distance from the front row of holes to the free face. When the bench face is not vertical the burden on this front row of holes varies from crest to toe.

spacing (S) - is the distance between holes in any given row.

Subgrade (J) - Generally the holes are drilled below the desired final grade. This distance is referred to as the subgrade drilling or simply the sub-drill.

Stemming (T) - A certain length of hole near the collar is left uncharged. This will be referred to as the stemming length (T) whether or not it is left unfilled or filled with drill cuttings/crushed rock.

Bench height (H) – is the vertical height from the toe to the crest.

drilled length (L) - is equal to the bench height plus the sub-drill.

length of the explosive column (L_e) - is equal to the hole length minus the stemming. This column may be divided into sections (decks) containing explosives of various strengths separated by lengths of stemming materials.

Sometimes the explosive strength is varied along the hole, i.e. a higher strength bottom charge with a lower strength column charge. As will be seen in the next section, the different dimensions involved in a blast design are not arbitrary but closely related to one another. The selection of one, for example the hole diameter, fixes within rather strict limits, many of the others.

**Spacing to burden relationship**

As can be seen in Figure 3-10, the hole spacing (S) and burden (B) can be directly related through the following relationship:

\[ S = K_s B \]

Where \( K_s \) is a constant relating spacing to the burden.

![Figure 3-10: Plan view of bench showing first row.](image)

**Burden to hole diameter relationship**

Note Figure 3-10, where it is shown that for a particular burden and spacing, each hole diameter is expected to break a particular volume of rock.

The volume of rock to be broken by A UNIT of hole length is:
A particular amount of energy \( (E_V) \) is required to break the a unit volume of rock. The total energy for a unit of hole length is therefore

\[
E_R = V_R \times E_V = B \times S \times E_V
\]

however, since \( S = K_S B \)

\[
E_R = K_S B^2 E_V
\]

Since \( K_S \) and \( E_V \) are constants, the required amount of explosive energy is directly proportional to \( B^2 \).

\[
E_R \propto B^2
\]

The amount of explosive energy available is determined by the explosive volume that can be loaded into that unit of length of borehole:

\[
V_e = \frac{\pi}{4} D_e^2
\]

Where \( D_e \) is explosive diameter and \( E_e \) is the explosive bulk strength (strength by unit volume). Therefore the explosive available is determined using:

\[
E_A = \frac{\pi}{4} D_e^2 E_e
\]

Since \( E_e \) is a constant related to explosive type, the \( D_e \) can be seen as directly proportional to \( E_e \)

\[
E_A \propto D_e^2
\]

If using packaged explosives, as is sometimes the case at pit perimeters where perimeter blasting techniques are employed, the charge diameter \( (D_e) \) may be less than the diameter \( (D) \) of the hole. However, where bulk blasting agents are used, the entire cross-sectional area of the hole is filled with explosive. Therefore hole diameter \( (D) \) is equal to explosive diameter \( (D_e) \). This would make the explosives available also directly proportional to hole diameter:

\[
E_A \propto D^2
\]
Furthermore, this would also require that the required explosives and available explosives to be equal since additional explosives cannot be added beyond the hole diameter.

\[ E_A = E_R \]

however, remember that:

\[ E_R \propto B^2 \quad \text{therefore} \quad D^2 \propto B^2 \quad \text{therefore} \quad D \propto B \]

therefore the diameter is can be determined from a proportionality constant \( K_B \), relating to the hole diameter:

\[ B = K_B D \]

Therefore as the burden increases, the diameter should also increase proportionally, as seen in Figure 3-11.

![Figure 3-11: Effect of hole diameter on burden](image)

**Subdrill to burden relationship**

The toe region is a highly confined volume. Therefore extra explosive energy must be applied to assure adequate fragmentation. This extra explosive power is generally provided by extending the drill hole below the toe elevation and filling the so-called subdrill length \( J \) with explosive. There are several different rationales used for selecting the appropriate length. Here an explanation based upon explosive run-up distance will be presented. The results are essentially the same with all techniques.
Figure 3-12: Toe confinement

There is a certain distance (called the run-up distance) characteristic of the initiating system/explosive which the shock wave must travel away from the point of initiation before steady state conditions are reached in the explosive column. To break the confined toe, the borehole pressure should be as high as possible. As seen in previous classes, the explosion (borehole wall) pressure ($P_e$) is proportional to the square of the detonation velocity:

$$P_e \propto VOD^2$$

The elevation in the hole at which steady state velocity is reached should not be higher than the bench toe elevation. To be conservative the minimum run-up distance will be assumed to be 6D.

Figure 3-13: Run-up distance to achieve steady state VOD.

In addition, the primer is seldom placed directly at the bottom of the blasthole due to the presence of cuttings and water. A normal offset is of the order of 2D. Therefore, the distance from the drilled end of the hole to the toe elevation (the subdrill distance $J$) should be:

$$J \approx 8D \quad \text{therefore since} \quad D \propto B$$

$$J \propto B \quad \text{resulting in a proportionality relationship} \quad J = K_f B$$
Stemming to burden relationship

Near the hole collar, the rise of the explosive should be controlled so that the possibility of breaking upward toward the horizontal free surface should be 'as difficult' or more difficult than breaking, as desired, toward the vertical free face. This can be seen conceptually, by the placement of a spherical charge having the same distance from the collar as the burden. Therefore the “as or more difficult” relationship can be summarized as:

\[ V \geq B \]

![Figure 3-14: Section view comparing the spherical charge (a) and cylindrical charge (b) minimum distance from the collar](image)

An initial assumption is made in that the degree of equivalence of the charges will depend on the proximity to the charge and that this relationship is linear and expressed as:

\[ T_c = K_{TC} B \]

From Figure 3-14, the following relationship is evident:

\[ T = V - \frac{T_c}{2} \]

considering the "as difficult" relationship where \( V = B \) and considering how \( T_c = K_{TC} B \):

\[ T = B - \frac{K_{TC} B}{2} = B \left( 1 - \frac{K_{TC}}{2} \right) \]

which can be simplified to

\[ T = K_T B \]

where \( K_T = 1 - \frac{K_{TC}}{2} \)

Determining \( K_{TC} \) is difficult. The other relationships are derived in a later section however, this relationship proof is explained here. In bench blasting, Langefors & Kihlstrom (1963) have empirically derived the
spherical/cylindrical charge equivalence is as shown in Figure 3-16. To explain the significance of the curve, consider a bench containing two vertical side-by-side blastholes. The burden is the same for both. Rather than discussing the collar region which is the subject of this portion, this example will involve the toe region. The reason for this is that the explanation is easier and the principle is the same. Consider a spherical charge of quantity $Q_o$ placed at the toe elevation in one of the holes. In the second blasthole a cylindrical charge with a linear charge concentration of 1 kg/m of hole is emplaced. The bottom of the charge is at toe elevation and then the column extends upward towards the collar, as seen in Figure 3-15.

![Figure 3-15: View facing bench comparing equivalent cylindrical and spherical charges.](image)

The length of the elongated charge is expressed in multiples of the burden $B$. For a cylindrical charge of length $B$, the total charge would be $B \times 1$ (remember that the explosive volume per length emplaced is 1 kg/m). From the Figure 3-16 one can see that at the toe this elongated charge has only the equivalent breaking power of a spherical charge of weight $0.6 \times 1 \times B$. This is understandable since the energy contained in that part of the elongated charge near the collar must travel a much longer distance to reach the toe and in the process the energy is spread over a much larger volume of rock. The energy density by the time it reaches the toe is much less than that produced by energy which has traveled a shorter distance.

![Figure 3-16: Langefors & Kihlstrom’s Toe breaking equivalence of spherical and cylindrical charges](image)
For a linear charge of length 0.3B the total charge has a mass of 0.3 x 1 x B. From the curve it is seen that this has the same effect at the toe as a spherical charge placed directly at the toe elevation with a mass 0.3 x 1 x B. For charges shorter than 0.3B this relationship holds as well, i.e. the elongated charge of a given weight has the same effect at the toe as a spherical charge of the same weight. For elongated charges with lengths greater than 0.3B, the effect at the toe diminishes rapidly with increasing length. The same effect could be achieved by considering the elongated charge extending from the toe elevation downward. Thus an elongated charge extending from 0.3B below the toe to 0.3B above the toe elevation (for a total explosive weight of 0.6 x 1 x B) would, according to the curve have the same breaking capacity as a spherical charge with a weight of 0.6 x 1 x B placed directly at the toe elevation. This can be seen in Figure 3-17.

![Figure 3-17: Equivalent spherical and cylindrical charges](image)

In transferring this concept to the collar region one finds that

\[ T_C \leq 0.6B \] and noting that \( T_C = K_{TC}B \)

\[ K_{TC} \leq 0.6 \] and where \( K_T = 1 - \frac{K_{TC}}{2}, \)

\[ K_T = 1 - \frac{0.6}{2} = 1.4 \frac{1}{2} = 0.7 \]

**Bench height to Burden relationship**

To this point in the discussion there has been no specific mention of the bench height. If one continues to increase the scale (hole diameter) as shown in Figure 3-18, the center of the charge progresses further and further down the hole. The limiting condition is when the center of charge reaches the toe elevation (Figure 3-19). This occurs for a hole diameter which yields a burden just equal to the bench height. The fifth and last of the fundamental relationships is:

\[ H = K_H B \]

where \( K_H \) is a constant relating bench height to the burden. The value of KH is therefore:
As a rule of thumb, to derive a method of limiting the choice of hole diameter, note how:

\[ B = K_B D \text{ therefore } H = K_H K_B D \] and since \( K_H \geq 1 \), we can derive \( H \geq K_B D \) resulting in a limiting

\[ D \leq \frac{H}{K_B} \]

![Figure 3-18: Burden to diameter relationship.](image)

![Figure 3-19: Limiting the charge diameter and burden.](image)

**Ratios for initial design**

In the previous section, five relationships were derived for preliminary blast design. This section will discuss the general values of the constants which is used as base designs from which to further optimize based on other design constraints. Table 3-1 provides the summary of this section. The remainder provides the reasoning and proof behind these initial values.

**Ratio \( K_s \)**

As will be covered in a later lecture, the optimum burden and spacing ratio depends on the energy coverage of the bench (among other variables). When using a square pattern, the best energy coverage is with \( K_s=1 \) however, empirically, there is little difference between \( K_s=1 \) to \( K_s=1.5 \). For a staggered drilling pattern, the best energy coverage is with \( K_s = 1.15 \). Note that a staggered pattern provides more uniform energy coverage.
The most critical and important dimension in blasting is that of the burden. There are two requirements necessary to define it properly. To cover all conditions, the burden should be considered as the distance from a charge measured perpendicular to the nearest free face and in the direction in which displacement will most likely occur. Its actual value will depend on a combination of variables including the rock characteristics, the explosive used, etc. But when the rock is completely fragmented and displaced little or not at all, one can assume the critical value has been approached. Usually, an amount slightly less than the critical value is preferred by most blasters.

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

A convenient guide that can be used for estimating the burden, however, is the $K_B$ ratio. Experience shows that when $K_B = 30$, the blaster can usually expect satisfactory results for average field conditions. To provide greater throw, the $K_B$ value could be reduced below 30, and subsequently finer sizing is also expected to result.

Light density explosives, such as field-mixed ANFO mixtures, necessarily require the use of lower $K_B$ ratios (20 to 25), while dense explosives, such as slurries and gelatins, permit the use of $K_B$ near 40. The final value selected should be the result of adjustments made to suit not only the rock and explosive types and densities but also the degree of fragmentation and displacement desired.
To estimate the desired $K_B$ value one should know that densities for explosives are rarely greater than 1.6 or less than 0.8 g/cm$^3$. Also, for most rocks requiring blasting, the density in g/cm$^3$ rarely exceeds 3.2 nor is less than 2.2 with 2.7 far the most common value. Thus, the blaster can, by first approximating the burden at a $K_B$ of 30 make simple estimations toward 20 (or 40) to suit the rock and explosive characteristics, densities for the latter exerting the greater influence. As a rule of thumb consider:

- For light explosives in dense rock use $K_B = 20$,
- For heavy explosives in light rock use $K_B = 40$,
- For light explosives in average rock use $K_B = 25$,
- For heavy explosives in average rock use $K_B = 35$.

**Ratio $K_J$**

The most common value of $K_J$ is 0.3. In certain sedimentary deposits with a parting plane at toe elevation subdrilling may not be required. In very hard toe situations, the subdrilling may be increased over that indicated by using $K_J = 0.3$. However it is probably better to consider using a more energetic explosive. It must be remembered that the subdrill region generally forms the future crest/bench top for the bench below. Unwanted damage done at this stage may have a long and costly life. In addition excessive subdrill results in:

- A waste of drilling and blasting expenditures
- An increase in ground vibrations
- Undesirable shattering of the bench floor. This in turn creates drilling problems, abandoned blastholes and deviations for the bench below.
- It accentuates vertical movement in the blast. This increases the chances for cutoffs (misfires) and overbreak.

**Ratio $K_T$**

The minimum recommended value for $K_T$ for large hole production blasting is $K_T = 0.7$. Some specialists suggest the use of $K_T = 1.0$.

Collar and stemming are sometimes used to express the same thing. However, stemming refers to the filling of blast holes in the collar region with materials such as drill cuttings to confine the explosive gases. But stemming and the amount of collar, the latter being the unloaded portion of a blasthole, perform other functions in addition to confining gases.

Since an energy wave will travel much faster in solid rock than in the less dense unconsolidated stemming material, stressing will occur much earlier in the solid material than compaction of the stemming material could be accomplished. Thus the amount of collar that is left, whether or not stemming is used, determines the degree of stress balance in the region. The use of stemming material then assists in confining the gases by a delayed
action that should be long enough in time duration to permit their performing the necessary work before rock movement and stemming ejection can occur. For stress balance in bench-blasting of massive material, the value of $T$ should equal the $B$ dimension.

Placing the charge too close to the collar can result in backbreak, flyrock and early release of the explosive gases with resulting poor fragmentation. On the other hand, increasing the length of stemming may reduce the energy concentration in the collar region to the point where large boulders result.

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

**Ratio $K_H$**

Currently most open pit operations have $K_H$ values which are approximately 1.6 or more. Too small a $K_H$ value will result in substantial cratering.

**Summary of Ratios**

The table below simply summarizes some of the basic ratio values and equations.

**Table 3-1: Ratio Summary**

<table>
<thead>
<tr>
<th>Name of Relationship</th>
<th>Equation</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Spacing - Burden</td>
<td>$S = K_S B$</td>
<td>1-1.5</td>
</tr>
<tr>
<td>Burden – Diameter</td>
<td>$B = K_B D$</td>
<td>$\approx 25$</td>
</tr>
<tr>
<td>Subdrill – Burden</td>
<td>$J = K_J B$</td>
<td>$=0.3$</td>
</tr>
<tr>
<td>Stemming – Burden</td>
<td>$T = K_T B$</td>
<td>$\geq 0.7$</td>
</tr>
<tr>
<td>Bench height - Burden</td>
<td>$H = K_H B$</td>
<td>$\geq 1.6$</td>
</tr>
</tbody>
</table>

**Powder Factor**

The following example is provided to illustrate the relationships developed in the last two sub-sections. A key indicator of blast design is the specific charge, also known colloquially as the ‘powder factor’. Design a blast considering these initial variables:

- Rock = syenite porphyry ($SG=2.6$)
- Explosive = ANFO ($\rho=0.8$, $S_{ANFO}=1$)
- Bench Height ($H$) = 15m
Hole Diameter (D)=381mm (15")
Staggered drilling pattern, vertical holes
1 blast = 4 rows of holes each containing 6 holes

from the relationship, we see:
- \( B = K_B D \), where \( K_B \approx 25 \) (assumed), therefore \( B = (25)(0.381) = 9.5 \) m
- \( S = K_S B \), where \( K_S = 1.15 \) (staggered), therefore \( S = (1.15)(9.5) = 11 \) m
- \( J = K_J B \), where \( K_J = 0.3 \), therefore \( J = (0.3)(9.5) = 3 \) m
- \( T = K_T B \), where \( K_T = 0.7 \), therefore \( T = (0.7)(9.5) = 6.5 \) m
- \( L = H + J \), hole length therefore equals to \( 15 + 3 = 18 \) m

As an assurance, in verifying \( K_H \), we find \( K_H = \frac{H}{B} = \frac{15}{9.5} = 1.6 \), considering the value is usually equal to or greater than 1.6, we find these parameters to be acceptable. To calculate the powder factor, the volume and weight of the explosive used is calculated:

\[
V_e = \frac{\pi}{4} D^2 (L - T) = \frac{\pi}{4} (3.81)^2 (18 - 6.5) = 1.31 \text{ m}^3
\]
\[
W_e = V_e \rho = 1.31 \text{ m}^3 \times 800 \text{ kg/m}^3 = 1049 \text{ kg per hole}
\]

total weight of explosive \( T_{\text{EXP}} = 1049 \text{ kg} \times 6 \times 4 = 25,176 \text{ kg} \)

The weight of the rock that will be broken is calculated:
\( T_R = V_R \times \rho = \text{number of holes} \times B \times S \times H \times \rho = 24(9.5)(11)(15)(2.6 \text{ t/m}^3) = 97,812 \text{ tons} \)

Hence, the powder factor using the explosive ANFO is found to be:
\[
PF_{\text{ANFO}} = \frac{T_{\text{EXP}}}{T_R} = \frac{25,716}{97,812} = 0.26 \text{ kg/ton}
\]

**Determination of \( K_B \)**

As seen in the above discussion, \( K_B \) is by far the most important constants in the design of blasts. Selecting the proper burden is therefore a key step in blast design, and the factor \( K_B \) allows the selection of an appropriate burden and diameter as seen in the equation:

\( B = K_B D \)

As mentioned above, the best initial estimate for \( K_B \) is 25, when using ANFO in rocks with an average density. However, consider the requirements for selecting a new \( K_B \) when explosives or rock type varies. The approach described below can be used as an approximation until field results are available to guide the designed toward more optimal solutions. Note that the
proof below is valid for the metric system. Consider this blast design has must be designed to other variables, including:

\( SG_E = \) specific gravity of explosive used
\( SG_R = \) specific gravity of the rock
\( PF_{\text{EXP}} = \) powder factor (kg/ton)
\( TF = \) tonnage factor (m\(^3\)/ton)

As seen in a previous example, the total weight that one borehole is expected to break can be calculated using the following:

\[
T_R = V_R \times \rho = B \times S \times H \times \rho_{\text{rock}} = B^3 K_S K_H SG_R
\]

Once the total amount of rock material to be removed is known, the amount of energy required is calculated using the powder factor:

\[
E_{\text{Requred}} = T_R \times PF_{\text{EXP}} = B^3 K_S K_H SG_R PF_{\text{EXP}}
\]

The amount of explosive available is defined by the size of the borehole, which can be given as:

\[
E_{\text{Avail}} = \frac{\pi}{4} D_e^2 (H + J - T) SG_E = \frac{\pi}{4} D_e^2 (BK_H + BK_J - BK_T) SG_E = B \frac{\pi}{4} D_e^2 (K_H + K_J - K_T) SG_E
\]

Setting the amount of explosives to the amount required results in:

\[
B^3 K_S K_H SG_R PF_{\text{EXP}} = B \frac{\pi}{4} D_e^2 (K_H + K_J - K_T) SG_E
\]

\[
B = D_e \left[ \frac{\pi}{4} \left( \frac{SG_E}{SG_R PF_{\text{EXP}}} \right) \left( \frac{K_H + K_J - K_T}{K_S K_H} \right) \right], \quad \text{and since } B = K_B D \text{ and assuming a bulk agent is being used,}
\]

\[
K_B = \left[ \frac{\pi}{4} \left( \frac{SG_E}{SG_R PF_{\text{EXP}}} \right) \left( \frac{K_H + K_J - K_T}{K_S K_H} \right) \right]
\]

The powder factor based on the explosive used is replaced with the equivalent ANFO powder factor, denoted by the variable \( PF_{\text{ANFO}} \):
The above formula can be used for several purposes, as will be seen in the following examples.

**Effects to pattern design in changing explosives**

One of the major ways the equation can be used is to study the effect of changes in the explosive on the blasting pattern while keeping other factors constant. Note that the bench height, \( K_H \), depends on the burden which also depends on \( K_B \). Therefore the burden can change. The approach can be seen as follows:

**Explosive 1:**

\[
K_{B1} = \sqrt{\frac{\pi}{4} \left( \frac{SG_{E1}}{SG_R} \right) \left( \frac{S_{ANFO}}{PF_{ANFO}} \right) \left( \frac{K_{H1} + K_J - K_T}{K_S K_{H1}} \right)}
\]

**Explosive 2:**

\[
K_{B2} = \sqrt{\frac{\pi}{4} \left( \frac{SG_{E2}}{SG_R} \right) \left( \frac{S_{ANFO}}{PF_{ANFO}} \right) \left( \frac{K_{H2} + K_J - K_T}{K_S K_{H2}} \right)}
\]

In most blast design alternatives, a constant powder factor is maintained. A ratio between the two \( K_B \) values, where the powder factor ratio is eliminated, can be expressed by:

\[
\frac{K_{B2}}{K_{B1}} = \sqrt{\frac{SG_{E2}}{SG_{E1}} \left( \frac{S_{ANFO}(2)}{S_{ANFO}(1)} \right) \left( \frac{K_S K_{H1}(K_{H2} + K_J - K_T)}{K_S K_{H2}(K_{H1} + K_J - K_T)} \right)}
\]

Taking this ratio and for now, ignoring the changes in \( K_H \) with changing burden, the equation is reduced to:
\[ \frac{K_{B2}}{K_{B1}} = \sqrt[2]{\left( \frac{S_{E} \times S_{ANFO}}{S_{E} \times S_{ANFO}} \right)} \]

The above is a first approximation of the \( K_{B2} \) and is equivalent to the square root of the bulk strength ratio for the explosives in question. To refine the \( K_{B2} \) value, an iterative process is used where:

1) Initial value of \( K_{B2} \) is substituted in the formula:
\( B_2 = K_{B2} D_e \)

2) Then the new \( K_{H2} \) is derived from the equation
\[ K_{H2} = \frac{H}{B_2} \]

3) This new value of \( K_{H2} \) is then inserted into the equation
\[ \frac{K_{B2}}{K_{B1}} = \sqrt[2]{\left( \frac{S_{E} \times S_{ANFO}}{S_{E} \times S_{ANFO}} \right) \left( \frac{K_s K_{H1} (K_{H2} + K_J - K_T)}{K_s K_{H2} (K_{H1} + K_J - K_T)} \right)} \]

4) If the resulting \( K_{B2} \) is then compared to the original estimate. If they are the same, the process stops. If not, then the new \( K_{B2} \) is used in step 1 and the process continues, until the value of \( K_{B2} \) converges.

**Effects to pattern design in changing rock types**

The effect to the pattern design when rock types change is very similar to that discussed in the previous subsection. Here, the initial approximation can be made by the ratio:
\[ \frac{K_{B2}}{K_{B1}} = \sqrt[2]{\frac{S_{R1}}{S_{R2}}} \]

Once again, the iteration process involving three equations is undertaken:

1) Initial value of \( K_{B2} \) is substituted in the formula:
\( B_2 = K_{B2} D_e \)

2) Then the new \( K_{H2} \) is derived from the equation
\[ K_{H2} = \frac{H}{B_2} \]

3) This new value of \( K_{H2} \) is then inserted into the equation
\[ \frac{K_{B2}}{K_{B1}} = \sqrt[2]{\left( \frac{S_{R1}}{S_{R2}} \right) \left( \frac{K_s K_{H1} (K_{H2} + K_J - K_T)}{K_s K_{H2} (K_{H1} + K_J - K_T)} \right)} \]
4) If the resulting $K_{B2}$ is then compared to the estimate at the beginning of the iteration. If they are the same, the process stops. If not, then the new $K_{B2}$ is used in step 1 and the process continues, until the values of $K_{B2}$ converge.

