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CHAPTER 1 – Purpose and Scope of Mining

MINING, A BASIC INDUSTRY

Mining and agriculture are the two basic (primary) industries that have contributed to the development of modern civilization. Mining supplies such items as structural material, e.g. stone, glass sand, clays and cement rocks, fuels, natural gas, coal and petroleum; abrasives, such as garnet and corundum; fertilizers, potash, phosphate and nitrates; industrial minerals such as sulfur, graphite and asbestos; ores of metallic minerals, such precious metals as gold and silver; base metals of copper, lead, zinc, iron and aluminium; precious stones such as diamonds; and fissionable materials. The extraction of these substances from the ground through mining is of vital importance to man’s economic existence and development.

MINERALS, ROCKS AND ORE

There are 104 elements known to man of which fifteen exist only under laboratory conditions. Fundamentally, elements have originated from the magmas or igneous rocks of the earth’s outer shell or crust. Eight elements constitute 98% by weight of the crust of the earth: these are oxygen 7%; silicon 28%; aluminium 8%; iron 5%; sodium, magnesium, potassium and calcium, less than 4% each. It is these common elements and some less common ones that constitute some 2000 varieties of minerals.

A mineral is an inorganic substance occurring naturally in the earth and having a consistent and distinctive set of physical properties and a composition that can be expressed by a chemical formula. At times this definition is applied to such organic substances as coal. Hence minerals are precise combinations of elements. Rocks, as distinct from minerals, are composed of assemblages of minerals.

When minerals are found in sufficient concentration to warrant extraction by mining, the mineralized area is considered an ore deposit. The definition of ore is a mineral that can be extracted from the ground at a profit. The term ore has an implicit economic connotation. Ore deposits would not occur if it were not for geological processes which have concentrated the elements, because most of the useful elements comprise a very small percentage of the earth’s crust.

LEGAL DEFINITION OF MINERALS

While the above definition is used for Common Law, under Mining Act Legislation the term mineral is a general definition for all minerals other than gold and all precious stones. Consequently, material such as limestone, which is a rock because of its abundance, when quarried is termed a mineral under the mining legislation.

MINING AND MINES CLASSIFICATION

Mining, defined broadly, is the act, process or work of extracting minerals or coal from their natural environment and transporting them to the point of processing or use.

A mine is therefore an excavation in the earth for the purpose of extracting minerals. Such excavations may be made at the surface or underground, or both surface and underground methods may be employed.
Mines are divided into three broad categories: metallic, non-metallic and coal. This type of grouping is convenient because of the generalized similarities within each category with respect to geology, mining techniques, processing and marketing.

Most base metal deposits are composed of several valuable minerals mixed with waste minerals termed Gangue. In mining, the objective is to extract as much of the valuable mineral as possible and leave behind the waste. The mine product material referred to as ore is a combination of valuable minerals and gangue.

In the case of metallic deposits the ore is usually processed at the mine to concentrate the valuable mineral for shipment to the smelter or other refinery facility.

The gangue is discarded.

Non-metallic deposits and fuels, unlike metalliferous deposits, contain more of the valuable mineral or element than of waste. Non-metallic minerals (also called industrial minerals) are prepared at the mine to a finished product, or the prepared mineral may be shipped elsewhere for additional treatment before being marketed.

Coal is usually washed at the mine to remove to the maximum extent possible, slate and other impurities before being shipped to power plants or coke ovens. Power plants at or near the mine site are referred to as mine mouth power stations.

Surface mines may be categorised into several types. An open-pit mine, which is an excavation open to the surface, is dug to extract metallic ores. In a typical open-pit a series of multi-benches or terraces are used to excavate the material. In strip mining, the overlying surface material, called overburden, is removed to expose the coal seam for excavation. A quarry is an excavation for extracting stone, sand or gravel. In a placer mine the valuable material is found on the bedrock beneath gravel in stream beds, fossil stream beds or flood plains and is recovered by panning, hydraulicking or dredging.

In underground mines a wide range of diverse extraction methods are employed. These include such approaches as block caving, top slicing, sub-level caving, longhole stoping, shrinkage stoping, open stoping, cut and fill and many others. The reason for the many methods of underground mining is that mineral deposits have different shapes and the mineral and host rock of each ore body have differing physical properties/characteristics, which affect the choice of mining method.

Coal deposits are mined underground by several methods but these are fewer than those used for mineral deposits because most economic coal deposits are similar tabular, relatively flat lying beds of great extent. The main methods of extracting coal underground is by bord-and-pillar and longwall mining.

Another important underground mining method is solution mining. In a solution mine, a liquid is pumped into the ground through a borehole to liquefy or dissolve a deposit which is then pumped to the surface. Examples are sulphur and salt mining. Chemical solutions may be used to extract copper and uranium from their natural environment.
The scope of mining activities from exploration for a mineral deposit through the finding, proving and developing, mining and processing to marketing the products, is best illustrated by a schematic diagram. (Fig. 1.1).

**MINING – CULTURAL SIGNIFICANCE**

The products of Mining have been used by anthropologists to designate steps in man’s culture, such as Palcolithic (old Stone Age), Neolithic (New Stone Age), Bronze Age (4000 - 1800 B.C. in the Orient; 2000 - 1000 B.C. in Europe) and Iron Age (following the Bronze Age). Standards of living of different peoples of the world are still compared on the basis of per capita consumption of the various metals.
Stone age man used flint for tools and weapons, not until 1835 A.D. that flint and steel for making fire were replaced by matches.

From earliest man to the first civilizations the products of the earth won by mining were essential for implements, building, trade, jewellery, cosmetics and treasure.

Between the great civilizations of antiquity and the Industrial Revolution there were not many changes in products mined. Not until relatively modern times have new elements and minerals been identified and put to use to serve society, in part creating the complex hardware which surrounds every-day living.

In particular, aspects of transportation, communication and construction have been revolutionized because of the use of newly developed materials. Agriculture too has been drastically changed through the use of mineral fertilizers such as phosphate, potash and nitrate.

Mining has had a political impact on the history of society because the acquisition of mines often meant wealth; today oil reserves are particularly important economically.

**MINING TECHNOLOGY – EVOLUTION**

Early miners used their hands and implements of wood, bone, stone and, later, of metal to dig and extract minerals. Gradually improved implements such as the wedge and mallet, baskets, ladders, windlasses and candles and oil lamps were introduced to enable early miners to dig mines up to 250 metres deep.

Perhaps one of the most notable treatises on early mining comes from the 16th Century. This is the De Re Metallica by Georgios Agricola published in 1556. An English translation was published in 1912 with modern editions available.

The Industrial Revolution created a demand for metals, which intensified the search for minerals and accelerated new mine development in the 17th and 18th Centuries. This need for minerals brought further important changes in mining practice especially the introduction in 1627, in Hungary, of black powder for blasting rock and in 1718, the use of pumping engines to dewater Cornish tin mines.

The steam engine and the air compressor supplied energy to mining far in excess of that afforded by human muscle. By the late 19th Century the air-driven rock drill had replaced hand drilling. The invention of electricity provided impetus for mechanization and made application of machinery more flexible.

These original developments appear as land-marks in the history of mining, but innovation and change have been continuous. Actual developments will be examined in the context of mining today in successive chapters. The trend is towards greater mechanization and higher capacity units with resulting increased efficiency in labour and improved productivity. This has been essential to ensure growth to meet increased demands for minerals.

**MINERAL ECONOMICS**

Mineral deposits are not evenly distributed throughout the world and few nations are self-sufficient in all mineral commodities.

Individual mineral deposits become valuable for several reasons: they may be very rich, or more accessible than other deposits; the location may be more favourable, or the commodity in greater demand. Hence the value of a deposit may not be intrinsic but related to the relevance placed upon it by man.
In some instances the mining company has created demand for a commodity by developing uses for it as was the case with nickel. Technological advances have made low grade deposits economically viable. This has occurred through economies of scale and improved methods of creating the material. A mineral classified as a curiosity may be reclassified as economic because of market development.

Because mineral deposits are a wasting asset, the economies of mining are closely allied with efficient extraction. The minerals are irreplaceable; as the deposit is mined it decreases in value. Therefore a mining operation needs to conserve its ore by planning mining so as to achieve maximum profit during the life of the mine and additional minerals must be searched for if the miner is to stay in business.