**Numerical Examples**

Consider the mine where the current design has the following parameters:
- Hole diameter = 12 ¼ inches
- Bench height = 40 ft
- Burden = 25 ft
- Spacing = 29 ft
- Subdrill = 7 ft
- Stemming = 17 ft
- ANFO: $S_{ANFO} = 1$
- $SG_{ANFO} = 0.82$
- $Q = 912$ cal/gm
- Rock: $SG = 2.65$
- $PF_{ANFO} = 0.5$ lbs/ton

$$K_B = \sqrt[2000]{\pi} \times \frac{4}{4} \left( \frac{SG_E}{SG_R} \right) \left( \frac{S_{ANFO}}{PF_{ANFO}} \right) \left( \frac{K_H + K_J - K_T}{K_S K_H} \right)$$ using imperial units

**Example 1: Changing diameter of holes**

What would the pattern be if changing to 15 inch diameter holes?

The first step in this problem is to derive $K_B$. First, the other ratios need to be derived:
- $K_H = 40/25 = 1.6$
- $K_J = 7/25 = 0.3$
- $K_T = 17/25 = 0.7$
- $K_S = 29/25 = 1.15$

These values are plugged into the following equation:

$$K_B = \sqrt[2000]{\pi} \times \frac{4}{4} \left( \frac{SG_E}{SG_R} \right) \left( \frac{S_{ANFO}}{PF_{ANFO}} \right) \left( \frac{K_H + K_J - K_T}{K_S K_H} \right) = \sqrt[2000]{\pi} \times \frac{4}{4} \left( \frac{0.82}{2.65} \right) \left( \frac{1}{0.5} \right) \left( \frac{(1.6 + 0.3 - 0.7)}{(1.6)(1.15)} \right) = 25.2$$

This is about what is expected from the recommended ranges discussed previously. For the 15 inch diameter holes, the first approximation for the burden would be:

$$B = K_B D_E = 25.2 \left( \frac{15 \text{ inches}}{12 \text{ inches/foot}} \right) = 31.5 \text{ ft}$$
However, this new value for the burden would result in a change in $K_H$. Therefore the new $K_H$ is found to be:

$$K_H = \frac{H}{B} = \frac{40}{31.5} = 1.27$$

This new value is then input back into the original $K_B$ equation:

$$K_B = \sqrt{\frac{2000\pi}{4} \left( \frac{SG_E}{SG_R} \right) \frac{S_{ANFO}}{PF_{ANFO}} \left( \frac{K_H + K_J - K_T}{K_S K_H} \right)} = \sqrt{\frac{2000\pi}{4} \left( \frac{0.82}{2.65} \right) \left( \frac{1}{0.5} \right) \left( \frac{1.6 + 0.3 - 0.7}{(1.6)(1.15)} \right)} = 24.1$$

This process is repeated through several iterations until a stable $K_B$ is found, which is:

$$K_B = 24.3$$

This results in a pattern with the following dimensions:

- $B = 30$ ft
- $S = 34.5$ ft
- $T = 21$ ft
- $J = 9$ ft

The resulting powder factor may be slightly different from the original on account of the rounding off, however note that this pattern would result in increased coarseness in fragmentation. To maintain fragmentation, the powder factor would have to be increased.

**Example 2: Changing Explosives**

What would be the change in the pattern in changing the explosive from ANFO to heavy ANFO with the following properties (original hole diameter of 12 ¼ inches):

- $SG = 1.10$
- $Q = 815$ cal/gm

We will need to use the weight strength of this explosive product with respect to ANFO, which is calculated:

$$S_{ANFO} = \frac{815}{912} = 0.89$$

We use the initial estimate, as derived in a previous section, and using the initial $K_B$ value from the previous question:

$$K_{B2} = K_{B1} \sqrt{\left( \frac{SG_E \times S_{ANFO}}{SG_E \times S_{ANFO}} \right)} = (25.2) \sqrt{\left( \frac{1.10 \times 0.89}{0.82 \times 1.00} \right)} = 27.5$$
The new burden would therefore be:

\[ B_2 = K_{B2}D_E = 27.5 \left( \frac{12.25 \text{ inches}}{12 \text{ inches/foot}} \right) = 28.1 \text{ ft} \]

The new \( K_B \) is then calculated to be:

\[ K_{B2} = \frac{H}{B_2} = \frac{40}{28.1} = 1.42 \]

This value is then substituted into the equation:

\[ K_{B2} = K_{B1} \sqrt{\left( \frac{SG_E \times S_{ANFO}}{SG_E \times S_{ANFO}} \right)^2 \left( K_{S1} K_{H1} (K_{H2}^2 + K_J - K_T) \right) \left( K_{H2} + K_J - K_T \right) \left( K_{H2} + K_J - K_T \right)} \]

\[ K_{B2} = 27.0 \]

The new burden is calculated to be:

\[ B_2 = K_{B2}D_E = 26.95 \left( \frac{12.25 \text{ inches}}{12 \text{ inches/foot}} \right) = 27.51 \text{ ft} \]

This process is repeated until a stable value of \( K_{B2} \) results, which for this example, the final \( K_{B2} = 27.0 \)

The blast pattern would therefore be (in feet):

\[ B = 27.6 \]
\[ S = 31.7 \]
\[ J = 8.3 \]
\[ T = 19.3 \]

**Stemming & Decking**

Many blast designs use decked charges formed by dividing the explosive column into two or more individual charges, initiated on the same or different delays, separated by inert stemming material. Decking is employed to:

- Reduce explosive use adjacent to weak rock zones, faults or clay seams
• Reduce charge quantity detonated at one time delay, lowering ground vibrations
• Bring the powder column up higher in the hole to assure good breakage near the collar

Decked charges should be separated by stemming materials at a length beyond which two adjacent decks do not affect one another. If interdeck stemming is too small, the deck designed to initiate on the earlier time delay may prematurely initiate the second deck. This situation is referred to as sympathetic detonation and may lead to excessively high ground vibrations or flyrock and a loss of fragmentation due to confinement from improper timing. A rule of thumb for the design of interdeck stem length is to employ the hole radius dimension in feet. The following example gives the design procedures for a blast design in which the explosive charge is limited to control ground vibrations.

**Example of Deck design**

Determine the change in layout if decking is needed in the layout by limiting the weight of explosive to 275 kg/delay, keeping the same powder factor. Note that the collar stem, bench height, and subgrade drilling remain unchanged. The basic data is:

- Rock = syenite porphyry (SG=2.6)
- Explosive = ANFO ($\rho=0.8$, $S_{ANFO}=1$)
- Bench Height ($H$)=15m
- Hole Diameter ($D$)=381mm (15”)
- Staggered drilling pattern, vertical holes

from the relationship, we see:

- $B = K_B D$, where $K_B \approx 25$ (assumed), therefore $B=(25)(0.381)=9.5$ m
- $S = K_S B$, where $K_S = 1.15$ (staggered), therefore $S=(1.15)(9.5)=11$ m
- $J = K_J B$, where $K_J = 0.3$, therefore $J=(0.3)(9.5)=3$ m
- $T = K_T B$, where $K_T = 0.7$, therefore $T=(0.7)(9.5)=6.5$ m
- $L=H+J$, hole length therefore equals to $15+3=18$ m
- A following calculations from section 0, above, we see that the powder factor is 0.26 kg/ton and that the amount of explosives per hole is: 1049 kg.

Solution:

The amount of explosives per unit length is calculated as:

$$\frac{W_{exp,lo}}{L} = \frac{\pi}{4} D^2 \rho_{exp,lo} = \frac{\pi}{4} (0.381)^2 (0.8) = 91 \text{ kg/per unit length}$$

Charge decks per hole is found to be: $275 \text{ kg} / 91 \text{ kg} = 3$ meters per deck. Therefore assuming if three decks are used, the overall explosive column
length is 9 meters leaving 1.5 meters of stemming length available meaning 0.75 meters of stemming between each deck.

\[ PF = \frac{(n \text{ decks})(W \text{ per deck})}{B \times K_s B \times H \times \rho_{rock}} = \frac{(3)(275 \text{ kg})}{B \times (1.15)B \times 15 \times (2.6 \text{ ton} / \text{m}^3)} = 0.26 \text{ kg/ton} \]

solving for B, the following is determined:

\[ 11.7B^2 (\text{kg} / \text{m}^2) = 825 (\text{kg}) \quad \text{therefore } B = \sqrt{\frac{825(\text{kg})}{11.7(\text{kg} / \text{m}^2)}} = 8.4 \text{m} \]

therefore the new \( S = (1.15)(8.4) = 9.7 \text{m} \)

**Concluding Notes on D, B, and PF**

Drillhole diameter, burden, and powder factor are the most important variables in blast design. Although this lecture provided the preliminary blast design layout equations, be aware that blasting is an never-ending process of fine-tuning and modifications. This approach is necessary due to the many factors that cannot be controlled such as geology and explosive loading conditions. Burden should be selected based on geology and explosive energy output. Hole diameter is usually set by the drill rig capacity which is matched to the range of hole depths anticipated for the job. It is desirable to select a size that will provide an adequate powder factor (the ratio of explosive distributing the explosive evenly throughout the hole depth). Fragmentation and particle size distribution are a function of hole diameter and burden. The capacity of the excavation equipment or requirements of downstream processing dictates the required fragmentation. Higher powder factors result in fine fragmentation and are therefore required for small capacity removal equipment such as a front-end loader. Higher powder factors result in coarser fragmentation and are typically used for rock removal using draglines and large shovels. Note that since the powder factors are lower, the drillhole diameters and burden are typically lower for quarries or mines using smaller equipment in weaker rock.

**Table 3-2: Powder Factor**

<table>
<thead>
<tr>
<th>Excavation Method</th>
<th>Range in Powder Factors, lb/yd³</th>
</tr>
</thead>
<tbody>
<tr>
<td>Surface metal mining</td>
<td>0.6–1.0</td>
</tr>
<tr>
<td>Surface coal mining</td>
<td>0.5–0.7</td>
</tr>
<tr>
<td>60 yd³ dragline</td>
<td>0.5–0.7</td>
</tr>
<tr>
<td>30 yd³ shovel</td>
<td>0.6–1.1</td>
</tr>
<tr>
<td>17 yd³ front-end loader</td>
<td>0.6–1.6</td>
</tr>
<tr>
<td>Coal mining blast casting</td>
<td>0.8–1.5</td>
</tr>
<tr>
<td>Quarrying</td>
<td>0.8–1.5</td>
</tr>
<tr>
<td>Construction</td>
<td>0.25–0.8</td>
</tr>
<tr>
<td>open excavations</td>
<td>2.0–3.0</td>
</tr>
<tr>
<td>trenching</td>
<td></td>
</tr>
</tbody>
</table>

Conversion factor: 1 yd³ = 0.7646 m³, 1 lb/yd³ = 0.5933 kg/m³.
ADDITIONAL RESOURCES IX

Additional learning resources include:

- Assignment 5 (you will learn much when doing this assignment.
- Lecture Module 4.1 – Basics of Blast Design

Geological Impacts on Blast Design

Prior to developing each blast design, care should be taken to review existing geology and geologic structure. This can be accomplished by simply review existing outcrops in, around, or adjacent to the blast site. A detailed drill log indicating discontinuities at various depths, may also be used.

Rock Properties

Blasting performance is usually influenced more by rock properties than by the properties of the explosive. Rocks show numerous planes of weakness, natural fissures, and cracks formed as a result of previous blasting. Thus there are planes of preferential fracture orientation in anyone or more of an infinite number of directions relative to the blasthole's axis. The following properties of rock may have a significant influence on blasting results.

Dynamic Compressive Strength

If the explosive’s outgoing strain wave exceeds the dynamic compressive breaking strength of rock an annulus of crushed rock is formed around the charge. This crushed zone is detrimental to the transmission of strain waves in the surrounding rock. Hence it is important that the dynamic compressive strength of the rocks in situ be determined, and an explosive of low density and velocity should be used in rocks with low compressive strength values.

Elastic Moduli

The elastic moduli give the behavior of rocks under stress and should be determined by sonic techniques (dynamic moduli) rather than by the use of mechanical tests (static moduli). It has been found that the explosion pressure should not exceed 5% of the dynamic Young's modulus in order to obtain optimum results from blasting. The post-detonation gas pressure exerted in the cracks between the blasthole and the free face pushes the burden forward and produces heave. The bulk modulus of a rock has to be known in order to calculate heave.

Density

The density of rock is closely correlated with its strength. An increase in rock density often results in a decrease in the displacement of a rockmass.
fragmented by blasting. Adequate displacement of higher-density rock can be achieved by following one of three courses:

- increasing the blasthole diameter,
- reducing the blasthole pattern, or
- changing to an explosive which has stronger heave energy.

**Porosity**

Porosity tends to reduce the efficiency of blasting operations. The lengths of strain-wave-induced cracks in a highly porous rock are calculated to be only about 25% of those in a non-porous rock of identical mineralogy. This implies that highly porous rocks are fragmented mainly by heave energy. Hence, post-detonated gases have to be kept trapped at high pressure until they have performed their task. This can be achieved by bottom priming and by having adequate stemming to prevent premature venting of gases.

**Internal Friction**

Internal friction is a relative measure of a rock’s ability to attenuate strain waves by the conversion of some of the mechanical energy into heat. It increases with a high degree of porosity, permeability, and jointing of the rock mass. Generally, internal friction values for igneous or metamorphic rocks are lower than for sedimentary rocks, which require high energy explosives for satisfactory blasting. However, if the rock pores are filled with water, the internal friction factor reduces considerably, giving easier passage of the strain wave and improved fragmentation.

**Water Content**

Water saturation considerably increases the velocity of propagation of strain waves, owing to the filling of pores with water, which is a good medium for elastic wave transmission. However, fluids in a porous rock reduce both the compressive and tensile strengths, owing to the lower friction characteristic between grain surfaces. If water is present in discontinuities adjoining a block of rock which is being blasted, strain waves may have a greater ability to weaken that rock mass by means of water being jetted considerable distances through interconnected fissures. This has a wedging action which will have a considerable influence on overbreak; and hence slope instability. It is therefore advisable in open-pit mining to dewater a rock mass where a 'permanent' slope, which is intended to remain for several years with minimum maintenance, has to be formed.

**In Situ Static Stress**

High in situ static stresses often exist well within the rock body and blasting results can be affected by these stresses. A typical example of such a phenomenon is where some of the radial cracks from the blasthole tend to curve off into the direction of the static stress field. There is also a strong possibility of the closing of microcracks by the static stress in the rock mass when the confining pressure is above 100-300 MPa. Again, when a stress field exists in a direction normal to pre-existing radial cracks around a
blasthole, it can be sufficiently strong to prevent extension of these cracks. Moreover, it may induce the formation of new cracks in the direction of the stress field. These types of unexpected blast results are often due to in situ stresses in the rock mass.

**Structure**

The bedding planes and joints in a rock mass tend to dominate the nature of the blast-induced fracture pattern. Maximum fragmentation is generally achieved where the principal joint planes are parallel to the free face. Where the angle between joint planes and the face is within a region between 30° and 60°, the blastholes may produce an irregular new face, owing to the formation of wide cracks behind the blastholes. When the joint planes are at right angles to the face, each block requires at least one blasthole in order to obtain satisfactory fragmentation (Figure 3-22).

![Figure 3-22: Blasthole placement in relation to joint planes.](image)

If there is a high density of joint planes normal to the face, it is worth while to consider adopting smaller diameter blastholes at closer spacings. Pronounced bedding planes can also inhibit the explosive's strain energy transmission from one stratum to the next. This has particular importance where the stratum near the collar of the blasthole is hard and is filled with a prescribed length of stemming material to avoid fly-rock or early escape of the explosive's gas energy. This situation can be dealt with in two ways. The first is to place a small 'pocket' charge centrally within the stemming column (Figure 3-23). The second method is to drill 'stab holes' half-way between the drilled burden and spacing, and lightly charge them although this may not be practical from an operational standpoint.
The physical distance between the structures and how that would affect blasting should be considered. Pre existing cracks in this area will direct or even dictate the fragmentation size. The three most negative geological effects (structurally related) on blast performance include:

- Rock that has open structures
- Zones of incompetence within the rock, in which structures are unpredictable
- Rock with alternate zone of competent and incompetent rock.

Close and tight rock structure are preferable as the explosive energy is not lost or vented. The problems arise when the energy is not confined or when the transmission of stress waves within the rock mass are interrupted. Open or widely separated structures can result in poor fragmentation due to:

- **Interruption of the explosive generated stress waves.** Causes an inconsistent formation of cracks
- **Disruption of confinement resulting in oversize.** Venting and airblast can also occur in weak seams or open layers of rock. This can be corrected through stemming and decking. These areas can be identified in the drilling process when the drill experiences slower penetration rates related to poor hole flushing.
Some solutions to addressing these problems include:
- Closer initiation intervals (achieves desired rock breakage before allowing existing cracks to open further)
- Altering design (burden, spacing, and hole diameter)
- Selecting more dense explosives or blasting agents (detonate at higher velocity)
- Use of multiple decks or cartridge explosives smaller in diameter into the stemming zone.

**Structurally induced Interruption of stress waves**

Figure 3-25 shows the type of radial cracking which one might expect when blasting a single hole in a brittle, massive rock formation. There will be a relatively few long cracks (6-8) spaced uniformly around the hole. As one approaches the hole the cracks will be shorter and more numerous.

![Figure 3-25: Idealized radial cracking surrounding a single hole.](image)

The maximum length (Re) of the radial cracks for a given explosive and rock type can be shown to be directly dependent on the hole radius. Thus as the hole diameter is increased from 150 mm to 310 mm the length of the longest cracks would be expected to about double. This is consistent with the design relationship

\[ B = K_b D \]

presented earlier since the burden should be related to the lengths of the cracks generated

\[ B \propto R_c \]

If the strength of the explosive used in a hole of given diameter is increased or decreased, the outer crack radius should change accordingly. This is reflected in the value of \( K_b \) chosen. Since in general, a larger diameter hole is less expensive to drill than one of smaller diameter (on a cost/volume basis) the natural conclusion would be to drill as large diameter holes as possible. Unfortunately fragmentation considerations would suggest just the opposite,
i.e. the holes should be smaller to better distribute the explosive throughout the rock mass. To illustrate this some simple geometric reasons will be given.

Figure 3-26 shows two possible blast patterns using different size holes but the same explosive. The specific energy (powder factor) is the same for both. A simplified representation of the radial cracks after blasting is shown in Figure 3-27 for each pattern. As the hole diameter is increased and the pattern expands, the distance between adjacent crack tips becomes greater. For the case shown

$L > L'$

Thus even though the energy density is the same, the fragmentation is more coarse. Generally as the pattern is spread, the powder factor (energy factor) must be increased to maintain acceptable fragmentation.

![Pattern A](image1)

![Pattern B](image2)

**Figure 3-26: Extent of cracking for two patterns with different hole diameters and same PF**
One way of maintaining proper fragmentation is to increase the powder factor by limiting the pattern spread to some proportion of the theoretical value. As shown in Figure 3-28, there is now an overlap of the longest fractures. Another way of accomplishing this would be to increase the energy of the explosive being used.

Therefore, even in massive rocks, because of the point introduction of energy into the rock and the fracture geometry, there are limiting hole diameters/burdens/spacings which yield acceptable fragmentation. It is well known that an actual rock mass generally contains many discontinuities of different types. If such structures (joints in particular) are now introduced, such as is shown in Figure 3-29, the story becomes even more complex.
Figure 3-29: Effect of jointing on fragmentation

The radius of influence for any given hole is significantly reduced since
- The radial cracks will not cross the gaps formed by the joints.
- The high pressure gases can be short-circuited by the less resistant joints compared with the fresh cracks. Therefore the primary fracturing effectiveness is reduced as well as that produced by a sustained heave of the fractured material.

Although these pre-existing cracks limit the formation of new cracks and provide avenues of escape for the explosive gases, mobilization of these is a major reason why the specific breakage energy in blasting is much lower than other processes which must attack the intact rock.

Figure 3-30 shows two potential drilling patterns in the jointed rock. The smaller diameter, closely spaced holes yield almost one hole per block and the fragmentation would be expected to be good. On the other hand, the larger holes on wide spacings could yield a large number of substantial blocks largely isolated from the effect of the explosive by the joints. Pattern (a) would have higher associated drilling and blasting costs than Pattern (b). By assigning costs to the degree of fragmentation and knowing the overall ideal fragmentation requirements, an 'optimum' pattern can be determined.

Figure 3-30: Possible blast patterns in Jointed rock

Structure Orientation
The orientation of the major structures can have a significant effect on blasting results.
There are three cases to be considered:
- Shooting with the dip
- Shooting against the dip
- Shooting along the strike
In shooting with the dip (Figure 3-31) one finds
- a tendency to get more back break
- less toe problems
- a smoother pit floor
- more movement away from the face and therefore a lower muckpile profile.

Note: less sub-drilling may be required in this case owing to the fact that the explosive energy may follow the strata downward, eliminating toe problems. Furthermore, inclination of the drillholes in the direction of dip may reduce backbreak but will also tend to ‘cast’ (throw) and spread the muckpile. A slight addition of delay time in the back row may provide relief to the back of the shot resulting in a more stable highwall.

![Direction of Blast →](image)

**Figure 3-31: Shooting with the dip**

When shooting against the dip (Figure 3-32) one finds
- less backbreak since the strata is dipping into the wall.
- the toe would be more difficult to pull.
- a rougher floor condition.
- the muckpile may be higher with less movement from the face.

In this situation, the rock tends to move upward, parallel to the plane of the joints. As a result of explosive energies migrating into the strata, a rock unit may cause back break. This situation may result in the creation of an unstable highwall. Also the muckpile would tend to be poorly displaced (creating more work during excavation). If a rock unit is massive, a saw-toothed effect along the floor and overhand at the crest of the slope could result. To eliminate toe problems, the blaster may consider:
- a blast using angled drill holes,
- high energy explosives in the toe area and/or additional subdrilling.
- Decking, satellite holes or small diameter explosive charges in the crest area might help to alleviate an overhang.
- Pre-splitting may also be a highwall stabilization option.
Finally, when shooting along the strike (Figure 3-33) one finds that:

- the floor can be highly sawtoothed due to the different rock types intersecting the floor.
- for the same reasons the back break is irregular.
- these are some of the worst conditions for those involved in drilling and blasting. To overcome this, the working face may be reoriented to a more favorable conditions.

ADDITIONAL RESOURCES

Additional learning resources include:

- Readings, Chapter 17: Rock and rock mass properties and their influence on the results of blasting, from: Jimeno, Carlos Lopez, Emilio Lopez Jimeno, and Francisco Javier Ayala Carcedo. “Drilling and Blasting of Rocks” translated by Yvonne Visser De Ramiro. Rotterdam: A.A. Balkema. (Mandatory for assignment 5) - also considered readings for Module 4.3 – Patterns and sequencing.
- Lecture Module 4.2 – Geological impacts on Blast Design.
**Patterns and Sequencing**

The formations should be examined to identify the strike and dip direction of the most prominent joints. In igneous and metamorphic rock formations, one should consider aligning the rows of holes parallel to the alignment of the dominant joint system. In sedimentary rocks, the drill holes should be placed in rows drilled parallel to the formation strike line.

When considering the option of altering the blasthole pattern by increasing the spacing (parallel to joints) and reducing burdens (perpendicular to joints) or using a staggered pattern, the direction of movement or angle of movement controlled by delay intervals should be considered. A diagram of blast hole location and firing time intervals will assist in predicting the true burden and spacing firing angles relative to existing geologic structure.

Millisecond (MS) delay blasting was introduced in open pit quarry blasting many years ago. Even when blasting to a free face, the rock movement time can be an important factor. This is particularly true in multiple row blasts. For a typical quarry with 15-foot (4.6 m) spacings, the initial movement at the free face may occur in 10 to 12 milliseconds, but the burden only moves about 0.5 foot (15 cm) in 10 milliseconds. With one or two rows of holes, the prime movement is directly out from the face. As the number of rows increases, the rock movement will tend toward the vertical. This is caused by the low velocity of the broken rock successively reducing the relief toward the free face. This can contribute to "tight" bottom as well as flyrock.

![Figure 3-34: Increasing numbers of rows increases upward movement due to lower velocity of previously broken rock.](image)

Delay Blasting techniques are employed to improve fragmentation, control of rock movement, overbreak, and to reduce ground vibrations. Delays are incorporated into the blast design using electric or nonelectric caps or delay connectors with detonating cord. The delay patterns used in design will determine the sequence of hole or deck initiations, thereby, dictate the
overall direction of blasted rock movement and resulting fragmentation. Depending on the S/B ratio, the actual timing (in milliseconds) between detonating charges will determine muck pile displacement height and distance from the bench.

Figure 3-35: Muckpiles illustrating the difference between millisecond timed and instantaneous.

Figure 3-35 illustrates the difference between two adjacent blasts. The blast on the left used millisecond timing and the blast on the right instantaneous firing.

It may also be important to provide additional nomenclature for the various types of blasts in terms of degree of fixation. Figure 3-36 shows a corner blast and face blast.

Figure 3-36: Corner and face blasts.

Effective Burden and Spacing
Depending on initiation sequence, an effective burden $B_e$ and effective spacing $S_e$ result as shown in Figure 3-37. The figure shows a variation of timing used for surface blasting called echelon or half chevron. The effective spacing is the distance between holes in a row defined by adjacent time
delays (e.g., delays by rows). Effective burden is the distance in the direction of resultant rock mass movement.

![Figure 3-37: Echelon or half chevron.](image)

The “V” (Vee) pattern, also known as chevron, (Figure 3-38) is applicable to most types of formations. It can be readily adapted to the square or rectangular pattern. When a V/MS delay pattern is used in conjunction with a square drill pattern, the angle of movement is 45 degrees to the open face. Therefore, a 10 by 100 foot (3 by 3 m) square pattern becomes a rectangular pattern with a 7.07-foot (2.1 m) burden and a 14.14-foot (4.3 m) spacing or the burden is only one-half the spacing. If a rectangular drill pattern is used, the angle of movement will vary in relation to the relative (also known as ‘effective) burden and spacing dimensions. The formula to determine the angle of movement in relation to the open face for a rectangular pattern is:

\[
\tan A = \frac{B}{S}
\]

where \( B \) = burden  
\( S \) = spacing  
\( A \) = angle of movement

Therefore effective spacing \( S_e \) and effective burden \( B_e \) would be calculated by:

\[
S_e = \frac{B}{\sin A} \quad \text{and} \quad B_e = \frac{S}{\sin A}
\]

Example:
Determine the effective burden and spacing for a V initiation pattern with an 8 ft drilled burden and 13 foot spacing on a square drilled pattern.

\[
\arctan \left( \frac{8}{13} \right) = 31.6^\circ
\]
\[
S_e = \frac{8}{\sin(31.6)} = 15.26 \text{ ft}
\]
\[
B_e = 13\times \sin(31.6) = 6.8 \text{ ft}
\]
Designing the Timing

The design of initiation timing for multiple-hole blasting is critical to the blasting effectiveness. If the interhole delay is too short, the movement of row burdens is restricted and fragmentation is poor. High ground vibrations result, and backbreak along the new high wall may persist, jeopardizing the stability of the slope. If interhole delays are too long, cutoffs of surface delays may occur. The minimum time for design is controlled by the stress wave travel distance \((= 2 B_e)\) in order for radial cracking to begin to develop, contributing to the detachment of the rock mass in the vicinity of the hole. This detachment forms an internal free face (or relief) to which successive detonations will interact with the reflection of stress waves. The minimum timing is, therefore:

\[
t = \frac{2B_e}{C_p} \times 10^3
\]

where \(t\) = stress wave travel time in milliseconds (ms)
\(B_e\) = effective burden or distance form the hole to the free face in feet
\(C_p\) is velocity of sound for the rock in fps

The maximum timing is that at which the burden is fully detached and accelerating as gas pressures build. Research has shown that stress wave travel time is a fraction of the time required to develop radial cracks. Furthermore, studies using high-speed photography indicate that the burden moves within a timeframe which is between 2 to 10 times the wave travel.
time to the face. Other research has shown that the time to burden movement ranges from 5 to 50 ms, and suggests an optimum range of timing for design between 1.5 to 2.5 ms/ft of $B_e$.

Timing studies have been performed to investigate resulting fragmentation and muck pile shapes. Reduced-scale research using a variation in delay ratios suggests improved fragmentation for timing between 11 to 17 ms/ft of $B_e$. Even more research has demonstrated improved fragmentation for S/B ratios of two at timing ratios of 1 ms/ft of $B_e$ or greater.

Substantial research has been undertaken in production-scale, multiple-row blasting resulting in recommended timing to improve various aspects of the resulting muckpile:

- For optimum fragmentation, some suggest delays of 1 to 5 ms/ft within rows and 2 to 15 ms/ft of $B_e$ between rows (or on the echelon).
- For optimum breakage and forward movement, from measured flyrock velocity, and gas venting, through the collar stemming, it was established that 3.4 ms/ft of hole spacing and 8.4 ms/ft of $B_e$.
- Forward throw and muckpile shape: similar work in which muck pile profiles were mapped indicates that optimum forward throw and muck pile height reduction occur for delay ratios of 4.2 ms/ft of $S_e$ and 10 ms/ft of $B_e$ while forward throw is minimized, resulting in high muck piles, with ratios of 1.5 to 2 ms/ft of $S_e$ and 5 to 6 ms/ft of $B_e$.
- For single-row production shooting and S/B of 1.2 to 1.6 that timing ratios greater than 1.2 ms/ft of $B_e$ are ideal.
- Rock Types: One researcher recommended 1.2 ms/ft of $B_e$ for multiple-row production blasting in hard rock, while using high powder factors and short stem lengths. A 2.4 ms/ft of $B_e$ was recommended for soft rock with long stem lengths and low powder factors.
- Ground vibrations: to control ground vibrations, it has been recommended that a 1.3 ms/ft of $S_e$ and 1.2 to 4.3 ms/ft of $B_e$ is to be used.

The timing ratios cited are found to vary over a wide range. A great deal of research on the effects of initiation timing cannot be compared due to the lack of similar variables such as geology, scale, and explosive type. Some researchers have recognized the need to qualify delay ratios, in a general way, based on existing fracture density. Competent dense rock requires lower delay ratios to achieve fine fragmentation, while weak fractured rock fragments best with higher delay ratios.

**Generalized Timing**

When taking both pattern shape and timing into account, the most desirable overall drilling/initiation pattern would the drilling pattern with the best energy distribution. Table 3-3 provides the effect of drilling patterns and S/B ratios on the area covered by fracture circles. However, the existence of fractured ground where existing
fracture planes limited the development of new radial cracks would reduce the efficiencies involved in maximizing the energy distribution, as discussed in previous lectures. Overall initiation patterns are best where each blasthole is initiated separately and in sequence. Alternatively, simultaneously initiated blastholes should be far enough apart to prevent mutual interaction between their stress fields. This is usually best achieved through the “chevron” or V patterns.

**Table 3-3: Effect of drilling patterns and S/B ratios on the area covered by fracture circles (energy distribution)**

<table>
<thead>
<tr>
<th>S/B ratio</th>
<th>Square pattern %</th>
<th>Staggered pattern %</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>77</td>
<td>98.5</td>
</tr>
<tr>
<td>1.15</td>
<td>76</td>
<td>100</td>
</tr>
<tr>
<td>1.25</td>
<td>75</td>
<td>99.5</td>
</tr>
<tr>
<td>1.5</td>
<td>71</td>
<td>94.6</td>
</tr>
<tr>
<td>2.0</td>
<td>62</td>
<td>77</td>
</tr>
</tbody>
</table>

Figure 3-39 provides a summary of events and their timing for a single blasthole allowing the reader to visualize the chain of events.

**Figure 3-39: The events and timing in a blasthole**
Skipping a Period
It is a common practice of many blasters to double the delay time on the last row. This provides additional time for the rock ahead of the last row to move forward so that the relief on the last row will be increased. This practice, called "skipping a period", will also reduce the upward ripping action and materially reduce the backbreak on the face. When the blast consists of as many as eight or nine rows, the timing on MS delays should provide the additional time without skipping a period. As shown in Figure 3-40, the No.1 through No.8 periods (25 through 200 MS) provided a nominal 25 milliseconds between each period. No.8 through No.15 (200 MS through 500 MS) provided a nominal 50 milliseconds between each period. No. 15 through No. 19 (500 MS through 1,000 MS) provided a nominal 100 milliseconds between each period. This sequence is provided only as an example for discussion: the actual sequences and intervals of detonator timing vary with manufacturer.

Figure 3-40: Blast timing sequence skipping periods
Always base timing designs on the limitations of detonator accuracy since delays of a given period have a range of actual firing times. Check with the suppliers to avoid overlap or crowding. Even with additional time between rows, the tendency still exists for the rock to stack if the number of rows is excessive. The hole diameter, burden and spacing, and height of face all have a pronounced effect on the number of rows that can be fired successfully without excessive stacking or without encountering high bottom. When the rock is broken, it will occupy on the average 30 percent more volume (swell factor) than it did in the solid. Remember that swell factor will vary with the type of rock.
In most cases the material has only two directions to move, to the front and vertically. Obviously, excessive movement in either direction will result in dangerous flyrock. If the number of rows is excessive, forward movement is limited, thus additional space for forward expansion cannot be provided.

Pattern Types
The next few sections will provide general guidelines for a few blast designs which can be altered by mine planners to meet needs and conditions. Remember that there are three basic drill pattern types into which these patterns can be designed: square; staggered; and rectangular. Note that various initiation systems can be used to achieve the various timing options. The specific numerical dimensions and timing of the figures reflects only suggestions or possible options. The shape, direction of movement and relative timing are what the figures are intended to represent.

“V” (Vee) Pattern
The V pattern (Figure 3-38) is applicable to most types of formations. It can be readily adapted to the square or rectangular pattern. It may be used with a staggered pattern, but is not as practical for ease of loading under field conditions. If a rectangular drill pattern is used, the angle of movement will vary in relation to the relative burden and spacing dimensions. Note that this pattern results in a high concentrated and centralized rockpile. Another advantage may be that the impacts between rock in different rows may additionally fragment the muck. The forward movement is controlled within reasonable limits and the broken muck is deposited in a windrow 90 degrees to the open face. Depending on the formation and the number of holes per row, it is sometimes desirable, as seen in Figure 3-41, to open the blast using two holes on the first delay. This will result in slightly more forward movement in the muckpile. Note that his type of drill pattern also results in an effectively staggered initiation sequence.
Figure 3-41: Chevron or V pattern with double hole initiation for slightly more forward movement.

**Echelon Delay Pattern**

When the blast area is open on two adjacent sides in an external corner, the blast should be designed to take advantage of the reduced degree of fixation. Note that these patterns would result in a lower, flatter muckpile. Other advantages include a minimization of tight toe problems and an easier connect-up (pattern tying) since the paths can be easily seen. Figure 3-42 is an example of an echelon delay pattern and is also known as a half-chevron.

Figure 3-42: Echelon Delay Pattern
**Channel Delay Pattern**

Operations are frequently opened on side hill terrain where there is not sufficient area opened in front of the blast for forward movement associated with the V or echelon patterns. In these cases a channel delay pattern can be used to confine the broken rock to the blast area. The design of the blast must be relatively narrow, in certain formations it may be desirable to skip a delay period to allow additional time for rock movement because only the direction of movement for the initial holes is vertical. The channel pattern will not hold all the broken rock within the blast area if the terrain is steep.

![Channel pattern](image)

**Figure 3-43: Channel pattern**

**Flat-face pattern**

The flat face pattern will move the broken material farther from the face and usually will result in rock of larger size. This pattern should only be used for special conditions. It is almost always used with a staggered pattern unless large blocky material is desired.

![Flat face pattern](image)

**Figure 3-44: Flat face pattern**
**Alternating hole delay pattern**

This pattern has been used with limited success in thinly laminated formations on very wide spacings. It is not recommended for most formations.

![Figure 3-45: Alternating delay pattern](image)

**Sinking Blast Pattern**

When opening a new pit or starting a new lift in an existing pit, it may be necessary to make a sinking blast, also known as a sinking shot or sink. A sinking blast varies from most blasts because there is no open face or relief, and the direction of blasted rock movement must be vertical. Since the entire blast will be ‘in the tight’, which levels of vibrations and the generation of flyrock are more likely. In most formations it is necessary to decrease the burden and spacing of the initial holes in the delay pattern in order to open an area of relief to which the remaining holes may break.

![Figure 3-46: Sinking shots](image)
Sequential Pattern Variant

Note that considering that various initiation products (det. cord, det. cord delays, shock tubes, electric detonators, programmable detonators, etc...) can provide any timing sequence possible, variants of the timing can be created allowing movement planes not directly aligned along drill holes. Note each hole in Figure 3-47 is initiated at different times yet is similar to the movement plane of an echelon blast. Blasting each hole individually may also aid in reducing vibrations from the blast as less explosive material is initiated simultaneously.

Figure 3-47: Sequential Firing

ADDITIONAL RESOURCES XI
Additional learning resources include:
- Lecture Module 4.3 – Patterns and Sequencing
Module 4: Blasting Applications

The section will cover the more advanced aspects of blast design in terms of its various applications.

Drift Design

The blasts in tunnels and drifts are characterized by the initial lack of an available free surface towards which breakage can occur; only the tunnel heading itself. Note that for tunnel rounds that are too large to be drilled in a single pass (as seen in many civil applications), can be developed in multiple stages as seen in Figure 4-1.

8 These notes were assembled directly from the following references:
- From Dr. Paul Lever’s 415 course notes Hartman, Howard L. Ed. SME Mining Engineering Handbook. 2nd ed. 1992
- Stefanko, Robert. Coal Mining Technology Theory and Practice. Littleton CO.: Society of Mining Engineers. 1983
- Mining-Technology.com, search: continuous mining
- Caterpillar Performance Handbook, 28th Ed.
- 2001 Class notes, from Bob Cummings
- The History of Explosives. http://sis.bris.ac.uk/~dj9006/explosives/history.html
The principle behind tunnel blasting is to create an opening by means of a cut and then stoping is carried towards the opening. The opening usually has a surface of 1 to 2 m$^2$, although with large drilling diameters it can reach up to 4 m$^2$. In fan cuts, the cut and cut spreader blastholes usually occupy most of the section. Stoping can be geometrically compared to bench blasting although it requires powder factors that are 4 to 10 times higher. This is due to drilling errors, the demand made by swelling, the absence of hole inclination, the lack of co-operation between adjacent charges and, in some areas, there is a negative action of gravity as happens in lifter holes.
The position of the cut has influence on rock projection, fragmentation and also on the number of blastholes. Of the three positions, corner, lower center and upper center, the latter is usually chosen as it avoids the free fall of the material, the profile of the broken rock is more extended, less compact and better fragmented.

Cuts.
The blasts in tunnels and drifts are much more complex than bench blastings owing to the fact that the only free surface is the tunnel heading. The powder factors are elevated and the charges are highly confined. On the other hand, burdens are small, which requires sufficiently insensitive explosives to avoid sympathetic detonation and at the same have a high enough detonation velocity, above 3000 ms, to prevent channel effects in the cartridge explosives placed in large diameter blastholes. This phenomena consists of the explosion gases pushing the air that exists between the column charge and the wall of the blasthole, compressing the cartridges in front of the shock wave, destroying the hot spots or excessively increasing the density of the explosive.

Cuts can be classified in two large groups:
- Parallel hole cuts
- Angled hole cuts.

The first group is most used in operations with mechanized drilling, whereas those of the second have fallen in disuse due to the difficulty in drilling. They are only applied in small excavations. In the following, the different types of cuts are explained in their order of importance, as well as calculation of the patterns and charges in the rest of the sections which are, generally speaking, independent from the type of cut applied.

The primary function of the cut remains the same regardless of the type of cut or its variations. To be successful, it must break the rock and move it forward. This creates a void which provides additional relief for the remaining holes to be fired later in a predetermined sequence.

Burn Cut (Hopler)
This is a closely spaced group of boreholes drilled parallel to the direction of advance and perpendicular to the existing face. They are blasted at or near the center of the face to break a roughly cylindrical opening to the intended depth of the round. The boreholes that surround this cut area are sequenced to fire later and break to this newly created opening. It is important that the burn cut holes be drilled accurately and parallel to each other. Improper location of the burn cut holes may result in "bootlegs," (lengths of borehole left after the blast that may contain un-exploded explosives. Bootlegs may be caused by:
sympathetic detonation from propagation between holes that are too close, thereby destroying the delay sequence
excessive burdens between the bottom of the holes.

![Figure 4-4: Various types of burn cuts (solid dots are loaded holes)](image)

The varied rock types and structures determine the drill spacings, detonator delays, and types of explosive that will successfully fragment and remove the rock to the full depth of these boreholes.

**Angle Cut**

An angle cut is a group of boreholes drilled at various angles inclined to the free face to provide as much freedom of movement for the rock as possible. Types of angle cuts include the Vee, the pyramid, and the drnw or hammer. Angle cuts generally require fewer holes and less powder-per-round than burn cuts. However, they are generally more difficult to drill and require more experienced miners. Usually the angle cut will move the rock further down the heading and produce coarser fragmentation out of the cut area. Longer drill steel is required in the cut holes to achieve the same depth as the surrounding boreholes in the round. In narrow headings it is difficult to drill a sufficiently wide angle to insure "pulling" the cut.
Figure 4-5: Angle cuts (Vee or wedge, top left; double Vee or baby cut, top right; three-hole pyramid, bottom left; and a draw cut, bottom right)

Cylindrical cut
The cylindrical cut can be considered a parallel hole cut, and is the most frequently used in tunnelling and drifting, regardless of their dimensions. It is considered to be an evolution or perfection of the bum cuts which will be discussed later on. This type of cut consists of one or two uncharged or relief blastholes towards which the charged holes break at intervals. The large diameter blastholes (65 to 175 mm) are drilled with reamer bits which are adapted to the same drill steel which is used to drill the rest of the holes.

All the blastholes in the cut are placed with little spacing, in line and parallel, which explains the frequent use of jumbos which come with automatic parallelism. The type of cylindrical cut most used is the four section, as it is the easiest one to mark out and execute. The calculation method for patterns and charges of this cut and for the rest of the tunnel zones, uses the Swedish theories and empirical updates.

Figure 4-6: Cylindrical cut

Blast Layout
Designing the actual blast is considerably difficult. Beginning with an initial design, an experienced miner would modify the pattern as ground conditions
change or as assessing blast results point to necessary changes. Design specifications are typically made for a ‘standard’ for each rock type in a mine, to be modified depending on the cut location, ground conditions, or other factors that may improve the results. Note that in most countries, drilling into bootlegs is not permitted. Therefore cuts usually alternate from side to side.

There are two mechanisms used to sequence the holes, some blast the back last, other the floor (ostensibly, to ‘fluff-up’ the muck that has compressed from downward blasting). Figure 4-7a shows the general areas that are initiated in order based on blasting the floor last while Figure 4-7b shows a layout when blasting the back last.

![Figure 4-7: Sequencing drifts](image)

Follow the mandatory readings to determine the charge lengths, densities, hole spacing, and lengths of the blast.

**ADDITIONAL RESOURCES XII**

Additional learning resources include:

- Lecture Module 5.1 – Drift Development (mandatory)
- Assignment 6 – in class (part of lecture)
Production Blasting

Metal-bearing orebodies are extracted in underground mines by various methods, depending on the size, orientation, depth and geological characteristics of the deposit. However, the excavation work is usually divided into two broad categories; development and production. Development involves tunnelling, shaft sinking, cross cutting, raising, etc., so that the ore bodies are easily accessible and transportable after excavation. The production work can be subdivided into two categories: short-hole and long-hole blasting.

Short-hole Production

In short-hole blasting the diameter and length of shotholes are usually limited to 43 mm and 4 m respectively. Short-hole blasting can be used in both stope and room-and-pillar mining. Short-hole 'breast stope' blasting is most commonly applied in narrow, tubular orebodies such as gold or platinum reefs. The usual blast pattern in a South African narrow reef of about 1.0 to 1.2 m stope width (or height) comprises staggered rows of 35 mm diameter blastholes. The length of the blastholes is about 1.2 m. Both rows are drilled at 70° to a line parallel to the stope face and close to the hanging wall and footwall, the burden of each hole being 0.5 to 0.6 m.

The most common types of explosive used include a cartridge of nitroglycerine-based semi-gelatine or emulsion explosive with a composition density of 1.25 g cm\(^{-3}\) and a column of pneumatically loaded ANFO with a density of between 0.8 and 0.9 g cm\(^{-3}\). In an average stope, there are about 120 shotholes per panel and, to prevent the hanging wall and footwall being damaged, the present practice is not to detonate two or more shotholes simultaneously. Many mines use a pyrotechnic system of initiating explosives in the shotholes. Capped safety fuse and igniter cord (burning speed of about 18 s m\(^{-1}\)) make up the carrier of the initiating system. The ends of the safety fuses protruding from the charged blastholes are connected sequentially with a trunkline of igniter cord, as seen in Figure 4-8.
Figure 4-8: Short-hole production blast.

The igniter cord itself is usually fired by an electric starter for igniter cord (ESIC). The ESIC consists of a plastic capsule into which a standard electric fusehead (fitted with short lead wires to energise the system with an exploder) and a pigtail of fast igniter cord which has been crimped. If a correct combination of burning speeds of safety fuse and igniter cord is achieved the shotholes are detonated in sequence at 4 s to 8 s intervals. The distance separating a detonating shothole and the flame front in the igniter cord is known as the burning front, and is usually between 3 m and 5 m. Experience has established that the likelihood of igniter cord trunkline cut-offs from rock projectiles decreases as the burning front increases. The powder factor varies between 1 to 2 kg/m³.

Longhole Production Blasting
Basically there are three long-hole blasting systems: ring blasting, bench mining and vertical crater retreat.

Ring Blasting
Ring blasting has wide application in massive are bodies with their high rate of extraction at low unit costs. The method requires three distinct operations
- The formation of a tunnel, called the ring drive, from a sublevel along the axis of the proposed excavation.
- The excavation of an empty space, called the slot, at the end of the ring drive, to the full width of the excavation.
- The drilling of sets of radial holes, called Rings, parallel to the slot at appropriate spacing and burden.
In ring drilling, the distance between two consecutive rings is called the burden, whereas the term spacing refers to the ends of the adjacent holes in the same ring, measured at right angles to one, and straddling the outline of the are block, using construction lines (Figure 4-10). Normally, the spacing/burden ratio is about 1.3, but it can be as high as 1.5.
When blasting into a confined slot, an allowance of at least 30% expansion from the solid is recommended, in order to achieve good fragmentation. Also the stemming length should not be more than two-thirds of blasthole length. A guide to the powder factor for some types of rocks is given in Table 4-1.

**Table 4-1: Powder factors for Ring blasting**

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Powder factor, ( q ) (kg m(^{-3}))</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartzite</td>
<td>0.4 – 0.8</td>
</tr>
<tr>
<td>Shale/schist</td>
<td>0.5 – 0.7</td>
</tr>
<tr>
<td>Basalt/dolerite</td>
<td>0.7 – 0.9</td>
</tr>
<tr>
<td>Granite</td>
<td>0.7 – 0.9</td>
</tr>
<tr>
<td>Haematite</td>
<td>0.9 – 1.1</td>
</tr>
</tbody>
</table>

Since the blastholes in a ring radiate from a centre point of the ring drive, the collars of these blastholes will be fairly close to each other. Hence the blastholes need to have a variable stemming length (Figure 4-11) in order to avoid serious overcharging in the ore body close to the ring drive.