**PRODUCTION STATISTICS – MINING ACTIVITY**

World metal and mineral production statistics are available from a variety of sources, but they seldom provide a reliable guide to the extent of mining due to wide variations in the grade of ore mined.

Reliable estimates of world production are available only for non-Communist countries. (See Table 1.1).

<table>
<thead>
<tr>
<th>CONTINENTAL AREA</th>
<th>OPEN PIT ORE PROD'N %</th>
<th>U.G. ORE PROD'N %</th>
<th>HARD COAL PROD'N %</th>
<th>LIGNITE PROD'N %</th>
</tr>
</thead>
<tbody>
<tr>
<td>North America</td>
<td>849</td>
<td>45.3</td>
<td>227</td>
<td>30.8</td>
</tr>
<tr>
<td>Central &amp; South America</td>
<td>343</td>
<td>18.3</td>
<td>47</td>
<td>6.4</td>
</tr>
<tr>
<td>Australasia</td>
<td>153</td>
<td>8.1</td>
<td>21</td>
<td>2.8</td>
</tr>
<tr>
<td>Europe</td>
<td>71</td>
<td>3.8</td>
<td>179</td>
<td>24.3</td>
</tr>
<tr>
<td>Africa</td>
<td>272</td>
<td>14.5</td>
<td>220</td>
<td>29.9</td>
</tr>
<tr>
<td>Asia</td>
<td>188</td>
<td>10.0</td>
<td>43</td>
<td>5.8</td>
</tr>
<tr>
<td>TOTALS</td>
<td>1876</td>
<td>100.0</td>
<td>737</td>
<td>100.0</td>
</tr>
</tbody>
</table>

**U.G. (UNDERGROUND)**

*Table 1.1 Estimated non-communist world ore capacities 1977 and continental coal output (millions of tonnes)*

The term 'ore' with reference to world statistics consists of 22 metals and minerals. These are: iron, copper, lead, zinc, tin, bauxite, gold, diamonds, platinum, manganese, nickel, uranium, chrome, mercury, molybdenum, asbestos, boron, potash, phosphate rock, tungsten, titanium and silver.
It is estimated that there are 7 000 mines in the Western World excluding coal mines. Total ore production capacity by both underground and open pit is about 2 700 million tones (1977). 90% of this production comes from approximately 1 000 mines with an annual production in excess of 150 000 tonnes per year. Recent trends have been towards an increase in size of mines where larger mines are more suited to mechanization and will generate cash flows sufficient to purchase mechanized equipment. Very large project development has been curtailed because of rapid cost inflation and low metal and mineral prices.

An analysis of mines producing more than 150 000 tonnes of ore per year is shown in Table 1.2. However, for a number of minerals and in some countries small mines do represent an important part of the industry.

Coal mining has seen considerable growth in recent years. Some 75% of the non Communist production is produced by mines with annual outputs in excess of 250 000 tonnes. In 1976 only 10 countries produced more than 10 million tonnes per year of hard coal.
Table 1.3 Western world hard coal production 1976

Mine production statistics for Australia are available from such publications as the Australian Mineral Industry Quarterly and Australian Mineral Industry Review prepared by the Bureau of Mineral Resources, Geology and Geophysics.

Table 1.4 shows Australian Mineral production for 1975 and 1976.

### Table 1.4 Australian Mineral Summary 1975 – 1976

<table>
<thead>
<tr>
<th>Country</th>
<th>MILLIONS OF TONNES</th>
</tr>
</thead>
<tbody>
<tr>
<td>U.S.A.</td>
<td>594</td>
</tr>
<tr>
<td>United Kingdom</td>
<td>124</td>
</tr>
<tr>
<td>India</td>
<td>102</td>
</tr>
<tr>
<td>West Germany</td>
<td>96</td>
</tr>
<tr>
<td>South Africa</td>
<td>76</td>
</tr>
<tr>
<td>Australia</td>
<td>74</td>
</tr>
<tr>
<td>Canada</td>
<td>25</td>
</tr>
<tr>
<td>France</td>
<td>22</td>
</tr>
<tr>
<td>Japan</td>
<td>18</td>
</tr>
<tr>
<td>Korea</td>
<td>16</td>
</tr>
</tbody>
</table>

**Table 1.3 Western world hard coal production 1976**

**Table 1.4 Australian Mineral Summary 1975 – 1976**

**MINING**

Mine production statistics for Australia are available from such publications as the Australian Mineral Industry Quarterly and Australian Mineral Industry Review prepared by the Bureau of Mineral Resources, Geology and Geophysics.