It should be noted that the degree of success in ring blasting depends on the degree of accuracy in designing and drilling the blasting holes. The blasthole
in each ring could be drilled upward as well as downward. If conditions permit, it is preferable to use one delay per ring, but this may generate a shattering effect in the adjacent rock. This phenomenon can be avoided by using two or even three different delays per ring. These delays are not alternated between holes but are apportioned to whole sections of the ring.

Figure 4-11: Stemming length for ring blasting

Bench Blasting
Bench blasting is essentially similar to surface excavation. First a development heading is excavated at the top sublevel to provide drilling space. Then, depending on the thickness of the ore body and/or the availability of drilling machinery, either vertical or horizontal blastholes are drilled to increase the height of the excavation (Figure 4-12). The blastholes can be from 32 mm up to 250 mm in diameter, depending on the amount of pull (which could be 6.0 m or more) and other factors such as quality of rock, fragmentation requirement, etc.
Vertical crater retreat (VCR)
Essentially, vertical or subvertical blastholes are drilled downward from the top level to the bottom level. A cuboid of ore body can be excavated from the lower level upward by a number of horizontal slices using the same blastholes.
It is imperative that the first set of charges in the blastholes breaks through into the undercut. Theoretically, spherical charges should be placed, to obtain maximum cratering effect. In practice, however, this is achieved when the deviation from the true spherical charge is not greater than a 1:6 diameter to length of explosive column ratio. In the vertical crater retreat method, gravity enlarges the crater dimensions by excavating the whole rupture zone. The size of this cavity can exceed the optimum distance of the charge from the back many times and its extent depends on rock properties and the local structural geology. The optimum distance for positioning the explosive charge should be determined through small-scale crater tests using the same explosive-rock combination.

The blasthole charging details have been well described in a case study where the blastholes were of 159 mm diameter and average length 35 m. First a square section wooden plug (100 X 100 X 200 mm) was lowered down the hole, using a 6 mm diameter polypropylene cord, to the desired depth of blockage, about 1.8 m above the free face (Figure 4-14). Then a small amount of 16 mm crushed rock was poured into the hole to obtain a seal. The average explosive charge length was 0.8 m, in the middle of which was a primer attached to a 10 g/m detonating cord down line. The explosive charge was first stemmed by 1 m of sandfill, followed by 1 m of crushed rock. Finally, two detonators of the same delay were used to initiate the down line. The average depth of slice achieved was 3.4 m.
The main advantage of the VCR technique is the decreased possibility of damaging the surrounding rock which in turn reduces the risk of dilution. This technique has been applied successfully in blasting out pillars between cemented cut and fill stopes. It is important to use an explosive of high energy and high-detonation velocity in the VCR method to achieve maximum advance with minimum drilling costs.

**ADDITIONAL RESOURCES XIII**

Additional learning resources include:

Lecture Module 5.2 – Production blasting (mandatory)

**Controlled Blasting**

The following discusses more of the theoretical aspects of blasting design in avoiding damage control. To facilitate the discussion, consider an area in the blast called the blast damage transition zone (BDT), as seen in Figure 4-15. The extent of each zone is characterized by a radius from the center of the production change. The zones, their extent (expressed as hole diameter) and the corresponding peak particle velocity (PPV values resulting when ANFO in medium strength rock are assumed can be similar to the value in Table 4-2.

![Figure 4-15: Diagrammatic representation of the BDT of a fully charged hole.](image)

**Table 4-2: BDT Characterization**

<table>
<thead>
<tr>
<th>Damage zone</th>
<th>Extent</th>
<th>PPV (m/sec)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Crushed ($R_c$)</td>
<td>4 → 6 D</td>
<td>20</td>
</tr>
<tr>
<td>Fractured ($R_f$)</td>
<td>12 → 15 D</td>
<td>5</td>
</tr>
<tr>
<td>Influenced ($R_i$)</td>
<td>50 → 60 D</td>
<td>1.5</td>
</tr>
</tbody>
</table>

These zones have been drawn on Figure 4-16 for a blast containing 2 rows of production blast holes (9-7/8" diameter). As seen, it is a square pattern with 5 holes in each row, a burden of 20' and spacing of 20'. In this example it has been assumed for simplicity that

$R_c = 5D = 4ft$

$R_f = 12D = 10ft$

$R_i = 55D = 45ft$
Figure 4-16: DBT for a two row production blast

After blasting, the situation is shown in Figure 4-17. In the BDT zone there exists a crushed and a fractured zone surrounding each production hole. Although the crushed zone as well as a small portion of the fractured zone may have been loaded out along with the rock from production rows 1 and 2, here the rock will be assumed to still be in place. The zone lying between the fracture zone and the boundary of the BDT consists of an inner portion which has been influenced by both rows of blastholes (to a distance of about 25') and an outer portion (20' in extent) influenced by just one row of production holes.

Figure 4-17: BDT after blasting two production rows.

Therefore in the design and implementation of any blast damage control techniques, one should take into account the BDT limit into account.

Line Drilling

Line drilling, as the name implies, involves the drilling of closely spaced holes along the limit of the excavation. This is shown with respect to the example case in Figure 4-18. The object is to create an artificial plane of weakness which serves to limit the extent of the fracture and influence zones from both the production holes and any buffer (helper) holes placed between the final production row and the perimeter. Generally, these line-drilled holes are not charged with explosive but, if charged, it is with detonating cord or a highly decoupled charge. The purpose for lightly charging the holes is to destroy the
integrity of the rock web. As can be appreciated, close drilling control is essential for the method to succeed. The holes must be drilled so that they all lie in one plane corresponding with the dip of the final pit wall. Some recommendations for hole spacing as provided in Table 4-3. To get hole spacing one multiplies the values in the table by the hole diameter expressed in the same units. When line drilling 6" diameter holes in copper ore, the hole spacing (c-c) should be 12".

![Line Drilling Diagram]

**Figure 4-18: Line drilling positioned along the planned final perimeter**

**Table 4-3: Factors for determining hole spacing**

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>Taconite</td>
<td>2.0</td>
</tr>
<tr>
<td>Copper ore</td>
<td>2.5</td>
</tr>
<tr>
<td>Asbestos ore</td>
<td>4.0</td>
</tr>
</tbody>
</table>

In the example there is 45 ft wide zone between hole row 2 and the perimeter. To excavate this rock one might consider using another row of production holes at normal burden and spacing and then one or two rows of smaller diameter buffer holes. For line drilling to be most effective:

- It must be used in conjunction with a buffer row
- The main excavation charges should be 1 to 3 rows from the pit limit.

Line drilling produces one of the best final surface - a smooth, clean face with no backbreak or crest fracture. However because of its high drilling cost, the method has not been commonly used in open pit work.

**Pre-splitting**

The pre-splitting technique also involves the careful drilling of relatively closely spaced parallel holes along the final perimeter. Now however the holes are lightly charged and shot instantaneously. The objective is to generate a line of cracks connecting the holes. In this way, it is intended to achieve nearly the same effects:

- Terminate the growth of the radial cracks
- Act as a barrier to the shock wave
- Provide an escape route for the explosive gases.
It attempts to produce the same effect as with line drilling but at a significantly lower cost. Obviously to be of any use the presplit line must be created prior to the blasting of any holes lying closer than I-BOT distance away.

Figure 4-19 illustrates typical presplit blast layout using 102-mm (4-in) diameter presplit holes for 250-mm (9y'-in) diameter production holes. For this type of blast, presplit holes would normally be drilled first, ahead of main production holes. The choice can then be made between loading and firing the presplit line or infilling the main blast. In the latter case, the presplit line would be fired instantaneously 100 to 150 millisecond before the main blast. As shown in the figure, the presplit line is formed ahead of the main blast and allows the gas being driven back from the buffer row through the radial cracks to terminate at the presplit line.

![Figure 4-19: Presplit blast coupled to a 250mm production blast](image)

The presplit row in Figure 4-19 has a spacing of 2 m for a 102-mm (4-in.) diameter hole and is inclined at 15° to the vertical. The presplit angle is somewhat dictated by rock structure although a slight angle is preferred regardless of structure for long-term stability as well as for best initial results with large production holes. The figure illustrates the upper bench where two benches will finally run together to form the final face between berms.
Presplit drill requirements become clear when presplit holes needed for the next bench are considered. The drill must be capable of drilling close to the previously produced bench face at an angle of 15° beneath itself so the face can be continued to depth. Currently, this means some form of drifter drill is required limiting the hole size to 102 to 127 mm (4 to 5 in.) diameter. The back row of the main production blast, termed the buffer row, must also be carefully designed with respect to standoff distance from the presplit row and spacing as well as explosives load. The inset sketch on the right side of Figure 4-19 shows how the top portion of the buffer row hole charge acts as a spherical crater charge breaking to the bench surface. Subsequently, main blastholes after the buffer row are designed at regular spacing, burden, and loading for the type of material blasted. One further point to note from Figure 4-19 is the subgrade or, more accurately, lack of subgrade used on the presplit and buffer row holes. This is to prevent damage to the bench below or to the wall at that point.

Figure 4-20 is a graph of recommended hold spacings used in presplitting as a function of hole diameter. These data cover various types of material, but are not fully specified in the open literature.

![Graph of recommended hold spacings used in presplitting](image)

**Figure 4-20:** Relationships between hole diameter and spacing for presplitting from various researchers.
Smoothwall blasting

In smoothwall blasting, as opposed to line drilling and pre-splitting, the final pit perimeter lies in the zone of influence from the final row of production holes. This is shown diagrammatically in Figure 4-21. Since the final row of holes lies in the influenced zone, some minor crest fracturing or backbreak may result but the amount of damage is much less than would be produced by the main production blast if no control blasting was used.

Figure 4-21: Smoothwall damage zones

There are five general followed in the design of the smoothwall row:

Rule 1
The burden, spacing and charge concentration of the smoothwall line of holes are selected so that the extent of the associated influence region does not exceed that of the production holes. The hole size for the smoothwall and buffer row holes may be the same as in the production round with the required reduction in influence zone dimension occurring through pattern adjustments and decoupling or smaller diameter holes may be used with or without decoupling.

Rule 2
The hole spacing is less than the burden. Often the relationship S/B= 0.8 is used

Rule 3
The holes in the smoothwall row are shot on the same delay with detonating cord downlines to assure as simultaneous detonation as possible.

Rule 4
The delay time between the helper row (that adjacent to the smoothwall row) and the smoothwall row should be chosen so that the smoothwall holes can shoot to a free face.
Rule 5
All of the smoothwall and buffer row holes are shot together with the main production round.

Today, smoothwall blasting is much more common to use than most other methods in mining operations because the method involves less drilling and less complicated blasting.

**Trim Blasting**
Trim blasting, as the name implies, involves trimming away some of the fractured and influenced rock from the pit perimeter after the production blast has been shot and cleaned up. The trimming may be accomplished using one or several rows of blast holes depending on where in the BDT zone it is desired that the final crest should fall. The design process differs from the smoothwall technique in that the layout begins at the actual pit perimeter and works outward toward the desired final pit limit rather than vice versa. This is shown diagrammatically in Figure 4-22. Some rules for selecting the burden, spacing and charge concentration as a function of hole diameter are presented in Table 4-4. As in all other types of perimeter control, accurate drilling is important. To achieve the best results, the holes should be drilled at the final pit slope angle. The boreholes are drilled in a line along the planned excavation limits, loaded lightly, and blasted to remove the undesired material. As noted earlier, a reduced explosive load can be obtained in various ways. The use of low density, bulk-loaded explosives in larger diameter holes is one way of improving the economics of the method.

![Figure 4-22: Trim hole row trimming the fractured and influenced rock.](image-url)
Blasting Ornamental Rock

Ornamental rock is all stone that is used, in blocks or slabs, for its aesthetic characteristics such as color, texture, shine, grain, etc. and technical such as strength, facility of elaboration, polish, etc. The most common types of rock can be generically classified in three large groups: granites, marbles and marmoreal limestones.

Granite is cut from the "bed" of the quarry with a jet piercing machine that produces a flame burning at approximately 3,000 degrees Fahrenheit. This high-velocity flame, created by burning oxygen and fuel oil, is directed at the granite to be removed, causing a continuous flaking action. As the flame nozzle is moved up and down, a channel is created around large sections in the quarry.

In some quarries, diamond wire saws are used. A long loop of small steel cable, impregnated with industrial diamond segments, cuts the sections free from the bed of the quarry. After a section has been completely wire sawed or channeled by the burner, it is separated from the bottom by explosives.

Likewise, when high-speed drills are used, rows of drilled holes are loaded with explosives. The explosives are detonated to free the sections of granite on all sides and on the bottom by explosives. The large sections are then broken into workable sizes by wedging. In this process, steel wedges are driven manually into holes previously drilled along the desired line of cleavage. The sections are readily forced apart and cross-wedged into rectangular blocks. Large cranes, or derricks, lift these blocks to the quarry's rim (Figure 4-23). Requirements for monumental granite are exacting, and only about 50 percent of the granite removed from the quarries finds its way into finished monuments. The remainder is consigned to commercial applications such as street curbing and gravel, or is sent to "grout piles" as waste products.
Figure 4-23: Ornamental stone materials handling – cranes.

Case studies show that the cutting methods consist in primary separation from the rock mass of a large block (100 to 4000 m$^3$), in parallel piped form, which is subdivided afterwards to achieve sizes that are easily handled and within the ranges that the transformation industries require, generally lengths of 1.8 to 3.5 m, widths of 1 to 1.50 m and heights of 0.9 and 1.2 m. The cutting technique is usually with explosives, although not exclusively, because cutting systems with helicoidal and diamond wire, with mounted rock cutters, with flame torching and with water jet kerfing are often applied. The blasting techniques are a special type of presplitting, but with slight variations as it is of maximum importance not to damage the rock and at the same time take into account the properties: strength, homogeneity, schistocity, fissurization, etc. Although it is difficult to give general recommendations for design in this type of blasting, as there are many different rock types and exploitation conditions, the following criteria should be of use:

**Drilling diameters.** They depend upon the phase of excavation and the type of rig used, but generally around 25 to 45 mm.

**Spacing.** It is established as a function of the rock properties and explosive charge characteristics. The usual interval is between 4 and 8 D. To be able to make an analytic calculation, the formula suggested by Berta can be applied:

$$ S = \frac{2 \times PE_s \times \rho_e \times d^2}{RT \times D} + D $$

where: $PE_s =$ Specific pressure (MPa),
$\rho_e =$ Density of the explosive (g/cm$^3$),
$d =$ Diameter of the explosive charge (m),
$D =$ Diameter of the blasthole (m),
$RT =$ Tensile strength (MPa).

**Explosives.** In the vertical benching planes detonating cords with a core of pentrite are usually used, while for the horizontal planes explosives of low detonation velocity are also used, as they generate a large volume of gases.
In these last planes of the cut, the structural properties of the rock mass should be used to advantage. In some countries, there is extensive use of charges prepared in connecting plastic tubes that contain powdery explosives with low density and detonation velocity, made up of nitroglycerine, sodium nitrate and other ingredients.

**Powder factors.** These vary greatly depending upon the type of rock, explosive and extraction phase.

**Charge configuration.** The explosive columns are generally designed to be continuous and decoupled with an air chamber although, in some cases such as in hard rock, to increase the energy transmitted to the rock by the detonating cords, the blastholes are filled with water. Also, if a blackening by explosion smoke of the cut surfaces is to be avoided, the holes can be filled with sand or drilling waste.

**Distribution of the charge in the borehole.** In order to eliminate breakings or fracturation in the corners of the blocks, it is suggested that empty holes be used at the end of the line or next to the free surfaces. Apart from this, in vertical blastholes there is no subdrilling and they are usually drilled to a few centimeters above the horizontal plane.

**Stemming.** They are necessary to use the maximum pushing power of the gases. As the rock characteristics become poorer, the heights are usually shortened to assure that the pressure of the gases do not act upon the rock for a long period and therefore produce damage. In general, with the detonating cords the stemmings are small, whereas with powder a larger confinement is necessary.

**Initiation.** As in contour blasting, instantaneous initiation of all the blastholes with detonating cord is recommended.

Example
A block of granite is to be extracted by drilling blastholes and blasting with detonating cord. What should the spacing between boreholes be when the initial data is:

- Tensile rock strength \( RT = 10 \text{ MPa} \).
- Drilling diameter \( D = 0.032 \text{ m} \).
- Diameter of the detonating cord core of pentrite \( d = 0.0034 \text{ m} \).
- Density of the pentrite charge \( \rho_e = 1.3 \text{g/cm}^3 \).
- Specific pressure \( P_{E_s} = 1200 \text{ MPa} \).

\[
S = \frac{2 \times 1200 \times 1.3 \times 0.0034^2}{10 \times 0.032} + 0.032 = 0.14m
\]
The ratio $S/D$ is equal to 4.37, which is within the practical interval of 4 to 8 $D$. If the rock were of worse quality with a tensile strength of 5 MPa, the spacing should be increased to $S = 0.26$ m.

**Underwater Rock Excavation**

Underwater blasting of rock is usually done to deepen the sea bed in and around a harbor to allow the passage of larger vessels, or excavate trenches for pipelines. The technique can also be extended to winning underwater mineral deposits. The most important factor is the placement of the charges in the designated area. This is accomplished in a number of ways:

1. Using divers to drill and to charge the blastholes.
2. Using shaped explosive charges fastened in a predetermined pattern onto a metal frame or high explosives packed in boxes and connected to a line of detonating cord and then lowered onto the sea bed.
3. Using a barge or platform to drill and charge the shots.

Method 1 is only economical if the water is shallow and the area of excavation is small. The procedure is then the same as for land excavation.

Method 2 is fairly labor intensive but economical for excavating selected small areas. Its success depends on the intimacy of the contact between the explosives and the rock, whose thickness to be removed by blasting in one pass is generally limited to one meter. In deep water (say $>10$ m) this is the only method that can be applied economically in the present state of technical knowledge. Moreover, in deeper water, a charge becomes effectively tamped by the pressure of the water, and consequently the efficiency of the explosive in breaking rock is increased. Individual charges are normally 10 to 25 kg, depending on rock type, and the corresponding charge ratio varies between 3.5 and 7 kg/m$^3$.

Most underwater blasting is done by method 3, that is, from a platform, which may be of either the floating or the jack-up type. Overburden drilling (OD) rigs are used by both types and the blastholes are normally between 51 and 102 mm in diameter. OD rigs are fitted with chucks so that the equipment can drive a hollow outer casing of steel tubing through soft material, as well as drill rock through the casing. Strict control on positioning each blasthole at its designed place is extremely important for the success of this method.
The sequence of operations is shown in Figure 4-24 and can be described as:

- The outer casing, fitted with a hollow ring bit with serrated edges, is driven through the water until the ring bit, by rotating, grips firmly into the top part of the bedrock (a).
- Then the casing is uncoupled from the drill chuck and the normal drill string is inserted through the hollow casing for blasthole drilling (b).
- When the required depth is reached (Note: normally, the subgrade length is the same as the burden), the drill string is withdrawn (c) and the charging procedure takes place through the hollow casing as follows: the primer cartridge with detonating cord or signal tube is pushed through the hole first by using a string of stemming rods with a flat-ended wooden coupling for easy handling of explosives (d).
- The length of the cord or tube should be adequate to allow for rise and fall of the tide when it is brought up to deck level inside the casing. The required cartridges are then added. When charging is complete, a slip-ring attached to a hemp rope is lowered to the bottom. Then as the outer casing is withdrawn, the tube or cord is retrieved from inside the tube (e) and brought up to deck level.
- A weight is attached to the end of the cord or tube, and the latter is placed on a roller away from the drilling activity. The gantry is then moved to the next line of holes and the operation repeated.

The explosive used in this type of operation should have high velocity of detonation and high density. It should have a good degree of water resistance so that 24-hour immersion does not affect its performance, yet its properties should be such that it would be made inert by a long period (say
one month) of immersion. The powder factor for this type of operation varies between 0.5 and 2.0 kg/m$^3$ depending on rock conditions. The waterborne blast-induced overpressure $P$ (kPa), can be calculated approximately from

$$P = 55 \times 10^3 \frac{m^{1/3}}{R}$$

where $m$ is mass of explosives in kg, and $R$ is the distance from the charge to the point affected by pressure, in meters. It is estimated that a peak waterborne overpressure not exceeding 40 kPa is safe for humans and animals.

**Controlled Blasting Rules of Thumb - Supplemental**

The following points are controlled blasting – presplitting and trim blasting – additions that blasting consultants consider important.

1. Critical geometric parameters are hole diameter, decoupling ratio ($d_{\text{powder}}/d_h$), spacing/diameter ratio ($S/d_h$), charge density (charge/unit length) and drill accuracy.
2. Critical geologic parameters are rock brittleness, rock strength, fracture density, and fracture orientation.
3. Typical $S/d_h = 10-14$ for pre splitting, 16 (to 18) for trim blasting, with the lower values necessary under less ideal conditions. $S/d_h$ is more likely to be too large than too small. If the ratio is too high a more rugged, more damaged slope will develop, and there is essentially no way to compensate through loading or timing adjustments.
4. Typical decoupling ratios = 0.1-0.3, the higher ratios being necessary in rock of higher tensile strength. Decoupling avoids shattering and reduces gas penetration.
5. Typical charge densities 0.2-0.4 lb/ft in-hole, face coverage 0.10 - 0.18 lb/sq ft, the higher values associated with stronger rock and/or larger hole diameters.
6. Use linear, distributed charges with cord (50 to 400 grain most common), presplit powder (skinny segmental or linkable cartridges), cartridges taped to cord at intervals, air decked loads, or mass decked loads, in order of increasing potential for rock damage.
7. Presplit works best in brittle, homogeneous rock, trim blasting in fractured, weaker rock.
8. Rock with closed fractures spaced less than 0.3S, or open fractures spaced 0.6S or less, will dissipate presplit elastic stresses. Compensate by closer hole spacing.
9. Fracturing striking less than 15 degrees to a presplit line is very difficult to overcome. Fracturing striking 15-45 degrees is less difficult, but may require decreasing spacing by 25-50%. Unless the fractures are very tight and the rock is very weak, even closely-spaced presplit will tend to break to such fractures. Trim blasting may be more cost-effective in such cases.
10. Presplit shot together with production must be timed at least 50 ms ahead of small-diameter (<4") production holes, at least 100 ms ahead of medium-diameter (4"–8") production holes, and at least 200 ms ahead of large diameter (>8") production holes. More time is better. The shot width should be at least 3 S (spacing lengths) for presplit confinement.

11. Presplitting: simultaneous firing is ideal. However, if the resulting charge weight exceeds vibration/scaled distance criteria, presplit holes can be delayed as much as 25 ms hole-to-hole, but with less effectiveness. To meet vibration concerns, start by grouping presplit in as many simultaneously-fired holes as possible, delaying 9 ms between holes or groups of holes, and then add time as necessary. Trim blasting: hole-to-hole timing is best; add an extra delay period or two for trim hole relief.

12. Standoff between presplit and buffer (if used) or production holes should be $0.3-0.5 \times B_{aProduction}$ ($0.5-0.75$ if trim blasting). The back break from production or buffer holes is needed for additional fragmentation in front of controlled holes. Refine standoff according to the buffer/production arrangement.

13. Presplit holes ideally are not stemmed, to permit release of gases. To control noise, they can be stemmed, $0.5S < T < S$. Stem trim holes $0.7S - 1.0S$.

14. Angled controlled holes perform best, but seldom are drilled flatter than 50 degrees or so.

**ADDITIONAL RESOURCES XIV**

Additional learning resources include:
- Lecture Module 5.3 – Controlled Blasting (mandatory)
- Guest Lecture – Bob Cummings

**Environmental Issues**

The use of explosives is probably the most widely used means of fracturing rock. Rock can also be excavated by methods other than blasting such as the use of rippers, hydraulic bursters, plugs and feathers, etc. Techniques involving explosives differ from other available systems in that the energy applied is released in a matter of milliseconds. If the energy release process is not adequately controlled, there is a potential danger of environmental disturbance. Environmental disturbance includes the effects of airblast, fly-rock, changes in the natural profile of the ground, dust, fumes, and ground vibration.

The blasting engineer should be aware of the need for defense against allegations of damage caused by the above factors. Before any explosives are used, it is good practice to carry out a detailed survey of all properties that might conceivably be considered at risk of damage, and then to keep meticulous records of all blasts.

It is now appropriate to analyze the energy distribution after the detonation of a quantity of explosives. The explosive's potential energy can be
manifested as heat, rock fragmentation, fly-rock, airblast, and ground vibration. Depending on the placement of the explosive charge, the percentage distribution of the above factors may vary, except for heat emission, which will be constant in all situations. For example, if a quantity of explosive is detonated on the ground surface, the major part of the explosive's energy will be converted into airblast, whereas if the same quantity of explosive is buried very deep in the rock, the majority of the explosive's energy will be manifested as ground vibration. Each aspect which can cause environmental problems is now discussed.

**Airblast**

Air overpressure due to an explosive charge (for a given amount) increases rapidly as the confinement decreases. Severe airblast is caused either by inaccurate charging, with the consequent wastage of explosive energy in the air, or by the firing of exposed detonating cord in a sensitive area.

The ideal blasting condition for minimum airblast effect is where temperature decreases with increasing altitude, causing a decrease in the sound wave velocity, inducing the waves to bend upward away from the ground. On the other hand, if the air becomes warmer with increasing altitude, an increase in sound wave velocity results, causing the waves to be returned to ground by refraction. Moreover, if there is a strong wind in a particular direction the refracted sound waves are channeled into the wind path. This phenomenon greatly increases the airblast pressure at focal points.

The most damaging cases of airblast are caused by unconfined surface charges. The resulting airblast overpressure may be estimated from the following formula:

\[
p = 185 \left( \frac{R}{m^3} \right)^{-1.2}
\]

where
- \( p \) = pressure (kPa),
- \( m \) = explosive charge (kg)
- \( R \) = distance from charge (m).

For confined borehole charges, airblast overpressure may be estimated from

\[
p = 3.3 \left( \frac{R}{m^3} \right)^{-1.2}
\]

When assessing the effect of airblast it is usual to correlate this with the possibility of cracking panes of glass. The pressure waves generated by
explosives consist of energy over a wide range of frequencies, some of which are audible, but most of which occur below 20 Hz frequency in the concussion range, which is not detectable by the human ear, but can damage structures. While the most frequently mentioned complaint resulting from airblast is cracked plaster, research has shown that window panes fail before any structural damage results. Certain well-known symptoms caused by the peak overpressure levels are given in Table 4-5.

<table>
<thead>
<tr>
<th>Symptoms</th>
<th>Peak (kPa)</th>
<th>Overpressure (dB)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rattle of loose windows</td>
<td>0.03</td>
<td>140</td>
</tr>
<tr>
<td>Failure of poorly-mounted window pane</td>
<td>0.69</td>
<td>150</td>
</tr>
<tr>
<td>Damage to strongly-mounted window pane</td>
<td>6.9</td>
<td>170</td>
</tr>
<tr>
<td>Cracking plaster</td>
<td>&gt;6.9</td>
<td>&gt;170</td>
</tr>
</tbody>
</table>

Noise from drilling and blasting can be annoying, and sometimes even painful. The technology for reducing noise and vibration from pneumatic and hydraulic drill operations without losing performance does not yet exist. On the other hand, close attention to the design of the canopy for compressors can bring about almost silent machines. The engine should be fitted with twin silencers and the inside of the canopy should be coated with sound-absorbing plastic foam to eliminate natural panel resonances.

The sources of objectionable blast noise are:
- the use of lay-on or plaster charges often used in secondary blasting;
- poorly stemmed holes;
- blown-out shots resulting from poorly designed blast patterns;
- exposed surface lines of detonating cord. In sensitive areas, a covering of 20 cm of stemming material over detonating cord will bring about a significant reduction in noise.

Atmospheric conditions such as temperature inversion and wind can lead to the phenomenon of focusing. Normally, temperature decreases with height; correspondingly, the velocity of sound in air also decreases with height, causing the sound waves to curve away from the surface. However, in a temperature inversion (air temperature increasing with height), sound velocity increases with height, causing the sound waves to bend downward towards the surface. The above conditions apply on the assumption that there is no appreciable wind. A combination of wind and temperature inversion can take place several kilometres from the blast and the overpressure due to airblast may increase by a factor of 100. Usually ideal atmospheric conditions for blasting exist in the early afternoon. These airblast wave effects can be seen from Figure 4-25.
Fly-Rock
Fly-rock is the term for undesirable projectiles of blasted material. In a particular type of rock there is a compatible relation between the height of the explosives column in the holes, drilling pattern, and charge ratio. When this is compromised, the explosive's gas energy is vented violently into the atmosphere and propels rocks in front of it. Moreover, deviation of blastholes in surface mining operations can effectively reduce the burden, causing fly-rock. Proper stemming has an important role in trapping explosive gases in the blasthole to do useful work such as rock fragmentation and throw. Crushed angular rock is recommended as the ideal stemming medium and the stemming length should not be less than the burden distance. The major causes of fly-rock in mining are shown in Figure 4-26.
Moreover, an out-of-sequence shot has the same effect as an overburdened hole and is liable to cause fly-rock. In excavation work close to buildings, special precautions may be taken such as placing a thick rubber mat (such as heavy conveyor belting).

Some very limited field studies reported by various researchers that suggest that for granite the maximum throw ($L$) as a function of the hole diameter ($d$) and specific charge is as shown in Figure 9.12. When the specific charge ($q$) is $q \leq 0.2 \text{ kg/m}^3$ there is no throw. For other values of $q$ the maximum throw is expressed by

$$L = 143 \ d \ (q - 0.2)$$

where

- $d = \text{hole diameter (inches)}$,
- $q = \text{specific charge (kg/m}^3\text{)}$,
- $L = \text{maximum throw (m)}$.

A typical specific charge in bench blasting is $0.6 \text{ kg/m}^3$. In this case the maximum throw expression becomes:

$$L = 57d$$

For a 10 inch hole diameter the maximum expected throw would be $L = 570$ m. There are a number of different situations in which the actual conditions depart markedly from the ideal:

- The explosive extends too high in the hole so that cratering to the upper surface occurs.
- An irregular face brings the explosive column too close to the free face resulting in cratering.

The lack of confinement offered by both of these situations provides a weak link for the gas to exploit. The rock plug involved is pushed out in an early
stage of the gas expansion process and the expansion energy is expended in propelling a relatively small volume of rock at high velocity. Hence the throw distance can be very great. Of all various types of flyrock, the most dangerous situation occurs when cratering occurs on the top part of the bench near the collar. If the weakest link in the system is the column of stemming and not the collar rock, this can be ejected much like a projectile from a cannon barrel. Some empirical and computer simulation have examined the maximum throw and boulder size as a function of hole diameter. It was found that for granite with a specific gravity of 2.6, the relationships for the maximum throw (L) involving rocks of diameter $\phi$

$$L_{\text{max}} = 260 \ d^{2/3}$$

$$\phi = 0.1 \ d^{2/3}$$

where $d =$ hole diameter (inches),

$\phi =$ boulder diameter (m).

Thus for a 10 inch diameter borehole the boulder size would be $\phi = 0.1 \ (10)^{2/3} = 0.47 \text{ m}$.

**Disturbance of the Natural Ground Profile**

In an open-cut mining operation by blasting, a turmoil of the natural surroundings is often experienced with the end result of residual benches of bare rock. This can resemble a deep scar in the immediate environment. If the mine working is below the natural profile of the surrounding country the void can be infilled with waste material, or can be filled with water to form a recreation area. However, if the workings are in elevated ground where older workings are actively eroding with rock fall and spalling onto the mine floor or benches beneath, the problem requires different treatment. A method of restoration blasting has recently been adopted in various areas, and has proved to be successful in providing an aesthetically pleasing landscape. This is the application of a carefully designed blast pattern in the berm (bench), initiated to reproduce a predicted sloping profile with infilled materials. The screen blast piles will mask the earlier scorch marks, and this will be the medium for selective establishment of vegetation.

**Dust**

Dust is mainly produced while drilling and could be a considerable nuisance in high winds. Apart from incorporating a dust collecting unit in the drill rig itself, or using wet drilling, very little can be done to suppress dust from this operation. Clearly the driller should wear a protective anti-dust mask. In a properly designed shot for rock blasting, very little dust is scattered. However, in demolition work, particularly of a tall structure, dispersal of fine dust is inevitable. This could be partly controlled by sprinkling a jet of water, and thereby creating an artificial curtain of fine mist to contain the falling
dust in a limited area. Jets of water are also sprayed in some surface mining operations where dust has caused problems in nearby habitats.

**Fumes**

The detonation of a commercial explosive in ideal conditions produces water vapor, carbon dioxide and nitrogen. However, in addition a small amount of undesirable poisonous gases, such as carbon monoxide and oxides of nitrogen, known as fume or toxic gases, are hazardous in confined spaces such as underground mines, tunnels, etc. Careful thought must be given to mechanical ventilation of such areas with fans so that fumes are diluted to a harmless level. This aspect is adequately safeguarded by legal requirements laid down by the relevant authorities. The fume characteristics of a cartridge explosive relate to the conditions where the explosive is fired in its cartridge. Removal of explosive from its cartridge will upset the oxygen balance and unfavorably affect the explosive's fume qualities. The presence of water in the blasthole may also adversely affect the fumes produced by a blast.

**Ground Vibrations**

When an explosive charge is detonated in a borehole, a pressure wave will be generated in the earth surrounding the hole. As the pressure wave travels away from the borehole it forms a seismic or vibration wave by displacing the particles around it. This excursion or oscillation of the individual particles is measured to determine the magnitude of the blasting vibration.

For elastic wave transmission the strain is directly proportional to particle velocity. Since structural damage, in general, is strain related, the use of peak particle velocity is accepted as the principal parameter for vibration measurement. The longitudinal, vertical, and transverse components of the seismic waves caused by blasting are generally measured by instruments. Calculation of the resultant of these three components is sometimes preferred and countries have different standards as regards the suitability of measuring planes, or the resultant. The same amount of charge at a fixed distance does not necessarily produce the same magnitude of vibration. The vibration is largely influenced by the prevailing constriction and by certain physical site factors. The peak particle velocity (ppv) of ground motion can be related to distance and instantaneous explosive charge by the following equation:
ppv = \left( \frac{R}{m} \right)^{-P}

where \( R = \text{distance in meters,} \)
\( m = \text{explosive charge in kg,} \)
\( K = \text{site factor constant, and} \)
\( P = \text{site exponent constant.} \)

The ratio \( \left( \frac{R}{m} \right)^{-P} \) is commonly known as scaled distance (SD). SD is a useful parameter for comparing one set of vibration results to others. Obviously, as the value of SD increases, the magnitude of ppv decreases. When blasting has to be done close to a sensitive structure, the following procedure determines the optimum charge. In a given situation when the same blasting is further away (the SD is more than 12), several (at least 6) vibration recordings are taken. On the other hand, if there is any possibility of causing damage to the structure by normal production blasts, a test shot (using a small amount of explosives so that the SD value is well over 12) has to be monitored at various distances.

The simplest method of establishing the site constants is to use log-log graph and put the measured values straight onto the axes, that is, vertical (ordinate) for ppv and horizontal (abscissa) for SD. Such a graph is known as a regression curve. Allowing for a certain amount of scatter due to the variation in blasting constriction, the graph is a straight line. The constants \( K \) and \( P \) are obtained, respectively, from the intercept of the regression line on the ppv axis, and by calculating the slope of the regression line. Figure 4-27 is such a graph obtained by plotting ppv against SD for 10 monitored results. After drawing the best fitting line, two other lines parallel to the first one are also drawn to encompass all points in the graph.

Example
Consider that there is a sensitive structure where maximum ppv is stipulated as 5 mm s^{-1}, and this structure is 300 m from the blast area, what will be the maximum instantaneous charge?

By drawing a horizontal line from ppv = 5, three values of SD can be obtained: 30, 40, 50. From this, it can be assumed determined that

\[
SD = \frac{R}{m^2} = 50
\]

will be the best solution. The safest maximum charge per delay becomes 36 kg from:
Figure 4-27: Obtaining site factors for ground vibrations.

It is essential for the blasting operator to know and to be able to control what will happen when an explosive charge is detonated. The factors listed in Table 4-6 identify the phenomena important in blasting.
Table 4-6: Factors that influence ground motion

<table>
<thead>
<tr>
<th>Variables within the control of mine operators</th>
<th>Influence on ground motion</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Significant</td>
</tr>
<tr>
<td>1. Charge weight per delay</td>
<td>X</td>
</tr>
<tr>
<td>2. Delay interval</td>
<td>X</td>
</tr>
<tr>
<td>3. Burden and spacing</td>
<td></td>
</tr>
<tr>
<td>4. Stemming (amount)</td>
<td></td>
</tr>
<tr>
<td>5. Stemming (type)</td>
<td></td>
</tr>
<tr>
<td>6. Charge length and diameter</td>
<td></td>
</tr>
<tr>
<td>7. Angle of borehole</td>
<td></td>
</tr>
<tr>
<td>8. Direction of initiation</td>
<td></td>
</tr>
<tr>
<td>9. Charge weight per blast</td>
<td></td>
</tr>
<tr>
<td>10. Charge depth</td>
<td></td>
</tr>
<tr>
<td>11. Bare vs. Covered primacord</td>
<td></td>
</tr>
<tr>
<td>12. Charge confinement</td>
<td></td>
</tr>
<tr>
<td>Variables not in the control of mine operators</td>
<td></td>
</tr>
<tr>
<td>1. General surface terrain</td>
<td></td>
</tr>
<tr>
<td>2. Type and depth of overburden</td>
<td></td>
</tr>
<tr>
<td>3. Wind and weather conditions</td>
<td></td>
</tr>
</tbody>
</table>

Figure 4-28: Ground vibrations results from 2500 blasts in 40 different operations.

A collection of scaled distance data for determination of peak particle velocity of some 2500 blasts from 40 mines, quarries and construction sites in the USA is shown in Figure 4-28, the graph being drawn from a considerable amount of scatter. The factors responsible for this scatter are the changes of geological conditions, differences between types of explosives, difference in the geometry of the blasts, and experimental error. The line labeled 50 is the best-fit line, whereas the other two are bounds below which 84% and 95% of
the data fall. The maximum ppv recommended by Standards Australia is given in Table 4-7.

### Table 4-7: Recommended maximum peak particle velocities.

<table>
<thead>
<tr>
<th>Type of building or structure</th>
<th>Particle velocity mm s⁻¹</th>
</tr>
</thead>
<tbody>
<tr>
<td>Houses and low-rise residential buildings; commercial buildings</td>
<td>10</td>
</tr>
<tr>
<td>not included in description below</td>
<td></td>
</tr>
<tr>
<td>Commercial and industrial buildings or structures of</td>
<td>25</td>
</tr>
<tr>
<td>reinforced concrete or steel construction</td>
<td></td>
</tr>
</tbody>
</table>

This table does not cover historic buildings, particularly if they are in an indifferent state of repair, or some sensitive structures such as reservoirs and buildings with long-span or suspended floors. In the absence of particular site-specific data, a working recommendation is a maximum ppv of 5 mm/s.

### Vibration reduction

By adapting the blasting method, drilling pattern, charging scheme, and ignition pattern, the size of the vibrations can be controlled. Among other things, the size of the vibrations depends on:

- Cooperating charge
- Confinement conditions
- The character of the rock
- The distance from the blasting site
- The geology, e.g., overlay soil types

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

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

At the moment of detonation, there should be as little confinement as possible. This can be obtained by:

- A carefully adapted ignition pattern, so that all the holes will break the burden in the easiest way
- Increased hole inclination (of the drillhole)
- A voiding too large burdens and choke blasting

For blasting at a shorter distance than 100 m, the risk for interaction between the different intervals is small. The risk for cooperation between the
intervals increases with large blast in quarries: for instance, where any structures sensitive to vibrations are situated a large distance away. The size of the vibrations is then influenced by:

- Charging level
- Interval times
- The resonance frequency of the ground (which depends on the depth and the character of the ground)
- The local geology

The cost for careful blasting near built-up areas increases very rapidly with decreasing permissible vibration level. The increase in costs primarily depends on the following factors:

- Drilling - smaller or greater number of drillholes.
- Charging - more detonators and higher cost of labor
- Blasting - more rounds and longer stand-up time

The costs of planning and control work will also increase in:

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

**Cooperating Charges – Ground vibration**

In the preceding section, it was shown how the maximum detonating weight of charge can be estimated when the vibration level and the distance are known. The weight of the charge in question is the maximum total weight of charges that can be initiated at the same time. Using delay detonators, it is possible to blast rounds with considerably higher total charge weights per delay interval. The higher the interval number, the larger a total charge weight per interval can be used because the scatter of the delay time increases with the nominal delay time. The cooperating charge is defined as the total charge per interval, multiplied by the reduction factor appropriate for the interval used.

In the USA, the USBM states that the delay interval should be equal to, or greater than, 8 ms in order that the two charges be considered separate charges.

The expression cooperating charge is somewhat inappropriate as it is only applicable at certain distances. When blasting two separate charges with detonators having the same interval number (Figure 4-29), if the vibrations are observed at a short distance (A), the charges do not cooperate; while at a long distance (B), interference and reinforcement of the two vibrations may occur. Whether two charges, initiated one after the other, cooperate or not depends on the following factors:

- Time interval between initiations
The effect of Delayed Explosion

Reduction of explosive weight per delay is perhaps the greatest factor in reducing the probability of structural damage due to ground vibration. Table 4-8 highlights this point for a situation where the total charge is 2900 kg, the ground vibration monitoring distance is 220 m, and the expected ground vibration is given in three different situations.

Table 4-8: Effect of delayed charges on ppv for a total charge of 2900 kg at a distance of 220m.

<table>
<thead>
<tr>
<th>Mode of Detonation</th>
<th>ppv in mm s⁻¹</th>
</tr>
</thead>
<tbody>
<tr>
<td>Instantaneous</td>
<td>75</td>
</tr>
<tr>
<td>Two delays with equal charges</td>
<td>43</td>
</tr>
<tr>
<td>Four delays with equal charges</td>
<td>25</td>
</tr>
</tbody>
</table>

Each delayed charge generates its own seismic wave which is separated from the subsequent delayed wave. However, precise timing for the detonation of each hole is imperative for effective delay blasting. The cardinal point for successful delay blasting is that the seismic waves from any detonating blasthole shall pass all other blastholes before any of them are initiated. If two seismic waves resulting from two blastholes meet at a point, the resultant motion will be the sum of the two motions and the vibration level will be significantly increased. Moreover, the seismic wavelength of the composite motion varies from a single wavelength to nearly twice the length of a single wave. At the point of maximum overlap, the period and frequency are those of the single wave. Since the period may approach twice that of a single wave, the frequency will be reduced by half. This condition may
produce a region of high seismic risk due to increased motion and reduced frequency of vibration.

**Effect of Geological Factors**

The intensity of ground vibration is often influenced by the following properties of rock:
- the elastic properties of the medium, which determine the propagation velocity of the seismic waves (predominant range and type of waves);
- the moisture content of the medium and the ground-water level;
- the topography and the geological structure, which may have an influence in focusing seismic waves;
- the absorption characteristics of the medium.

**Risk Assessment**

Near most sites where blasting is necessary, there are structures whose sensitivity will limit the maximum permissible vibrations. Client or contractor must then decide on the maximum charge that can be detonated without causing damage in the neighborhood. Where small-diameter holes are used, this maximum charge may be the charge having the same vibration effect as that from several holes detonated with detonators having the same nominal time delay. It is called the maximum cooperating charge. Because the cost increases considerably if the maximum cooperating charge has to be reduced, the economy of the blasting job is greatly influenced by that decision. Too large a cooperating charge may result in damages to neighboring buildings, damage claims, and even court disputes. Too conservative a decision on the size of the maximum cooperating charge will result in excessively increased costs and project time. To optimize the blasting work, it is necessary to carry out risk analysis in order to determine, first, what size vibrations the environment will accept and second, how large a charge can be blasted at a certain distance without exceeding that vibration limit. Before the blasting operations can begin, a risk analysis should be made involving a careful examination of the factors that can affect the blasting operations. The probability that a correct decision will be made increases as more information is made available. The decision data should be based on as many points as possible in the list below.

Checklist for risk analysis:
1. Has a geological examination been made regarding working site as a risk area?
2. Is there a potential risk for lowering the groundwater level?
3. What is the nature of foundations and underground parts of buildings in the area?
4. What is the type of construction and the condition of buildings within the risk zone?
5. Is there any equipment (such as computers, electron microscopes, laser equipment, relays...) which are sensitive to vibrations in the neighborhood?

6. Are there any underground objects (tunnels, cable trenches, telegraph cables, oil cisterns, district heating culverts,...) that might be damaged by blasting?

7. What connections are there between vibration values, cooperating charges, and distances?

8. How are the inhabitants in the neighborhood influenced?

9. Has information about the blasting job been distributed to the neighbors?

Sustainable Development Issues in Rock Excavation

Sustainability issues in rock excavation, aside from esthetics of the excavation and implications of the operation itself, deal primarily with communities around the explosives consumers and manufacturers/transporters. Communities can be exposed to noise, fly-rock, ground vibration, fumes, transportation or storage accidents, and dust. As the importance of the community’s concerns continue to rise, engineers should be more sensitive and knowledgeable of the needs of the community. Adequate information is key. In some regions, the issue of explosive theft is important as criminal elements can obtain explosives for illicit purposes. Adequate storage and compliance with all licensing and transportation safety protocols should be taken.

ADDITIONAL RESOURCES XV

Additional learning resources include:

- Lecture Module 5.4 – Vibrations (mandatory)
- Assignment 6 – in class (part of lecture)

Evaluation of Blast Results

Once the blast has been carried out, it is necessary to analyze the obtained results, as its interpretation will give the successive modifications of the design parameters for the following rounds. This is the basis for the optimization process. To achieve a global evaluation, the following aspects must be analyzed:

- Fragmentation and swelling of the muckpile.
- Geometry, height and displacement of the muck pile.
- State of the remaining rock and bench floor.
- Presence of boulders in the pile.
- Vibrations, fly rock and airblast produced by the blast.
Fragmentation and Swelling of the Muckpile

Apart from the classification of size distribution or screening of the muckpile in treatment plants, there is no method which enables a quantitative evaluation of fragmentation in conditions that would be trustworthy. Size distribution is the basic tool within the optimization process of blasting, as it is the only means of comparing the fragmentation obtained when a study is to be done on the sensitivity of the design parameters. Due to high costs and necessary time to achieve the complete size distribution curve, in mining operations the following approximate methods are used:

- Qualitative visual analysis
- Photographic methods
- Photogrammetry methods
- High-speed photography
- Study of loading equipment productivity
- Volume of material that requires secondary blasting
- Bridging delays at the crusher
- Partial screening
- Image analysis by computer.

Qualitative visual analysis

This is the most widely used and usually the only analysis applied. The muckpile and the general aspect of the blast is observed immediately afterwards and the engineer responsible for the evaluation makes a subjective assessment. However, changes in fragmentation can only be distinguished when the differences are great, even if the person in charge has vast experience. The application of this technique is not very precise; it does not allow an exact distribution of the sizes and very frequently there is no written report on the results. In general, it is only good for an initial contact with the blast results so that the specialists can later make a complete study.