Table 1.4 shows Australian Mineral production for 1975 and 1976.

### Table 1.4 Australian Mineral Summary 1975 – 1976

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Unit of Quantity</th>
<th>1975</th>
<th>1975*</th>
<th>December</th>
<th>March</th>
<th>June</th>
<th>September</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper (a)</td>
<td>tonne</td>
<td>218.961</td>
<td>214.427</td>
<td>51.241</td>
<td>49.451</td>
<td>55.658</td>
<td>57.432</td>
</tr>
<tr>
<td>Gold (a)</td>
<td>kilogram</td>
<td>16.386</td>
<td>15.401</td>
<td>3.564</td>
<td>3.701</td>
<td>4.224</td>
<td>4.868</td>
</tr>
<tr>
<td>Iron ore and concentrate (b)</td>
<td>'000 tonnes</td>
<td>97.651</td>
<td>95.255</td>
<td>23.850</td>
<td>24.950</td>
<td>21.861</td>
<td>26.452</td>
</tr>
<tr>
<td>Lead (a)</td>
<td>tonne</td>
<td>407.901</td>
<td>398.323</td>
<td>98.552</td>
<td>91.248</td>
<td>112.758</td>
<td>120.442</td>
</tr>
<tr>
<td>Manganese ore, metallurgical</td>
<td></td>
<td>1.554.909</td>
<td>2.154.167</td>
<td>578.753</td>
<td>302.457</td>
<td>389.467</td>
<td>442.961</td>
</tr>
<tr>
<td>Nickel (a)</td>
<td></td>
<td>75.825</td>
<td>83.145</td>
<td>19.837</td>
<td>18.825</td>
<td>20.022</td>
<td>24.246</td>
</tr>
<tr>
<td>Petroleum—</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Natural gas</td>
<td>mill. cu. metres</td>
<td>5.026</td>
<td>5.292</td>
<td>1.477</td>
<td>1.388</td>
<td>1.773</td>
<td>1.985</td>
</tr>
<tr>
<td>Silver (a)</td>
<td>kilogram</td>
<td>726.218</td>
<td>779.870</td>
<td>206.694</td>
<td>182.269</td>
<td>214.427</td>
<td>233.300</td>
</tr>
<tr>
<td>Tin (a)</td>
<td>tonne</td>
<td>9.577</td>
<td>10.109</td>
<td>2.561</td>
<td>2.556</td>
<td>2.518</td>
<td>2.302</td>
</tr>
<tr>
<td>Titanium—</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ilmenite concentrate (c)</td>
<td></td>
<td>981.433</td>
<td>984.789</td>
<td>266.894</td>
<td>270.548</td>
<td>234.382</td>
<td>284.022</td>
</tr>
<tr>
<td>Rutile concentrate (d)</td>
<td></td>
<td>340.350</td>
<td>355.343</td>
<td>104.440</td>
<td>80.010</td>
<td>89.071</td>
<td>76.403</td>
</tr>
<tr>
<td>Tungsten concentrate (e)</td>
<td>unit</td>
<td>188.800</td>
<td>245.005</td>
<td>65.154</td>
<td>67.909</td>
<td>99.396</td>
<td>68.077</td>
</tr>
<tr>
<td>Zinc (a)</td>
<td>tonne</td>
<td>500.646</td>
<td>487.604</td>
<td>115.721</td>
<td>111.833</td>
<td>119.961</td>
<td>139.965</td>
</tr>
<tr>
<td>Zircon concentrate (f)</td>
<td></td>
<td>382.217</td>
<td>418.410</td>
<td>114.038</td>
<td>99.529</td>
<td>102.464</td>
<td>107.861</td>
</tr>
</tbody>
</table>

(a) Total metallic content of minerals produced. (b) Excludes iron oxide not intended for metallic extraction. (c) Excludes ilmenite. (d) Tungsten oxide concentrate.
CHAPTER 1 PURPOSE AND SCOPE OF MINING