Photographic method

This technique has been used in various ways and methods of analysis. Some researchers used photographs of the muckpile and chose at random around 15% of the total to analyze sizes and number of the fragments with a superimposed grid. After studying five blasts under similar conditions, the results showed a deviation of more or less 9.6% in the mean fragment size.

The most important source of error was in assuming that the surface fragment distribution was representative of the total mass. Reid used a series of photographs of the muckpile at different moments of digging, placing scaled targets in the field of vision for dimensional purposes, as seen in Figure 30.
Figure 30: Gird method of evaluating fragmentation and field environment

This system is one of the most useful and also gives a graphic documentation for analysis and comparison of different rounds. The only inconveniences are:

- Time consuming in preparation and study, and
- It is difficult to quantify the small sizes.

**Photogrammetric method**

The photogrammetric methods give more precision than in conventional photographs. As the investment in equipment and accessories is relatively high, it is only used as a complement to the main applications such as topographic control of the cuts and muckpiles, geological studies, etc.

The advantage of the photogrammetry is that it allows a tri-dimensional study of the muckpile, thereby aiding in the calculation of each fragment size and the volume and swelling of the pile. The drawbacks, apart from cost, are the need for qualified personnel to use and interpret the method.

**High-speed photography**

The use of high-speed photography in the evaluation, design and, above all, control of the blasting has been widely undertaken. Some have considered it as a technique for the evaluation of fragmentation. The main problem is that the gases and dust in the environment obscure the vision of crack formation and muckpile displacement.
The information taken from the analysis of the high speed photographs can be classified as:

- **Qualitative:**
  - First rock movements
  - Confinement of the stemming
  - Trajectory of the muckpile movement.

- **Quantitative:**
  - Exit time of the blast accessories
  - Time and efficiency of the gas confinement
  - Acceleration, direction and velocity of the fragments
  - Velocity of stemming ejection
  - Projection and displacement of the muckpile.

The study of these data, along with detailed information of the design parameters of the blast and of the whole of the operation, is very useful for the detection and definition of:

- The existence of misfires and their causes
- Incorrect explosive charges
- The effect of sub-drilling, of the presence of water and of the stemming
- Determination of the best initiation sequence
- Yield of the chosen initiation system
- Global movement of the muckpile
- Source of oversize at the face
- Muckpile displacement and
- Profile geometry of the muckpile.
Figure 31: High speed camera bench blast
Digital processing of images

Computer technology has opened the door to image analysis of the evaluation of muckpile fragmentation. The modern methods of image analysis quantify geometric aspects with images in two dimensions, such as the area, number, perimeter, shape, size and orientation. The procedures take in the following stages:

1. Image input. The image is captured by a camera, usually video, and subjected to an automatic digitization process. This means the conversion of the optic image to a digital format of picture points pixels, and giving each one a certain brightness, or grey level value from 0 (black) to 255 (white).

2. Scaling. The scale of the image is defined, normally using a marker placed on the muckpile as a reference.

3. Image enhancement. This stage uses digital filters which permit an enhanced viewing of the fragments. For example, low pass or Gaussian filters which eliminate noise, a shading filter which corrects illumination defects, etc.

4. Image segmentation. At this stage, the fragments are separated from the rest of the background to produce a binary (black and white) image. A grey level is defined, then the pixels with values above this level are turned to white (fragments), and they will be taken into account whereas the pixels below this level will be darker (background) and are turned to black.

5. Binary image manipulation. The segmentation process is never perfect, as the contours of some segments will bridge together and others will be confused with the background. To correct this, an iterative process of dilation, erosion and line thinning is used.

6. Measurement. This system, after identifying each object in the binary image as an independent fragment, measures the diameter of an equivalent area circle and classifies them.

7. Stereometric interpretation. In this step the distribution of two dimensional size is converted into volumetric sizes or tri-dimensional. This conversion demands the use of stereometric principles along with some empirical relationships.

Studies of loading equipment productivity

This technique of fragmentation evaluation is based upon the assumption that the digging rates are an inverse function of the coarseness of the muckpile and a direct function of the swelling of the same. The presence of oversize, reduced swelling and poor toe condition will be immediately reflected in productivity. If the technique is applied correctly, a precise evaluation can be obtained.

The lost time that is not directly related to the condition of the muckpile, such as waiting for the trucks, mechanical breakdowns, shifting the shovel or clean-up operations, should all be taken into account. The studies should be
made with the same machines and operators to eliminate the experience factor or erroneous estimates.

Studies have been undertaken to monitor the continuous current motors of the rope shovels, recording the signals on magnetic tape and processing the data on the computer. The call for extra power can be caused by the following:

- Defective floor breakage with toe appearance.
- Insufficient swelling of the muckpile.
- Excessive displacement of the pile, and
- Coarse size distribution.

**Boulder count and secondary breakage**

Any fragments of rock produced by blasting that cannot be handled by the mining equipment is referred to as boulders or oversize. The sizes of these blocks depend upon each operation and during the same should be set aside for fragmentation, with the procedures described in the following chapter.

The relative volume of the oversize should be maintained at a minimum, not only because of the high fragmentation cost, but because they affect the whole of the operation by giving low digging yield due to lost time dedicated to taking them aside and also delays at the crusher.

**Bridging delays at the crusher**

The production of any crusher depends basically upon the coarseness of the material that enters, and indirectly this can be an index of fragmentation taken from the yield and the energy consumed per treated ton. The interruptions at the crusher due to oversize should be controlled, as well as the wear of the steel plates.

**Screening**

This is the only precise method of quantitative fragmentation evaluation. In small operations this technique can be used with representative samples, but in large open pit mines, it would be impractical because of the high cost and time involved.

**Geometry of Muckpile, Its Height and Displacement**

The configuration of the muckpile is governed by:

- The geometric design parameters: height of bench, blasthole inclination, burden, spacing and stemming.
- Powder factors.
- Initiation sequences and delay timing.
The optimum geometry depends upon, in each case, the digging and haulage system used.

- **Profile I** represents the ideal situation for front end loaders, but if the available equipment are rope shovels, the yield will be low and time consuming in collecting the material and cleaning up around the shovel.

- **Profile 2** requires minimum cleaning labor and the productivity is high, but there can be safety problems involved for the operators due to the fall of rock from great height.
  - Excessive clean up area, low productivity for rope shovel, high safety conditions.
  - Low clean up area, high productivity, dangerous.
  - Low clean up area, good productivity, safe.

- **Profile 3** reflects optimum conditions for rope shovels. This control procedure can be done with topographic equipment, aided by transversal photographs.

**Figure 32: Different geometries of the muckpile**

**Condition of the Remaining Mass**

Once the muck pile has been loaded, it is possible to observe if there has been overbreak, and the quantity of damage to the remaining mass. The evaluation of the damage caused by the blasts to the remaining rock can be carried out by any of the methods of geomechanic characterization of the rock mass, but for what is wanted, the system in table below is one of the most used for its simplicity and pragmatism.
### Analysis of the Bench Floor

Once the blast has been evacuated, the bench floor can show the following:

- Toe in front of holes,
- Toe between holes,
- High floor,
- Low floor.

The toe between blastholes is due to overspacing and this should be decreased. The appearance of a high floor systematically could be due to a plane of weakness or to insufficient subdrilling and bottom charge. When the floor level is lower than was designed, the bottom charge and subdrilling should be diminished and the loading operation observed.

### Boulders

Large sized blocks can appear in the muckpile in the following areas:

- On the top or crest,
- On the floor (usually occurs with toe problems),
- Within the pile, and
- In front of the pile.

The boulders on top could be caused by a hard level or poor fragmentation of the top part of the bench. This is corrected by increasing the explosive column or by placing a small pocket charge in the stemming. The boulders at floor level are due to a weakness plane in the lower part of the bench. They are eliminated by the same method as for toes: decreasing the burden and increasing the bottom charge, the subdrilling and the between-row delay.
The boulders in the interior of the pile are due to incorrect drilling, a poor blast yield because of defective priming or an alteration of the explosive such as local dampening of ANFO, and to a pernicious effect of the inner joints with respect to the opening of the grid. The boulders in front of the pile could originate in an excessive breakage of the previous blast in the same zone.

**Vibrations and Airblast**

If the delay timing and the initiation sequence of a blasting are not adequate, the following results will come about:

- Poor fragmentation and insufficient swelling of the muckpile
- Uncontrolled fly rock
- High vibration levels, and
- Low frequency vibrations.

By analyzing the vibrations registered with a seismograph and later modifying the indicated parameters, a better use of the explosive energy can be obtained, giving a better size distribution of the muckpile and lower vibration intensity and frequency, which are potentially less dangerous.

As to airblast, there are different mechanisms that cause it, but one of the most important is a premature escaping of the gases to the atmosphere after stemming ejection. For this reason, the studies with high-speed cameras can help define the optimum stemming height and the ideal type of material so as to achieve a larger confinement of the explosive without negatively affecting the fragmentation in the zone where the inert material is placed.

**Blast Evaluation in Underground Mining**

In underground mining, characterizing the quality of the resulting excavation is a key method of evaluating the blast results, especially for development. Overbreak is commonly calculated by comparing the planned excavation with the resulting excavation. Hard toes, bootlegs, and oversize is also a common method of characterizing the quality of a blast. Dilution, a factor of both the design and execution of a blast may also be used as a sign of blast evaluation. In certain methods, such as VCR, cratering at the top of the blasthole is a common indicator of a lack of stemming. Damage to ventpipe or other utilities can also be a sign of inadequate stemming or improper blast preparation procedures.

**ADDITIONAL RESOURCES XVI**

Additional learning resources include:
- Lecture Module 5.5 – Blast Results (mandatory)
Module 5: Mechanical Excavation

This module on mechanical excavation considers the mechanical excavation basics and theoretical aspects of bits for excavation. Mechanical excavation can be considered the removal of rock from in-situ by mechanical means. Several groups of equipment will be considered in this section including continuous mining machines, roadheaders, longwalls, and rippers.

5.1 Mechanical Excavation Basics

The two main types of mechanical breakage for production purposes include indenters and drag bits (also known as picks). Both are considered to break the rock in the primary breaking process. The main difference between indenters and drag bits is that an indenter breaks rock by applying a force that is predominantly in a direction normal to the rock surface. Comparatively, a sharp drag bit applies the main force in a direction predominantly parallel to the rock surface. The breaking mechanism for both is actually a tensile fracture.

Intenders are used more widely than drag bits primarily due to the shear loading of the rock by the drag bit. The cutting edge of a drag bit is subjected to bending resulting in tensile stresses along the bit edge. As most cutting tools are made of tungsten carbide, a brittle material weak in tension, this makes the possibility of catastrophic failure likely. However the tools can be mounted in a fashion so that the bit contacts with the rock in a mainly compressive fashion, as seen in the Figure 5-1.

Figure 5-1: Longwall shearing machine whose cutting bits attack the rock in compression

Various theoretical models have been developed that can compare drag bits with indenters. Several will be presented with an emphasis on comparison between drag bits and indenters. This will allow the two modes of the primary breakage process to be compared. Various theoretical models have been developed as presented in the SME Mining Handbook (Chapter 9.1) yet few have admittedly been proven empirically. Furthermore, results from lab-based rock cutting experiments worldwide have reported variances in cutting conditions and variables such as pick shape and rock type. However, generalities do emerge from these experiments and are regarded as fundamental features of rock pick cutting.
Important variables in understanding these generalities include:

\[ F_C = \text{Mean force} \]
\[ F_N = \text{Peak force} \]
\[ E_S = \text{Specific Energy (relates to the cutting force to amount of rock produced)} \]

Various picks have been tested, such as; chisel, round bottom, v-front, v-bottom, and pointed. However, these many picks can be categorized into three main groups (as seen in Figure 5-2: Pick Shapes);

- Pointed tool.
- Simple chisel,
- Round bottomed tool,

![Figure 5-2: Pick Shapes - Pointed tool, simple chisel, and pointed tool. (\( \beta = \text{Back clearance angle}, \ \alpha = \text{rake angle} \)](image)

Both \( F_C \) and \( F_N \) increase with \( d \) (depth of cut) for all pick shapes. From Figure 5-3, it can be seen that this relationship is approximately linear. (not linear when using wide picks at depths much less than their width but machines such as these do not exist.)
From Figure 5-4, it can be seen that Specific energy varies inversely with depth of cut for all pick shapes, therefore:

\[ E_s = \frac{\text{constant}}{d} \]
Cutting and normal forces decrease monotonically with increasing rake angle as seen in Figure 5-5. Most of the benefit to pick forces has been achieved at a rake angle of 20°, beyond which further marginal improvement is at an increasing penalty to pick strength and its potential to survive.

![Figure 5-5: Mean force vs. rake angle (F\textsubscript{c} vs. \phi)](image)

Figure 5-5 shows that increasing back clearance angle reduces pick forces up to about 5°, beyond which forces are independent of this angle.

![Figure 5-6: Mean force vs. back clearance angle (F\textsubscript{c} vs. \beta)](image)

---

9 monotonic defined for mathematics: Designating sequences, the successive members of which either consistently increase or decrease but do not oscillate in relative value. Each member of a monotone increasing sequence is greater than or equal to the preceding member; each member of a monotone decreasing sequence is less than or equal to the preceding member.
Higher productivity will be achieved by cutting slowly and increasing depth than when cutting at a higher speed yet more shallow cut since:

- Cutting speeds of up to 5 m/s and beyond have no discernible affect on pick forces or specific energy.
- Hence speed has no direct effect on pick forces
- Cutting efficiency improves with cutting depth

For a chisel shaped pick, $F_C$ and $F_N$ increase linearly with pick width as seen in Figure 5-7.

![Figure 5-7: Mean force vs. pick width ($F_C$ vs. w)](image)

Pick forces increase with rock strength, and it has yet to be determined what characteristic of rock strength is the predominant influence. Figure 5-8 compares the mean cutting force with both the tensile and compressive strength of various rocks. A further issue related to cutting forces is the degree of rock saturation as increased saturation results in a decreasing mechanical strength of the rock (the reason for this is not clearly understood).
In terms of pick shape, when operating at the same rake and clearance angles and depth of cut, the pointed pick requires the least cutting and normal force. The chisel pick requires the greatest forces. However, considering that the chisel pick cuts a considerably larger volume of rock than the other two shapes, it cuts with the lowest specific energy and therefore is the most efficient shape (pointed pick least efficient). Due to the increased penetrating capability of the pointed pick, for a given available normal force, pointed picks operate more efficiently than the chisel bit. (Hence, pointed picks can but deeper for a given level of force, whereas chisel bit cut more material for a given depth of penetration. Operationally, picks are better.)
5.1.1 Synopsis

Drag bits are an efficient means of breaking rock. Although most are unable to survive in the hard-rock environment, they require substantially less energy than other mechanical cutters. Two main problems remain to be solved:

- Drag bit materials that have far improved wear resistance and strength
- Machines to provide sufficient thrust force to keep drag bits at an effective depth of cut, especially as bit wear proceeds.

**ADDITIONAL RESOURCES XVII**

The above information is additionally supplemented by:

- Readings – Chapter 9.1 – Mining Engineering Handbook. (Mandatory, but this is for modules 1 and 2)
- Lecture module 2.1. (mandatory)

5.2 Longwall

These lecture notes review the design and operational aspect of the longwall mining method. Particular focus will be on the cutting mechanism and design. Important issues related to longwall, such as conveyor design, ventilation and roof support, are left to materials handling and rock mechanics courses.

Longwall mining has one of the simplest layouts, provides continuous production and can provide a safer work environment. The panel layout is conducive to good ventilation. Therefore this method is considered to be better than room and pillar methods used to mine similar coal deposits. Since this system uses full caving, fewer residual pillars remain hence coal recovery is higher and surface subsidence is relatively uniform and complete.

Longwalls are used to mine flat horizontal coal seams of thicknesses ranging from 1.1 to 4 meters. The depth of overburden can range from anywhere between 60 to 820 meters. Figure 5-10 shows a typical plan view of a longwall layout and the nomenclature of the key design variables. Longwall panels are flanked by panel entries on both sides of the main entries, typically excavated by a continuous mining machine. The immediate entries on both sides of the panel are called the head entry and tail entry. The head entry is used for air intake and the transportation of coal on conveyors, personnel, and supplies. The tail entry is used as the air exhaust. Panel sizes and length are generally determined by:

- Experience
- size and shape of the coal seam
- geologic conditions
- location of surface structures (buildings can subside horizontally when in the middle of a panel but would subside differentially if it straddles a pillar),
- capacities of the transportation system
- ventilation
- power capacity (of equipment)

Panel widths vary between 120 to 293 meters (centre to centre) and have lengths between 1220 to 1830 meters. The economic factors pushing for longer and wider longwalls include:
- Reduction of the development cost as fewer panel entries would be required
- Increases in recovery and recovery rate (few longwall assemblies, downtime between panels)

Other issues that begin to surface in longwalls include:
- Downtime (maintenance & repair) in large panels
- Capital cost of equipment in a single panel

Figure 5-10: Typical Longwall retreat method
5.2.1 Mining System Description

Figure 5-11 shows a diagram of a coal operation. Coal at the face is cut by a shearer or plough and loaded onto an armoured flexible conveyor (AFC) and conveyed to the head entry T- junction. The coal is loaded onto a stage loader which empties onto the entry belt conveyor. Powered sectionals supports are used to support the roof along the whole face. The AFC and powered supports are advanced hydraulically after each cutting cycle of the shear and the roof behind the supports are allowed to cave. The area between the rear edge of the supports and faceline is called the face area while behind the supports is called the gob. Panel entries are where the regular conveyors are maintained by roof bolting or other methods. However, at the tail entry where ventilation circuits are set up, cribs are usually required to strengthen the support.

![Figure 5-11: Longwall shearing system](image)

Technical factors of longwall mining technology include:

- The longwall width is limited to the power and structural strength of the face conveyor.
- Increasing panel width increases roof exposure time (when the shearer or plow moves across the face, an area of roof is opened. This small area may cave prior to the supports moving forward to secure the back).

Now that the basics of longwall mining have been established, a focus on the actual rock excavation aspects will be explored in more detail.
5.2.2 Selection of Cutting Machine

The two main types of longwall machine are the shear loader and the plough. Shearers are used more widely in the US. Ploughs are simply a large blade that is thrust into the rock and dragged across the face resulting in the removal of a slice of coal. Key variables in the selection of the type of cutting machine are:

- Mining height
- Seam structure
- Roof bonding strength of coal
- Cutting/ploughing resistance of coal.

Since shearers are the most common machine, only shearers are discussed. Major parameters in shear design are:

- Type of shearer,
- Dimensions,
- Haulage speed,
- Power capacity.

There are several types of shearers:

- Double-ended ranging-drum (DERD): two shearing drums mounted on cutter, articulated so that coal seams of varying height can be mined. Can mine thicknesses of 1.47 – 3.96 meters. Is by far the most employed shearer.
- Single-ended ranging-drum (SERD) for thicknesses of 1.52 meters.
- Single-ended fixed drum (SEFD) for thicknesses of 1.26 to 1.37 meters

5.2.3 Dimensions of Shearer

Equipment suppliers provide various models with varying dimensions and power available. The key variables in selecting the shearer can be seen in Figure 5-12:

![Figure 5-12: Shearer dimensions](image)

Where:

\[ H_c = H_b - \frac{B}{2} + L_a \sin \alpha + \frac{D}{2} \]

And where:

- \( H_c \) - mining height
- \( D \) - diameter of the drum
\( H_b \) – height of ranging arm  
\( L_a \) – length of ranging arm  
\( B \) - body depth  
\( \alpha \) – angle of the ranging arm  
The maximum cutting height should not exceed 2 D.

A nomograph for determining shearer dimensions and mining height is shown in Figure 5-13. Note that the conversion factor is 1 inch = 25.4 mm.

**Figure 5-13: Nomograph for shearer dimensioning**
Example 1: Shearer dimensioning
Use the nomograph in Figure 5-13 to select the drum diameter $D$, given the mining height and major dimension of a shearer: where: $H_c = 90$ in. (2.29m), $H_b = 50$ in (1270mm), $L_a = 70$ in (1778mm), $B = 20$ in. (508mm) and $\alpha = 30^\circ$

The dotted line on the nomograph can be followed from point a through to f in Figure 5-13. This results in a value of $D$ being 30 inches (762 mm). In practice, in order to reduce the loading resistance of the drum and to increase loading efficiency, the drum diameter is selected larger than the minimum value, typically 75-80% of the mining height, in this example, $D$ would therefore be equal to 60 inches.

ADDITIONAL RESOURCES XVIII
The above information is additionally supplemented by:
- Lecture for module 2.2 – Longwall (mandatory)
- Readings – Chapter 20.1 – Longwall Mining, from SME handbook (optional)

5.3 Continuous Mining Machines\footnote{These notes were assembled directly from the following references:}
Module 5.3 will cover underground continuous mechanical mining technology and, continuous surface mining equipment. These notes were assembled directly from the following references:
- Stefanko, Robert. Coal Mining Technology Theory and Practice. Littleton CO.: Society of Mining Engineers. 1983
- Mining-Technology.com, search: continuous mining
5.3.1 Underground continuous mining machines

Underground continuous mining machines typically operate in room and pillar mines in coal, salt, Trona, or potash (essentially soft material in horizontal orebodies). The machines are an alternative to drilling and blasting operations. Underground blasting however, is undesirable in coal operations due to the risk of secondary explosions caused by suspended coal dust or methane gas. Other advantages of continuous mining is combining the operations of cutting, drilling, blasting, and loading into one operation by a single machine. The continuous nature of this mining method eliminates the problems involved in the cycling of equipment. As the number of equipment is reduced, so are manpower requirements. However, these advantages are offset in some part by concentrating the reliance of production on a few headings. Maintenance issues therefore become more critical.

Continuous mining equipment has evolved significantly over the last 100 years. While most of this equipment attack the solid face, ripping out and loading the coal in one step, sometimes using very different processes. Only those machines still in use today are reviewed.

5.3.1.1 Boring Machines

Figure 5-14 shows a boring machine. These machines have rotating arms equipped with bits that bore out the coal. The boring machine has a cutting chain on the bottom that creates a flat working bottom and an upper trimming chain, which when working together, produce a roof configuration that is arched or ovaloidal opening. The arms revolve relatively slowly, produce a much coarser product than that of a ripper miner, minimizing gas and dust problems. Since the machine uses its tracks to provide the thrust, it cannot be used to bolt simultaneously as advancing. The machine can have drills mounted on a slide carriages and have met with success at bolting and mining simultaneously.
The main disadvantage of this type of machine is its size. It typically occupies the entire stope. The tips of the arms can be extended or contracted hydraulically over certain limits to provide some machine clearance during tramming, however, the machine must move considerably slowly. Ventilation is also an issue when dealing with its size; therefore these machines are typically used in potash mines, not coal mines. Further discussions on potash mining will be discussed later.

Figure 5-15: Milling head continuous mining machine

5.3.1.2 Milling head miner
Milling head miners are the most popular continuous miner in operation in US coal mines. The machine is a ripper with disks or wheels located on arms. The disks rotate in a vertical plane. To mine the coal between the disks, the arms to which the disks are attached oscillate or are provided with a side splay or horizontal movement. While the machine can be equipped with a full-face head about 4.6 m (15 ft) wide, usually the width of the head is less so that two passes are required to mine the width of a room or entry.
The usual mode of operation is to push the machine into the face on its crawler while the head is in an elevated position and the disks are rotating. The milling machine sumps (pushes) into the top and shears downward. Loading arms located under the head load the coal. While the basic machine had an oscillating feature when it was introduced, most miners use the same bit attack with a rigid head and the machine is usually referred to as the Hardhead miner. In this type, bits are spirally wound on a fixed shaft so that each bit cuts an infinite number of vertical planes during a single revolution, accomplishing the same purpose without oscillating and with subsequently fewer maintenance problems. This machine is cheaper as well as requiring less maintenance, it is, therefore, quite popular today. In addition, its head can be tapered to provide an arched opening where desired, although the optimum shape with regard to ground control should be heeded. The usual procedure is to mine an opening cut the width of the machine [usually 3 m (10 ft)] then mine the other side of the face. Since this machine cuts by advancing the crawler to the face, drills for roof-bolting would have to be attached to the chassis in a flexible manner, negating its widespread use for concurrent bolting. However, some equipment allows both bolting and mining simultaneously.

5.3.1.3 Boom-Type Miner

A boom-type miner or road-header consists of a rotary cutter-head mounted on an articulated boom, a crawler assembly, and a gathering arm system. The design of the machine permits a concentration of power at a single cutting bit at a time, enabling it to cut harder rock than drum-type continuous miners can excavate. While it is used for mining coal and other minerals, its primary use is for driving mine entries and tunnels in civil engineering projects.

Roadheaders were introduced in the post World War II era, and at present there are over 2800 employed worldwide, with the majority outside the US.
However, about 140 road-headers are in use in North America with approximately 75% of the machines working in mines. Roadheaders can excavate any size and shape of cross section: circular, horseshoe, arched, and rectangular. Unlike conventional drilling and blasting, the smooth cutting action of the cutterhead does not weaken the rock due to overbreak. Thus lighter support may be used or less concrete, aiding cost. The machine is being used to enlarge mine entries in a bituminous mine in central Pennsylvania by removing bottom rock. The wide flexibility in cutting at various angles also increases the utility of the roadheader when compared to the continuous miner, as seen in Figure 5-17.

Figure 5-17: Mining of Steep seams with roadheader and miller head (drum-type) miners
Figure 5-18: Mining in variable height seams

Figure 5-19: Double head Roadheader
5.3.1.4 Performance Analysis

Predicting the cutting rate of these machines requires information on:
- rock properties, both in terms of the rock and rock mass;
- machine properties, including rock-tool interaction models.

A few key measures include:
- **ICR**: instantaneous cutting rate
- **OCR**: operational cutting rate (includes utilization)

Typically, the OCR is 0.45–0.60 x ICR, while 0.30(ICR) when trimming

Utilization is affected by:
- Support installation
- Surveying
- Pick replacement,
- Maintenance/repairs
- Haulage delays
- Shift changes

In the calculation of utilization, boom repositioning and final profiling are not included.

Advance rate is calculated by:

\[
\text{Advance rate} = \frac{\text{OCR} \times \text{utilization} \times \text{Penetration rate}}{\text{face area}}
\]

Be warned: theoretical cutting models for roadheaders vary in applicability for machine types, and are not good. TBM models work better, however, TBM performance remains to be calculated empirically, as before:

SE=specific energy
HP=headpower
Where: SE = HP / ICR

Most predictions use RMR, RQD and UCS.

The rock cuttability index, in kg/cm$^2 = \sigma_c \frac{RQD^{7/5}}{100}$; can be used in conjunction with Figure 5-21.

![Figure 5-21: Rock Mass Cuttability Index vs. cutting rate](image)

Note that Figure 5-21 can also be used for predicting road-header productivity as a function of rock mass properties.
5.3.1.5 Tunnel Boring Machine

This type of machine can be used to drive circular tunnels from 5.7 ft (1.75 m) to more than 36 ft (11 m) in diameter in rock types that range from weak, loosely consolidated to very strong and abrasive. In almost all cases breakage is effectuated by roller cutters mounted on the cutting head, as seen in Figure 5-23. Because these cutters break the rock by indentation, these machines are characterized by very high thrust requirements. This thrust is provided by hydraulic rams that press the cutterhead into the rock face. The thrust reaction force is reacted through gripper pads that are pressed hydraulically against the tunnel walls. The rock broken from the face by the cutters falls to the floor where it is scooped into buckets mounted around the gage of the cutterhead. This debris is lifted in the buckets to the tunnel crown, whereupon it is tipped onto a belt conveyor that runs through the center of the machine.
The most common type of cutting tool employed on these machines is the disk cutter. In some cases the cutting edge of this tool is a hardened steel surface and in other cases it is a row of cemented tungsten carbide buttons that are press-fitted into the disk rim. The most common cross section for a
hardened steel disk cutter is seen in Figure 5-26. The advantage of this design is that the tool area presented to the face is maintained resulting in relatively constant penetration rate. As the cutters become blunted, it is necessary to increase both the machine thrust force and torque. Other cutter types include the kerf cutter, which is simply multiple disks mounted on the same hub, and the pineapple cutter, which is a frustrum with cemented tungsten carbide buttons press-fitted onto the surface.

Figure 5-26: cross section of disc cutter

Figure 5-27: Kerf and Pineapple cutters

A ranking of cutting efficiency of these tool types, in terms of specific energy, places the steel disk cutter as the most efficient, the disk-button cutter next, and the pineapple cutter as least efficient. However, the wear resistance, and therefore the capability of cutting strong abrasive formations, is the reverse of this efficiency ranking. Consequently, steel disks tend to be used for cutting weaker, less abrasive rocks, and pineapple cutters are used for machining the most abrasive and toughest formations. Several other comments can be made about the cutting behavior of disk cutters. First, in contrast with drag bits, the efficiency of the rock breakage process does not decrease when disk cutters are used in a groove deepening (when multiple passes are made by a tool taking a series of shallow cuts in a kerf before producing major rock chips) mode. This is fortunate because in practice, groove deepening is the cutting procedure most commonly employed with tunneling and with other boring machines. Second, similar to the findings for drag bits, an optimum spacing exists between an array of disk cutters working a rock face. The value of this optimum spacing depends on the depth of cut taken and on the rock type. However, whereas with drag bits an optimum s/d value of 2 to 3 is typical (see Figure 5-28), with disk cutters this value is more typically in the range 5 to 10. Third, the efficiency of the rock
breakage process is independent of whether the grooves are cut simultaneously, with multiple disks on a single hub, or sequentially, with independent disks.

![Spacing vs. Depth Relationship](image)

*Figure 5-28: Spacing versus depth relationship*

Skidding has a major influence on cutter longevity (more will reduce longevity). This dragging or ploughing action is particularly acute for rollers as the center of the cutting head.

TBMs are widely used in civil applications. The primary advantage is that this machine is quasi-continuous, compared with the drill-blast-muck cyclical method. Therefore tunnel rates are somewhat more constant and higher where TBMs are used. Roughly, the speed of advance for a drill-blast-muck crew would be 10 feet per shift (ideal), whereas on a TBM, the advance rate would be about 33 feet per shift. A key environmental aspect is the lack of ground vibrations when compared to blasting. A further considerable advantage is the lack of secondary damage caused by the machine when compared to the damage cause by blasting. The disadvantages are:

- High capital costs
- Tunnel cross section is circular
- Large turning radius (100 meters)
- Cumbersome machine and system installation.

Considering these disadvantages, a rule of thumb is that a tunnel of at least 2 km in length to justify the cost of a machine setup. The lack of flexibility, size, and cost of the equipment has made TBM not fully accepted in mining. Some attempts have been made to develop hardrock continuous mining machines that are more flexible, however, the bit wear costs remain prohibitive. A prototype oscillating disc cutter appears to show promise but has yet to be fully developed into a working production unit.
The TBM is considered to be the primary equipment in the field of rapid excavation. Progress in rapid excavation can be measured by several performance parameters: (1) hardness of the rock, (2) time percentage of machine availability, (3) diversity of application, (4) rate of advance, (5) specific excavation rate, and (6) cost of advance. Several of these parameters deserve to be discussed further here. First, hardness of rock is an area in which some progress is being made. Tunneling through rocks up to 30,000 psi (207 MPa) has become fairly common; however, some tunnels have been driven through rock approaching 50,000 psi (350 MPa) in compressive strength.

TBM specific excavation rates have also gone up over the period of rapid excavation usage. Case studies dating back to 1980 have resulted in empirical tables of rock types with corresponding excavation rate. Note that the metamorphic, granitic, and volcanic rocks have TBM specific excavation rates of 8 to 360 $\times 10^{-6}$ ft$^3$ of rock extracted per lb of force per cutter per revolution ($0.00005$ to $0.0023$ m$^3$ of rock extracted per kN per cutter per revolution) and the rates in sedimentary rocks are 8 to 720 $\times 10^{-6}$ ft$^3$ of rock per lb of force per cutter per revolution ($0.00005$ to $0.0046$ m$^3$ of rock per kN per cutter per revolution).

![Figure 5-29: Empirically derived excavation rates of TBMs by diameter for igneous and metamorphic rock types](image-url)
Although improvements have been made in these parameters, machine availability is still a problem area. Studies have shown that TBM availability is 35 to 50%. There are many good reasons that the TBM is not engaged in the excavation process a greater percentage of the time. Maintenance, backup equipment, ground control, cutter replacement, and other delays all contribute to the problem.

One of the major deterrents to applying rapid excavation in underground mining is the rather massive dimensions of the typical TBM with its trailing gear. This is evident in the application of a TBM to development in an underground mine (Stillwater). In this project the mining company had to work with a TBM manufacturer to reduce the turning radius of its TBM to about 200 ft (60 m) in order to effectively use the device in mine development. The original turning radius of the machine (350 ft or 106 m) was simply too great to be able to effectively maneuver the machine underground.

Another important limitation of the TBM in mining projects is the economics of conventional versus rapid excavation development, see Figure 5-31). For tunnels of less than about 22,000 ft (6 km), the TBM normally cannot provide a cost that is lower than the costs of conventional development practices. This eliminates its use for many mine development projects.
Figure 5-31 Cost comparison between tunnel boring and conventional drill and blast.

Although TBMs have been used for development at the Stillwater Complex in Montana and at the Ray Mine in Arizona, the rapid excavation revolution has still not established this process as commonplace in providing horizontal development openings in mines. In the advance of vertical openings, rapid excavation has greatly improved mine development and civil works, particularly in the area of raise boring. In many mines, raise borers for development of stoping operations are routinely used and have replaced conventional development in all but unusual circumstances.

### 5.3.1.6 Raise / Blindhole / Shaft Boring Machines

These machines use the same button roller cutter technology as TBMs, but they drive vertical or steeply inclined holes rather than tunnels.
Raise Borers are machines used to produce a circular excavation either between two existing levels in an underground mine or between the surface and an existing level. The process of raise boring is shown in Figure 5-32. First, the boring machine is set up on the upper level, and a small-diameter (of the order of 9 in. or 230 mm) pilot hole is drilled, usually with a tricone bit, down to the lower level. When this hole is completed, the drillbit is removed, at the lower level, and replaced by a reamer head having a diameter with the same dimension as the desired excavation. Some type of roller cutters are mounted on the reamer head. This head then is rotated and pulled back up towards the machine. The rock debris falls by gravity into the lower excavation where it is removed. These machines are very effective in driving raises, and they have become very popular, particularly in hard-rock underground mines. Frequently the direct costs of driving a raise, in terms of dollars per foot (dollars per meter), are reduced by using these machines. In addition, however, this raising system offers other significant advantages, such as:

- safety—conventional drill-and-blast raising is notoriously dangerous.
- improved excavation rates
- improved productivity.
circular shape combined with the lack of blasting damage results in an excavation of greater strength and integrity than a hand-driven raise. Reduced friction for ventilation raises.

Applications of raise boring are in both civil and mining environments, specific applications are summarized in Table 5-1.

**Table 5-1: Applications for boring**

<table>
<thead>
<tr>
<th>Mining</th>
<th>Civil</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Material Transfers (ore or rock passes)</td>
<td>1. Penstocks and surcharge chambers in hydroelectric projects</td>
</tr>
<tr>
<td>2. Ventilation</td>
<td>2. Redirection and retrieval of water in hydroelectric projects</td>
</tr>
<tr>
<td>3. Personnel access</td>
<td>3. Storage of petroleum, pressurized gas, and nuclear waste</td>
</tr>
<tr>
<td>4. Ore production</td>
<td>4. Road and rail tunnel ventilation</td>
</tr>
<tr>
<td>5. Slots for longhole</td>
<td>5. Storm water storage and drainage</td>
</tr>
<tr>
<td></td>
<td>6. Equipment access (pipes, hoses, cables)</td>
</tr>
<tr>
<td></td>
<td>7. Water inlets and outlets for fish farms.</td>
</tr>
</tbody>
</table>

**Figure 5-34: Raise boring diagram – Up and downward boring.**

Blindhole Borers are machines that will produce a circular excavation without the need for a pilot hole. The machine is set up on a level in an underground mine, and a steeply inclined hole, generally several feet (meters) in diameter, is excavated upward from this level.
Boring Site Preparation.
Correct site preparation eliminates major delays and adds noticeable efficiencies to the raise boring operation. Raise boring site preparation begins with a comprehensive plan of the site layout. Site planning considerations must include:

- Derrick mounting systems
- Selection of bailing fluid
- Bailing fluid and cuttings discharge
- Storage and positioning of drill string components
- Overhead clearances
- Floor space and equipment positioning
- Compressed Air
- Water
- Electric power
- Lighting
• Communications
• Ventilation.

**Horizontal boring**
Use of horizontal raise boring in civil engineering projects is becoming increasingly popular as compared to alternatives since:

- no trench is to be dug
- no vibration or environmental side-effects of drilling and blasting
- less set-up time and cheaper than TBMs.

Combined with directional drilling (undertaken when drilling the pilot hole), a directional tunnel can be built.

![Figure 5-37: Directional boring in urban environment.](image)

**Figure 5-37: Directional boring in urban environment.**

![Figure 5-38: horizontal borehead](image)

**Figure 5-38: horizontal borehead**

**5.3.1.7 Underground mining case studies**
The first case study of an underground continuous mining system is exploration of the Boulby potash mine in the UK. Note that the shaft had to be excavated by freezing the surrounding rock mass so that a minimum of water inflow would occur. A special impermeable lining was then constructed
long the shaft in the section under the aquifer. This can be seen in Figure 5-39.

**Figure 5-39: Cross section of the Boubly potash mine strata**

To cope with greater stresses in the ore below 1,200m and to raise output above 3Mt/y by 2001, CPL converted its continuous miners to remote-controlled operation (so they can be safely used in shaly sections without prior blasting). The mine is currently establishing a mine-wide digital microseismic monitoring system and has devised a two-road with stubs (rather than four-road) stress-relieving room-and-pillar technique which achieves more mining per shift. Jeffrey and Joy continuous miners discharge to Joy electric shuttle cars, which run to feeder breakers on the main conveyors to the hoisting shaft.
Figure 5-40: remote operation allows distance mining and the operator to view from different angles.

Case Study: Rocanville Saskatchewan
Rocanville uses a long room-and-pillar method of mining. Ore is mined from rooms in three passes, separated by pillars supporting the overlying strata. Five automated Marietta two- and four-rotor continuous miners, each capable of extracting 650t/h of ore, form the production fleet. The run-of-mine ore is loaded on to extensible conveyors attached to the continuous miners. These connect to the main haulage conveyors, which move the ore to skip-loading pockets at the shafts, where it is hoisted to surface. In addition to automating its Marietta miners to measure ore grades and mine selectively, the company installed a central control room at Rocanville from which the entire mine, surface and underground, is controlled. The central control system, supplied by Allen-Bradley, oversees all aspects of the mine and mill operation, and has maximized ore extraction and recovery since it was commissioned.
Several types of cuts are made in mining potash. These can be seen in Figure 5-43, which shows various boring type cuts.
5.3.2 Continuous Surface Mining Technology

5.3.2.1 Surface Miners
Surface Miner combines the individual processes normally associated with traditional technology, such as drilling, blasting or tearing, extraction, transportation, crushing and loading. In particular, the fact that blasting is no longer required leads to a considerable reduction in environmental pollution from, gasses, dust, noise and flying rocks. Selective mining of thin or sloping layers results in a clear separation of material (overburden, minerals etc.). Eliminating dilution increases the efficiency of transportation and further processing. Surface Miners have reached a rated capacity of 1,250 bm³/h when engaged in lignite mining. This means that layers of up to approximately 1.2 m thickness can be mined in one step.
Surface Miners are capable of continuously mining stone with 50 MPa or 100 MPa, irrespective of discontinuities and without blasting. This includes:

- lignite
- coal
- anthracite
- bauxite
- phosphorite
- marl
- limestone
- hard sediment stone (sandstone, certain conglomerates, breccien
- clays, loam
Surface Miners are suitable for:

- Mining for the extraction of minerals and overburden
- Road construction for preparing roads, motorways and waterways
- Earthworks for preparing construction sites etc.

Figure 5-45: Surface miner ideal for thin strata

Figure 5-46: Cutting capacities for various Wirtgenamerica products

Figure 5-47: Surface miner ideal for defined thin vertical or flat seams
5.3.2.2  Bucket wheel excavators

Originally designed for relatively easy digging materials (gravel, sand, loam, marl, clays, and lignites), bucket wheel excavators (BWEs) can now dig in relatively hard material. These have included compact sediments such as shales, black coal, some limestones, and tar sands. One of the most successful applications of BWEs has been in German and Australian lignite mines. The machine digs out the material using a large wheel with buckets that revolve as the wheel turns. The teeth on the individual buckets are the primary ground engaging tools that break out the material from the ground. BWEs are most often attached to a conveyor network where waste material is sent to a spreader or to an ore stockpile.
BWE considerations:
- Hard consolidated materials, large boulders or blocky material cannot be handled
- Sticky material build up in buckets and can gum up the conveyor (although, with modern systems, sticky material can be handled);
• Abrasive material produces excessive wear on the teeth (some modern tooth design can significantly increase tooth life).
• The digging face should be stable
• Actual output is usually only 45-110% of theoretical
• Very limited flexibility (can be flexible in certain geological and equipment situations)
• High capital costs but may be the most economical method of mining weak flat tabular deposits.
• Some advantages in direct reclamation (environmental benefit);
• Relatively little manpower
• Must closely match downstream equipment (as in all mining operations)
• Low total costs
• Long life
• There must be a matched system linking BWE, side-slewage conveyor belt system and spreaders

Advantages of Continuous Excavators
• Have lower impact loading than comparable single-bucket machines
• Tends to reduce dynamic stresses, machine mass, maintenance costs, power consumption
• Reduced slewing (swing) speed, reduced digging impacts, reduction in ground bearing pressure
• BWE can mine both thin overburden and deeper overburden
• Conditions possible where single bucket (dragline) is impossible

Theoretical output of machine is:

\[ Q_{th} = \frac{60Fs}{\text{Swell Factor}} \]

where \( Q \) is theoretical output in yd\(^3\)/hr (or m\(^3\)/hr) bank,
\( F \) is capacity of single bucket
\( S \) is number of bucket discharges per minute
Swell factor is that of the material being excavated.

The theoretical output of BWEs range between 200 to 20,000 m\(^3\)/hr.

Largest disadvantage is their size, which limits flexibility. Therefore the mine must have strict discipline in terms of the mine plan, such as:
• Sufficient reserves to justify the capital cost
• Relatively horizontal stratification over a wide area of the deposit
• Uniform geologic conditions (i.e. absence of major faulting, severe undulations, large variations in overburden thickness

Specific cutting force
The digging resistance of the material being excavated is limited by:
• The bucket wheel drive power
• The mechanical strength of the bucket
• Machine mass (service weight)
Despite empirical research, no direct relationship mathematical model for predicting specific cutting force between the key variables has been established. These key variables include:

- Digging resistance
- Intact rock strength
- Jointing
- Bedding
- Tooth shape and sharpness
- Angle of attack
- Wedge angle of the bucket cutting lip

Hard Ground operation is possible with large, modern BWEs that have high wheel rotation speeds with slower slewing speeds. Material that cannot be dug out with a cable shovel such as hard clays, phosphates, sandstones, and frozen Tar Sands, can be excavated economically with BWEs. Provided that rock does not contain large hard boulders, and that the ground’s strength ranges between 15-18 MPa.

**Case Study: Big Brown Mine, Texas**

**General**

- The Big Brown Mine, located near Fairfield in Freestone County, is composed of two active mining areas that supply fuel to the Big Brown Steam Electric Station.
- Big Brown Mine fuel is used to generate approximately 7% of TU Electric's total electricity generation.
- TXU Mining receives about 17% of its total lignite production from the Big Brown Mine.
- About 12,000 acres have been mined at since mining began in 1971.
- The Mine areas have a combined total pit length of approximately 7 miles.
- Approximately 72 million cubic yards of overburden are moved per year. The total amount of overburden have been excavated since mining operations began is approximately 1.4 billion cubic yards.

**BWE Specifications:**

- **Height**: 41 feet
- **Width**: 33 feet
- **Length**: 144 feet
- **Weight**: 374 tons
- **Bearing Pressure**: 14.5 P.S.I.*
- **Wheel Diameter**: 26 feet
- **Wheel Speed**: 5.125 R.P.M.***
- **Number of Buckets**: 12
- **Bucket Size**: 1.08 yd³
- **Belts**: 2 (1 receiving, and 1 discharge)
- **Belt Width**: 60 inches
- **Belt Speed**: 885 F.P.M.**
- **Boom Length**: 49 feet (receiving) 82 (discharge)
- **Crawler Size**: 8 feet (width) 29 feet (length)
**Design Digging Height**......49 feet  
**Design Digging Width**.......60 feet BWE will dig 3 feet below level of tracks  
**Ground Speed**............... 32.8 F.P.M.**  
**Personnel Required**........ 1 operator and 1 oiler

**ADDITIONAL RESOURCES XIX**  
The above information is additionally supplemented by:  
- Assignment 2  
- Lecture for module 2.3  
- Reading – Chapter 13.5 – Continuous Excavators, SME Handbook, (Optional)

### 5.4 Dozers and Ripping\(^{11}\)

Dozers can be utilized for ground preparation when using a ripper, which is mounted to the rear of the tractor. Rippers are used widely in civil engineering projects to excavate slopes and wide swaths of surface area. Rippers are used in quarries to excavate sand and gravel deposits. Rippers are also used to some extent in mining for surface preparation and in coal production.

In the ripping process, the ripper shank is pulled through the soil or rock to loosen or fragment the material, which can then be loaded by a scraper or front-end loader, or handled by the dozer blade. Ripping is an inexpensive method of breaking discontinuous ground or soft rock masses. Figure 5-51 shows a ripper in action (back of dozer). In this operation, the dozer excavates the exposed coal and pushes the loosened material into windrows. Front-end loaders then load the material into haul trucks.

Rippability is determined by:  
- Compressive strength  
- Bedding planes, joints and fractures;  
- Britteness;  
- Softness (from weathering).

There are three ripper types, as seen in Figure 2 through Figure 4  
- Radial;  
- Parallelogram;  
- Adjustable parallelogram.

\(^{11}\) This lecture and its notes were complied using sources from:  
- Caterpillar Performance Handbook, 28th Ed.  
- Caterpillar Ripping Handbook, 12th Ed.  
- 2001 Class notes, from Bob Cummings  
- 1997 Class notes, from Sean Dessureault’s Surface Mining Course Notes for UBC’s Mining and Mineral Processing Department.  
Students are welcome to view these sources in my office.
In a Hinge-type ripper, the linkage carrying the beam and shank pivots about a fixed point at the rear of the tractor. As the shank enters the ground and penetrates to maximum depth, the tooth angle is constantly changing. Hinge-type rippers offer the advantage of an aggressive entry angle, but cannot be adjusted to compensate for varying conditions. A Parallelogram-type ripper allows the linkage carrying the beam and shank to maintain an essentially constant tip-ground angle regardless of tooth depth. This type of ripper has advantages over the hinge-type when ripping above maximum depth, but does not provide the aggressive tooth angle necessary for hard-to-penetrate materials. The Adjustable Parallelogram ripper combines the features of both the hinge-type and parallelogram rippers. It can vary the tip angle beyond vertical for improved penetration and can be hydraulically adjusted while ripping to provide the optimum ripping angle in most
materials. The ripper may have a multishank arrangement where multiple shanks are pulled by a single dozer (typically up to three).