1.8


<table>
<thead>
<tr>
<th>Mine product</th>
<th>Unit of quantity</th>
<th>1976</th>
<th>1977</th>
<th>1978</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Quantity</td>
<td>Value ($'000)</td>
<td>Quantity</td>
</tr>
<tr>
<td>Selenium in mine products</td>
<td>tonne</td>
<td>35</td>
<td>(b)</td>
<td>1301</td>
</tr>
<tr>
<td>Silicon glass, chemicals, etc.</td>
<td>'000 tonnes</td>
<td>1 381</td>
<td>8 052</td>
<td>1 302</td>
</tr>
<tr>
<td>Sillimanite</td>
<td>tonne</td>
<td>567</td>
<td>18</td>
<td>550</td>
</tr>
<tr>
<td>Silver concentrates</td>
<td>tonne</td>
<td>3 067</td>
<td>4 488</td>
<td></td>
</tr>
<tr>
<td>Silver content</td>
<td>kilogram</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Silver-gold ores and cons</td>
<td>tonne</td>
<td>45</td>
<td>60</td>
<td></td>
</tr>
<tr>
<td>Silver content</td>
<td>kilogram</td>
<td>55</td>
<td>(b)</td>
<td></td>
</tr>
<tr>
<td>Silver-lead concentrates</td>
<td>tonne</td>
<td></td>
<td></td>
<td>54</td>
</tr>
<tr>
<td>Lead content</td>
<td>tonne</td>
<td>3</td>
<td>(b)</td>
<td></td>
</tr>
<tr>
<td>Silver content</td>
<td>kilogram</td>
<td></td>
<td></td>
<td>69</td>
</tr>
<tr>
<td>Silver in mine products</td>
<td>kilogram</td>
<td>778 658</td>
<td>(b)</td>
<td>856 110</td>
</tr>
<tr>
<td>Slate industrial</td>
<td>tonne</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Tantalite-columbite cons</td>
<td>kilogram</td>
<td>118 000</td>
<td>1 348</td>
<td>157 000</td>
</tr>
<tr>
<td>Tin cons.</td>
<td>tonne</td>
<td>21 622</td>
<td>66 960</td>
<td>21 255</td>
</tr>
<tr>
<td>Tin content</td>
<td>tonne</td>
<td>10 531</td>
<td>(b)</td>
<td>10 577</td>
</tr>
<tr>
<td>Tungsten</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Scheelite cons</td>
<td>tonne</td>
<td>2 699</td>
<td>17 787</td>
<td>3 286</td>
</tr>
<tr>
<td>WE2 content</td>
<td>mtu</td>
<td>201 836</td>
<td>(b)</td>
<td>240 356</td>
</tr>
<tr>
<td>Wolfram cons</td>
<td>tonne</td>
<td>699</td>
<td>4 354</td>
<td>827</td>
</tr>
<tr>
<td>WO3 content</td>
<td>mtu</td>
<td>48 924</td>
<td>(b)</td>
<td>56 933</td>
</tr>
<tr>
<td>Uranium oxide</td>
<td>tonne</td>
<td>423</td>
<td>8 600</td>
<td>420</td>
</tr>
<tr>
<td>Vermiculite</td>
<td>tonne</td>
<td>716</td>
<td>45</td>
<td></td>
</tr>
<tr>
<td>Xenotime cons</td>
<td>tonne</td>
<td>7</td>
<td>5</td>
<td>12</td>
</tr>
<tr>
<td>Zinc ores and cons</td>
<td>tonne</td>
<td>846 348</td>
<td>135 491</td>
<td>869 138</td>
</tr>
<tr>
<td>Carbonat</td>
<td>tonne</td>
<td>431 815</td>
<td>(b)</td>
<td>450 745</td>
</tr>
<tr>
<td>Zinc in mine products</td>
<td>tonne</td>
<td>468 586</td>
<td>(b)</td>
<td>491 608</td>
</tr>
<tr>
<td>Zinc oxide</td>
<td>tonne</td>
<td>420 185</td>
<td>38 727</td>
<td>398 229</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total value</td>
<td></td>
<td>4 254 652</td>
<td>4 810 058</td>
<td>4 982 534</td>
</tr>
</tbody>
</table>

(a) Excludes Victoria.
(b) Included in value of mineral in which contained.
(c) Raw coal.
(d) Includes retorted gold, etc.
(e) Excludes WA.
(f) Excludes refinery production.
(g) Includes premiums for other recoverable metals.