**Figure 5-53: Ripper Nomenclature**

Other ripper nomenclature includes:
- **Pryout**, also known as breakout – the maximum sustained force upwards generated by the lift cylinders
- **Penetration force** – the maximum sustained downward force, generated by the ripper lift cylinders measured at the ripper tip.
Figure 5-54: Nomenclature for rippers (see Table 5-2)

Table 5-2: Ripper Selection for Large CAT Dozers.

<table>
<thead>
<tr>
<th>TRACTOR/RIPPER</th>
<th>D10R</th>
<th>D11R</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ripper Type</td>
<td>Adjustable Parallelogram</td>
<td>Adjustable Parallelogram</td>
</tr>
<tr>
<td></td>
<td>Single Shank</td>
<td>Multishank</td>
</tr>
<tr>
<td>Dimensions:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ripper to Track</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ripper length behind track, shank vertical, ripper up (A)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>A With Pushblock</td>
<td>2.08 m</td>
<td>6°10'</td>
</tr>
<tr>
<td>B Without Pushblock</td>
<td>1.76 m</td>
<td>5°9'</td>
</tr>
<tr>
<td>C With Pushblock</td>
<td>2.48 m</td>
<td>8°2'</td>
</tr>
<tr>
<td>D Without Pushblock</td>
<td>2.16 m</td>
<td>7°1'</td>
</tr>
<tr>
<td>Tip to track distance, shank vertical (A)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>E Ripper Up</td>
<td>730 mm</td>
<td>28.7°</td>
</tr>
<tr>
<td>F Ripper Down</td>
<td>1130 mm</td>
<td>44.5°</td>
</tr>
<tr>
<td>Ripper Shank*</td>
<td></td>
<td></td>
</tr>
<tr>
<td>G Maximum digging depth</td>
<td>1370 mm</td>
<td>53.9°</td>
</tr>
<tr>
<td>H Dig adjustment per hole</td>
<td>355 mm</td>
<td>14°</td>
</tr>
<tr>
<td>I Total dig adjustment</td>
<td>710 mm</td>
<td>28°</td>
</tr>
<tr>
<td>Pitch Adjustment, ripper down:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>J Forward</td>
<td>18°0'</td>
<td>18°0'</td>
</tr>
<tr>
<td>K Backward</td>
<td>19.7°</td>
<td>19.7°</td>
</tr>
<tr>
<td>L Maximum reach at ground line</td>
<td>1.50 m</td>
<td>4°11'</td>
</tr>
<tr>
<td>M Maximum ground clearance under tooth (shank pinned in bottom hole)</td>
<td>1070 mm</td>
<td>42.1°</td>
</tr>
<tr>
<td>N Maximum ramp angle, ripper up (shank pinned in bottom hole)</td>
<td>36.9°</td>
<td>37.5°</td>
</tr>
<tr>
<td>Shank Section</td>
<td>100 × 400 mm</td>
<td>90 × 355 mm</td>
</tr>
<tr>
<td>Ripper Beam</td>
<td></td>
<td></td>
</tr>
<tr>
<td>O Overall width</td>
<td>NA</td>
<td>2.92 m</td>
</tr>
<tr>
<td>P Height</td>
<td>NA</td>
<td>460 mm</td>
</tr>
<tr>
<td>Q Length</td>
<td>NA</td>
<td>485 mm</td>
</tr>
<tr>
<td>Clearance under beam, shank vertical</td>
<td></td>
<td></td>
</tr>
<tr>
<td>R Ripper Up</td>
<td>NA</td>
<td>2.03 m</td>
</tr>
<tr>
<td>S Ripper Down</td>
<td>NA</td>
<td>380 mm</td>
</tr>
<tr>
<td>T Number of Pockets</td>
<td>1</td>
<td>3</td>
</tr>
<tr>
<td>U Shank Gauge</td>
<td>NA</td>
<td>1230 mm</td>
</tr>
<tr>
<td>V Track Clearance with standard shoe</td>
<td>97 mm</td>
<td>4°</td>
</tr>
<tr>
<td>W Width across widest part of lift cylinders</td>
<td>1.75 m</td>
<td>5°9'</td>
</tr>
<tr>
<td>Installed Weights:</td>
<td>Ripper with standard shank</td>
<td>7117 kg</td>
</tr>
<tr>
<td>Each additional tooth group</td>
<td>NA</td>
<td>524 kg</td>
</tr>
<tr>
<td>Ripper Forces**:</td>
<td>Penetration Force, shank vertical</td>
<td>205 000 N</td>
</tr>
<tr>
<td></td>
<td>Pryout Force, shank vertical</td>
<td>429 000 N</td>
</tr>
</tbody>
</table>

*Deep Ripper Shank is available for D10R & D11R single shank rippers. Hydraulic pin puller is standard with deep ripping shank. Deep Ripper Arrangement maximum digging depth is 1.86 m (6.1') for D10R and 2.18 m (7') for D11R.

**Forces are for a ripper on a tractor equipped with an EROPS, U-Dozer and performance track. Forces will vary slightly with other vehicle configurations.

NA — Not Applicable.

Good penetration is essential for high production and depends on:
- Material;
- Down pressure;
• Point or tip angle with ground;
• Ripping direction (material bedding planes, slopes)

Cost comparison: Blasting vs. ripping:
• Higher repair costs;
• Ripper tip replacement should be included in hourly costs;
• Machine life average 8000 hrs (continuous ripping)
• May be 1/3 to ½ of blasting, in proper application.

5.4.1 Determining Rippability.
Ripping is an inexpensive method of removing discontinuous ground or soft rock masses. Some weaker fragmented sedimentary rocks (less than 15 MPa compressive strength, example: mudstone) are not easily removed by blasting as they are pulverized in their immediate vicinity of the blasthole or may lift along bedding planes then fall back when gas pressure has been dissipated. Figure 5-55 shows a rough chart of various excavation methods considering several ground types. Table 5-3 provides excavation characteristics for a wide variety of material.

![Figure 5-55: Rock quality classification in relation to excavation processes.](image-url)
Table 5-3: Excavation Characteristics in relation to rock hardness and strength

<table>
<thead>
<tr>
<th>Rock hardness description</th>
<th>Identification criteria</th>
<th>Unconfined compression strength (MPa)</th>
<th>Seismic wave velocity (m/s)</th>
<th>Excavation characteristics</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very soft rock</td>
<td>Material crumbles under firm blows with sharp end of geological pick; can be poried with a lump; too hard to cut a triangular sample by hand</td>
<td>1.7–3.0</td>
<td>450–1200</td>
<td>Easy ripping</td>
</tr>
<tr>
<td>Soft rock</td>
<td>Can just be scraped with a knife; indentations 1 mm to 3 mm show in the specimen with firm blows of the pick point; has dull sound under hammer.</td>
<td>3.6–11.9</td>
<td>1200–1500</td>
<td>Hard ripping</td>
</tr>
<tr>
<td>Hard rock</td>
<td>Cannot be scraped with a knife; hand specimen can be broken with pick with a single firm blow; rock rings under hammer.</td>
<td>10.0–20.0</td>
<td>1500–1850</td>
<td>Very hard ripping</td>
</tr>
<tr>
<td>Very hard rock</td>
<td>Hard specimen breaks with pick after more than one blow; rock rings under hammer.</td>
<td>20.0–70.0</td>
<td>1850–2150</td>
<td>Extremely hard ripping or blasting</td>
</tr>
<tr>
<td>Extremely hard rock</td>
<td>Specimen requires many blows with geological pick to break through intact material; rock rings under hammer.</td>
<td>&gt;70.0</td>
<td>&gt;2150</td>
<td>Blasting</td>
</tr>
</tbody>
</table>

Table 5-4: Excavation Characteristics in relation to joint spacing

<table>
<thead>
<tr>
<th>Joint spacing description</th>
<th>Spacing of joints (mm)</th>
<th>Rock mass grading</th>
<th>Excavation characteristics</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very close</td>
<td>&lt;50</td>
<td>Crushed/shattered</td>
<td>Easy ripping</td>
</tr>
<tr>
<td>Close</td>
<td>50–300</td>
<td>Fractured</td>
<td>Hard ripping</td>
</tr>
<tr>
<td>Moderately close</td>
<td>300–1000</td>
<td>Blocks/seamy</td>
<td>Very hard ripping</td>
</tr>
<tr>
<td>Wide</td>
<td>1000–3000</td>
<td>Massive</td>
<td>Extremely hard ripping and blasting</td>
</tr>
<tr>
<td>Very wide</td>
<td>&gt;3000</td>
<td>Solid/sound</td>
<td>Blasting</td>
</tr>
</tbody>
</table>

Figure 5-56: Seismic velocities in relation to ripping

The most common method for determining if a rock mass is economically rippable is seismic refraction. Figure 5-56 shows the seismic velocity of
various rock types and their rippability. The upper limit of ripper operations is ground with a seismic velocity approximately 2 km/s. However, this generalized measurement only provides a very limited view of the issues in ripping operation. Seismic velocity can vary as much as 1 km/s in identical materials. Rippability assessments based on rock classification is far more accurate. An adapted Q system of rock classification can be used to define an excavability index, N, which can be calculated by the equation below, and supplemented by Table 5-5. Table 5-6 through Table 5-10 provide comparative tables from which an excavability index can be calculated.

\[ N = M_s \frac{(RQD)}{(J_n)} \times J_s \times \frac{J_r}{J_a} \]

**Table 5-5: Variables in excavability index**

<table>
<thead>
<tr>
<th>Variable</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>(M_s)</td>
<td>Mass strength number (amount of effort needed to excavate dry, homogeneous material with no discontinuities)</td>
</tr>
<tr>
<td>RQD</td>
<td>Rock quality designation</td>
</tr>
<tr>
<td>(J_n)</td>
<td>Number of joint sets</td>
</tr>
<tr>
<td>(J_s)</td>
<td>Reducing effect which the block shape and orientation has on the force needed to break out the material</td>
</tr>
<tr>
<td>(J_r)</td>
<td>Roughness of the most unfavorable joint sets</td>
</tr>
<tr>
<td>(J_a)</td>
<td>Degree of alteration</td>
</tr>
</tbody>
</table>

**Table 5-6: Mass Strength Number for Rocks (Ms)**

<table>
<thead>
<tr>
<th>Hardness</th>
<th>Identification in profile</th>
<th>Unconfined compressive strength (MPa)</th>
<th>Mass strength number (Ms)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very soft rock</td>
<td>Material crumbles under firm (moderate) blows with sharp end of geological pick and can be peeled off with a knife. It is too hard to cut a triaxial sample by hand</td>
<td>1.7</td>
<td>0.87</td>
</tr>
<tr>
<td>Soft rock</td>
<td>Can just be scraped and peeled with a knife; indentations 1–3 mm show in the specimen with firm (moderate) blows of the pick point</td>
<td>3.3–6.6</td>
<td>3.95</td>
</tr>
<tr>
<td>Hard rock</td>
<td>Cannot be scraped or peeled with a knife; hand-held specimen can be broken with hammer end of geological pick with a single firm (moderate) blow</td>
<td>13.2–26.4</td>
<td>17.70</td>
</tr>
<tr>
<td>Very hard rock</td>
<td>Hand-held specimen breaks with hammer end of pick under more than one blow</td>
<td>26.4–53.0</td>
<td>35.0</td>
</tr>
<tr>
<td>Extremely hard rock (very, very hard rock)</td>
<td>Specimen requires many blows with geological pick to break through intact material</td>
<td>53.0–106.0</td>
<td>70.0</td>
</tr>
<tr>
<td></td>
<td></td>
<td>106.0–212.0</td>
<td>140.0</td>
</tr>
<tr>
<td></td>
<td></td>
<td>212.0</td>
<td>280.0</td>
</tr>
</tbody>
</table>
Table 5-7: Joint count number (J_c) and Joint set number (J_n)

<table>
<thead>
<tr>
<th>Number of joints per cubic metre (J_c)</th>
<th>Rock quality designation (RQD)</th>
<th>Number of joints per cubic metre (J_c)</th>
<th>Rock quality designation (RQD)</th>
</tr>
</thead>
<tbody>
<tr>
<td>33</td>
<td>5</td>
<td>18</td>
<td>55</td>
</tr>
<tr>
<td>32</td>
<td>10</td>
<td>17</td>
<td>60</td>
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<td>30</td>
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<td>29</td>
<td>20</td>
<td>14</td>
<td>70</td>
</tr>
<tr>
<td>27</td>
<td>25</td>
<td>12</td>
<td>75</td>
</tr>
<tr>
<td>26</td>
<td>30</td>
<td>11</td>
<td>80</td>
</tr>
<tr>
<td>24</td>
<td>35</td>
<td>9</td>
<td>85</td>
</tr>
<tr>
<td>23</td>
<td>40</td>
<td>8</td>
<td>90</td>
</tr>
<tr>
<td>21</td>
<td>45</td>
<td>6</td>
<td>95</td>
</tr>
<tr>
<td>20</td>
<td>50</td>
<td>5</td>
<td>100</td>
</tr>
</tbody>
</table>

Note: For intact granular materials take \( J_n = 5.00 \)

Table 5-8: Relative ground structure number (J_s)

<table>
<thead>
<tr>
<th>Dip direction* of closer spaced joint set (degrees)</th>
<th>Dip angle† of closer spaced joint set (degrees)</th>
<th>Ratio of joint spacing, ( r )</th>
<th>( 1:1 )</th>
<th>( 1:2 )</th>
<th>( 1:4 )</th>
<th>( 1:8 )</th>
</tr>
</thead>
<tbody>
<tr>
<td>180/0</td>
<td>90</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
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<tr>
<td>0</td>
<td>85</td>
<td>0.72</td>
<td>0.67</td>
<td>0.62</td>
<td>0.50</td>
<td>0.43</td>
</tr>
<tr>
<td>0</td>
<td>80</td>
<td>0.63</td>
<td>0.57</td>
<td>0.50</td>
<td>0.41</td>
<td>0.34</td>
</tr>
<tr>
<td>0</td>
<td>70</td>
<td>0.52</td>
<td>0.45</td>
<td>0.41</td>
<td>0.34</td>
<td>0.28</td>
</tr>
<tr>
<td>0</td>
<td>60</td>
<td>0.49</td>
<td>0.44</td>
<td>0.41</td>
<td>0.34</td>
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</tr>
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<td>0.49</td>
<td>0.46</td>
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</tr>
<tr>
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<td>0.59</td>
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</tr>
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<td>0</td>
<td>20</td>
<td>0.84</td>
<td>0.77</td>
<td>0.71</td>
<td>0.64</td>
<td>0.58</td>
</tr>
<tr>
<td>0</td>
<td>10</td>
<td>1.22</td>
<td>1.10</td>
<td>0.99</td>
<td>0.92</td>
<td>0.85</td>
</tr>
<tr>
<td>0</td>
<td>5</td>
<td>1.33</td>
<td>1.20</td>
<td>1.09</td>
<td>1.02</td>
<td>0.95</td>
</tr>
<tr>
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<td>0</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
<tr>
<td>180</td>
<td>5</td>
<td>0.72</td>
<td>0.81</td>
<td>0.86</td>
<td>0.82</td>
<td>0.80</td>
</tr>
<tr>
<td>180</td>
<td>10</td>
<td>0.63</td>
<td>0.70</td>
<td>0.76</td>
<td>0.73</td>
<td>0.70</td>
</tr>
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<td>20</td>
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<td>0.57</td>
<td>0.63</td>
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<td>1.40</td>
<td>1.48</td>
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<td>1.33</td>
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<td>1.45</td>
<td>1.53</td>
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<td>90</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
</tbody>
</table>

* Dip-direction of closer spaced joint set relative to direction of rip
† Apparent dip angle of closer spaced joint set in vertical plane containing direction of ripping

\( \text{for intact material take } J_s = 1.0 \)

\( \text{for values of } r \text{ less than 0.125 take } J_s \text{ as for } r = 0.125 \)
### Table 5-9: Joint roughness number ($J_r$)

<table>
<thead>
<tr>
<th>Joint separation</th>
<th>Condition of joint</th>
<th>Joint roughness number ($J_r$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Joints tight or closing during excavation</td>
<td>Discontinuous joint</td>
<td>4.0</td>
</tr>
<tr>
<td></td>
<td>Rough or irregular, undulating</td>
<td>3.0</td>
</tr>
<tr>
<td></td>
<td>Smooth undulating</td>
<td>2.0</td>
</tr>
<tr>
<td></td>
<td>Slickensided undulating</td>
<td>1.5</td>
</tr>
<tr>
<td></td>
<td>Rough or irregular, planar</td>
<td>1.5</td>
</tr>
<tr>
<td></td>
<td>Smooth planar</td>
<td>1.0</td>
</tr>
<tr>
<td></td>
<td>Slickensided planar</td>
<td>0.5</td>
</tr>
<tr>
<td>Joints open and remain open during excavation</td>
<td>Joints either open or containing relatively soft gouge of sufficient thickness to prevent joint wall contact upon excavation</td>
<td>1.0</td>
</tr>
</tbody>
</table>

### Table 5-10: Joint Alteration number ($J_a$)

<table>
<thead>
<tr>
<th>Description of gouge</th>
<th>Joint alteration number ($J_a$) for joint separation (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tightly healed, hard, non-softening impermeable filling</td>
<td>&lt;1.0†</td>
</tr>
<tr>
<td>Unaltered joint walls, surface staining only</td>
<td>1.0</td>
</tr>
<tr>
<td>Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling</td>
<td>2.0</td>
</tr>
<tr>
<td>Non-softening, slightly clayey non-cohesive filling</td>
<td>3.0</td>
</tr>
<tr>
<td>Non-softening strongly over-consolidated clay mineral filling, with or without crushed rock</td>
<td>3.0§</td>
</tr>
<tr>
<td>Softening or low-friction clay mineral coatings and small quantities of swelling clays</td>
<td>4.0</td>
</tr>
<tr>
<td>Softening moderately over-consolidated clay mineral filling, with or without crushed rock</td>
<td>4.0§</td>
</tr>
<tr>
<td>Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock</td>
<td>5.0</td>
</tr>
</tbody>
</table>

* Joint walls effectively in contact
† Joint walls come into contact after approximately 100 mm shear
‡ Joint walls do not come into contact at all upon shear
§ Values added to Barton’s data
5.4.2 Operational issues:

5.4.2.1 Ripping technique:
Using hydraulic force to help fracture rock is an effective technique for ripping hard rock seams or pockets is to combine tractor drawbar power and hydraulic force. When the ripper has contacted the hard rock, the following procedure is undertaken:
1. The decelerator is used to control trackslip.
2. The ripper shank is controlled to adjust shank angle back slightly.
3. Engine speed is maintained high enough to allow the tractor to continue moving forward as the ripper shank angles back.
4. While the tractor moves forward, the ripper shank is returned to its original position (forward), combining ripper hydraulic force with tractor drawbar pull.

5.4.2.2 Ripping Downgrade
Ripping downgrade can increase production. If the job layout permits, the downgrade approach can be helpful when working a hard spot or seam. There are several key issues to be aware of when ripping downgrade.
- Traction on rock is less than on dirt.
- Avoid ripping on or creating slopes the tractor cannot climb.
- Avoid sideslopes.

5.4.3 Estimating Ripping Production
Ripper production rates: can be determined empirically by:
- Determining the average cycle time, measure average rip distance, rip spacing, and depth penetration, calculate production rate; or
- Record time spent ripping, remove and weigh (through surveying & volume calculations), then calculate production rate.

Since ripping is primarily compared to drilling and blasting, on a per ton or volume basis, an estimate of ripper production in these terms is needed. The simplest method is to record the time spent ripping, then remove the weight of the material. For example, consider an example where a CAT D10R dozer equipped with a single shank ripper excavates a path every 90 cm (width of tear) at an average speed of 1.6 km/h. After every 91 meters, the machine must turn (takes 0.25 minutes to raise shank, pivot dozer, and lower shank again). This is the equivalent of 1 pass. Consider that the ripper shank has a penetration depth of 61 cm.

Solution:
1.6 km/h = 26.7 m/min. Therefore \( \frac{91m}{26.7m/\text{min}} = 3.41 \text{ min} \)
3.41 + .25 (turn time) = 3.66 min/pass
if operator works 45 min/h, is would be possible to make 45/3.66=12.3 passes per hour.

The volume ripped can be estimated by: 91m x 0.61m x 0.90m = 49.1 BCM per pass

Therefore the volume per pass is equivalent to 12.3 x 49.1 = 604 BCM/h

Note that these estimates are approximately 10-20% higher than what would be expected on-site.

Productivity can also be estimated using the ripper production charts, such as those seen in Figure 5-57. However, it should be recognized that in these charts:

- Rip full time (does not doze simultaneously)
- Power shift tractors with single shank rippers
- 100% efficiency (60 min/hr)
- For all classes of material.

Figure 5-58 also provides estimates for ripper productivities for various material type.

![Figure 5-57: Ripper Production Chart](image-url)
Figure 5-58: Performance chart for D11

Figure 5-59: D11 with ripper
5.4.4 Ripping vs. Blasting

Improvements in ripping tractor capabilities have made ripping a viable alternative to blasting. As mentioned before, environmental factors will undoubtedly play an important role in reaching this decision. For example, in an urban area there may be restrictions prohibiting the use of explosives, making ripping a necessity. Political factors or the threat of terrorism severely restrict the use of explosives in some countries. But in most situations, where there is equal opportunity for the use of either method, the first consideration is probably one of cost - will it be cheaper to rip or blast? This initial-cost consideration must then be weighed with other influencing factors: the economics of fully utilizing equipment; the end use of the material; and transporting and loading methods.

Full utilization of the equipment available or already on the job can help determine the best method of loosening the material. This is because many earthmoving jobs already involve track-type tractors and scrapers for a sizable portion of the total yardage. If this equipment can be used to finish the job - rather than bring in a rock crew with drills, explosives, loaders and hauling units - it's not difficult to appreciate the savings involved. It's soon apparent that considerable effort can be expended to rip the material in order to keep scrapers on the job.

End-use of the material also influences the ripping vs. blasting decision. There are few size limitations when the rock is simply moved by a bulldozer and "wasted." If the material is used to form an embankment, however, very definite limitations are usually placed upon the size of the rocks to be accepted. Optimum compaction cannot be obtained if there are large rocks in the fill. Variations in ripping depth, spacing, and direction of passes usually can produce the desired material size. Blasting is at times unpredictable, as the desired rock fragmentation may be difficult to obtain and even require expensive secondary blasting (in effect, reblasting). Appreciable increases in crushe production have been realized by cement plants and aggregate quarries after switching from a blasting to a ripping operation.

The final comparison of ripping vs. blasting can be made in terms of how the material is to be moved. As we stated, dozed material presents few problems. Material top loaded into hauling units cannot be larger than the loading bucket. Scrapers can inexpensively haul materials which are well broken up and loosened. Elevating loaders and conveyors are high capacity systems. Their greatest advantage - high production - can be achieved only if the material is in small pieces and easy to handle. Generally, ripping is the most cost-effective method to achieve these requirements.

A cost analysis will indicate the economics of ripping over drilling blasting. This comparison indicates how ripper tip life is an important factor in deciphering the production needed for ripping to be cost effective. Ripper tips are the most expensive variable in the operating costs of ripping tractors, accounting for approximately 30 to 40 percent of total operating costs on the
largest tractors. In the final analysis, a ripping vs. blasting decision will depend on the total volume of material to be loosened and moved, on the production capabilities and costs of the ripping tractor(s) used, and on the size and relative efficiency of drilling and blasting techniques.

**ADDITIONAL RESOURCES XX**
The above information is additionally supplemented by:
- Lecture for module 2.4 (Mandatory)
- Assignment XX?