Table 1.5 Australian Mineral production
Detailed analysis of particular aspects of each mineral product may be traced via sources of statistics.

These are the Bureau of Mineral Resources, Geology 2nd Geo-physics, the Australian Bureau of Statistics and the Joint Coal Board as well as the State Mines Departments.

Western Australian production statistics may be drawn from two sources: the Australian Bureau of Statistics and the Department of Mines Annual Reports. Total mineral production for the period 1975-1977 for Western Australia is presented as Table 1.6. Details of principal mineral producers for Western Australia and basic detail on location is available from such publications as Minerals, and Mineral Development produced by the Western Australian Department of industrial Development.

Fig. 1.2 shows selected mining operations in Western Australia.
COMMODITY STATISTICS

As in 1975-76, 1976-77 saw both significant increases and decreases in the production of the major minerals. Full scale production by recently-commenced mines in the Eneabba region resulted in rutile and zircon production rising by 51.9% and 49.0% respectively; while bauxite production rose 25.1% as a result of increased mining activity in the Dwellingup area. Significant decreases were recorded in the production of salt (down 10.7%) as a result of low demand, and crude oil (down 6.8%) due to steady depletion of reserves at Barrow Island.

The total value of minerals produced in 1976-77 was 14.3% higher than in the previous year. The values per unit of production of most minerals increased during the year, though most such increases were fairly moderate. In the case of iron ore, for example, which currently accounts for over 60% of the total value of minerals produced, the average ex-mine value per tonne increased from $7.19 in 1975-76 to $7.84 in 1976-77—an increase of 9.0%. There were, however, some deviations from this general trend. Among the major minerals, extreme cases were zircon (average value per tonne down 63.3%) and tin concentrates (average value per tonne up 43.3%).

<table>
<thead>
<tr>
<th>PRODUCTION, MAJOR MINERALS</th>
<th>1975-76</th>
<th>1976-77</th>
<th>Change (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bauxite</td>
<td>8 743</td>
<td>11 028</td>
<td>+ 26.1</td>
</tr>
<tr>
<td>Coal</td>
<td>2 157</td>
<td>2 339</td>
<td>+ 8.4</td>
</tr>
<tr>
<td>Crude Oil</td>
<td>12 413</td>
<td>11 569</td>
<td>- 6.8</td>
</tr>
<tr>
<td>Gold bullion</td>
<td>10 091</td>
<td>9 955</td>
<td>- 1.3</td>
</tr>
<tr>
<td>Ilmenite Concentrate</td>
<td>866 203</td>
<td>929 276</td>
<td>+ 7.3</td>
</tr>
<tr>
<td>Iron Ore</td>
<td>86 092</td>
<td>88 999</td>
<td>+ 3.4</td>
</tr>
<tr>
<td>Nickel Concentrate</td>
<td>471 662</td>
<td>450 224</td>
<td>- 4.5</td>
</tr>
<tr>
<td>Rutile</td>
<td>65 570</td>
<td>99 632</td>
<td>+ 51.9</td>
</tr>
<tr>
<td>Salt</td>
<td>4 512</td>
<td>4 031</td>
<td>- 10.7</td>
</tr>
<tr>
<td>Tin Concentrate</td>
<td>940</td>
<td>866</td>
<td>- 7.9</td>
</tr>
<tr>
<td>Zircon Concentrate</td>
<td>111 782</td>
<td>166 518</td>
<td>+ 49.0</td>
</tr>
</tbody>
</table>

EX-MINE VALUE, MAJOR MINERALS*

<table>
<thead>
<tr>
<th></th>
<th>1975-76</th>
<th>1976-77</th>
<th>Change (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>$1 000</td>
<td>17 613</td>
<td>+ 24.2</td>
</tr>
<tr>
<td>Crude Oil</td>
<td>$1 000</td>
<td>29 363</td>
<td>+ 2.1</td>
</tr>
<tr>
<td>Gold Bullion</td>
<td>$1 000</td>
<td>27 156</td>
<td>+ 2.0</td>
</tr>
<tr>
<td>Ilmenite Concentrate</td>
<td>$1 000</td>
<td>15 360</td>
<td>+ 13.4</td>
</tr>
<tr>
<td>Iron Ore</td>
<td>$1 000</td>
<td>619 802</td>
<td>+ 12.6</td>
</tr>
<tr>
<td>Rutile</td>
<td>$1 000</td>
<td>13 391</td>
<td>+ 54.2</td>
</tr>
<tr>
<td>Salt</td>
<td>$1 000</td>
<td>24 396</td>
<td>+ 16.5</td>
</tr>
<tr>
<td>Tin Concentrate</td>
<td>$1 000</td>
<td>3 178</td>
<td>+ 35.7</td>
</tr>
<tr>
<td>Zircon Concentrate</td>
<td>$1 000</td>
<td>15 692</td>
<td>- 5.4</td>
</tr>
</tbody>
</table>

EX-MINE VALUE, ALL MINERALS $1 000

<table>
<thead>
<tr>
<th></th>
<th>1975-76</th>
<th>1976-77</th>
<th>Change (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>995 672</td>
<td>1 137 757</td>
<td>+ 14.3</td>
</tr>
</tbody>
</table>

* Values for Bauxite and Nickel Concentrates are not available for publication

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Table 1.6 Western Australian Mineral production
### Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>COUNTRY ROCK</td>
<td>The barren or low grade rock formation that surrounds a mineral deposit.</td>
</tr>
<tr>
<td>GRADE</td>
<td>The classification of an ore according to the market value of the desired material it contains.</td>
</tr>
<tr>
<td>GROUND</td>
<td>A mineral deposit and the associated rocks.</td>
</tr>
<tr>
<td>METHOD OF WORKING</td>
<td>The system adopted to work or extract a coal seam or ore body. It includes all the operations involved in breaking, handling and transporting of ore and waste rock; support of ground; ventilation of workings and provision of supplies.</td>
</tr>
<tr>
<td>MINE</td>
<td>A word of wide application. It may be defined as a system of excavations made for the purpose of getting minerals. A ‘mine’ usually involves the employment of persons below ground.</td>
</tr>
<tr>
<td>MINING METHODS</td>
<td>The systems employed in the exploitation of coal seams and ore bodies. The method adopted depends on a large number of factors, mainly the quality, shape, size and depth of the deposit, accessibility and capital available.</td>
</tr>
<tr>
<td>OPENCAST</td>
<td>A mine working open to the surface similar to a quarry.</td>
</tr>
<tr>
<td>OPEN CUT</td>
<td>Opencast pits are started along an outcrop and continued downward until the thickness of over-burden prevents further economic exploitation. The operations are highly mechanized, and may be divided into:</td>
</tr>
<tr>
<td>OPEN-PIT</td>
<td>(1) removal of overburden and (2) removal of exposed coal or mineral.</td>
</tr>
<tr>
<td>ORE BODY</td>
<td>A mass of ore of any shape which may include low-grade and waste as well as valuable minerals, but is separate in form and character from the surrounding country rock.</td>
</tr>
<tr>
<td>OVERBURDEN</td>
<td>The surface waste or worthless rock overlying a flat or moderately inclined economic deposit, thin enough to warrant its removal to expose and mine the deposit by opencasting.</td>
</tr>
<tr>
<td>PROJECT</td>
<td>A design, plan or scheme to develop a mine or a section of a mine, or to reconstruct an existing one. The term is applied to any major scheme in the planning stage involving new shafts and installations.</td>
</tr>
<tr>
<td>QUARRY</td>
<td>An open pit, mine or excavation, where stone, sand, gravel or mineral is obtained from open faces, with or without a waste rock overburden.</td>
</tr>
</tbody>
</table>
CHAPTER 2 – Mining Law

CONSTITUTIONAL STATUS OF MINING LAWS

Mining Law is essentially a branch of the law of real property: it concerns the acquisition of property for extracting contained minerals, and the rights. Privileges and duties that fall upon the holder once the rights are required.

When examining the historical aspects of legislation reference should be made to the Commonwealth of Australia Constitution established in 1900 even though, fundamentally, the States retained legislative power both in regard to mining and property rights.

COMMONWEALTH POWER WITH REGARD TO MINING RIGHTS

A. The Commonwealth can legislate for lands under its exclusive control such as the Australian Capital Territory and other islands and Territories under exclusive Commonwealth jurisdiction.

B. Under Commonwealth provision for defence, the Federal Government has almost unlimited power to control national resources and manpower, to impose restrictions and embargoes.

The Atomic Energy Act relates to radioactive substances such as uranium, thorium, plutonium, neptunium and their compounds. The Act requires the discovery of these substances to be reported to the Commonwealth Minister who administers legislation. Consent is required before deposit may be worked and material disposed of.

C. The Commonwealth also has the power to acquire property and pay just compensation for property acquired. Thus in conjunction with defence powers acquisition and control of mineral rights and resources is possible.

Each Australian State has exercised its rights through legislative power by enacting laws dealing with mining, minerals and exploration.

Each Australian State also has laws and regulations regarding safety, health and labour conditions applicable to the mining industry; these are not discussed in this section.

EXAMPLES OF STATUTES FOR MINING AND EXPLORATION

Commonwealth:
- Atomic Energy Act 1953-1958
- Petroleum (Submerged Lands) Act 1967

Western Australia
- Mining Act 1904 (now repealed)
- Mining Act 1978 (proclaimed 1st January 1981)
- Petroleum Act 1967
- Petroleum Pipelines Act 1969
- Petroleum (Submerged Lands) Act 1967
MINES AND MINERALS

The term mine fundamentally refers to the place where the operation of mining is carried on. Specific definitions of the term applicable to each State appear in the relevant Mining Act.

A Mineral is all minerals other than gold or all precious stones, so that such substances as sand, clay and stone are within the category of minerals by mining legislation.

Under common law, minerals comprise every inorganic substance other than subsoil and layers of soil, which sustains vegetable life, which form part of the earth’s crust.

OWNERSHIP OF MINES AND MINERALS

Traditionally mines and minerals were part of the land which they occupied. Under common law the owner of the surface land was entitled to everything beneath the surface down to the centre of the Earth. The Crown had the prerogative right to all gold and silver found in mines, and this right was only gassed to an individual if the Crown granted it.

More generally it was possible either for the Crown to grant a Crown grant, or in any private sale of land, to reserve to the Crown or seller the right to all or any specified minerals, and also to sell or grant only a particular stratum of the land.

Thus ownership can be severed from the ownership of the surface, so that surface land rights when sold are a stratum of land to a certain depth. The actual depth of land varies between 15 to 30 metres according to different ownership and land location.

Under land title legislation in the various States a separate land title can be issued for surface lands, for mines or minerals under the surface and for a particular stratum of land held in separate ownership. Each individual land title either by mode of issue or by some endorsement will indicate whether or not it includes mineral rights, unless these are excluded from the sale or already reserved to the Crown.

Mining tenements (such as a lease) may be granted under State Mining legislation for surface lands or minerals, or both. The rights to mine and extract minerals need to be set out under lease conditions. Under most State legislation provision is made where, to a date prior to 1900, all Crown grants of land reserve to the Crown the minerals which might be found in or under the land. The rights to minerals in those lands belong to the Crown.

In summary the landowner (private person as opposed to the Crown) has control of land which he Owns to the extent of:

1. Reservation of some or all minerals by the Crown.
2. Gold and silver which belongs to the Crown unless divested.
3. Certain substances (petroleum and radioactive materials), which are divested from the individual Owner and vested in the State (Crown).
4. The Commonwealth provisions of defence which may restrict rights related to property.
5. The Actual Mining Acts of the particular State or Territory that impose particular constraints. These Acts can grant mining rights over Crown land as well as confer rights to prospect and mine privately owned land. However, the legislation endeavours to compensate the private landowner for the loss of proprietary rights by granting him a share of royalty in respect of minerals removed from his land and compensation for loss or damage suffered by him through the mining operations.
MINING ACTS – GENERAL

Legislation of Mining Acts deals mainly with mining on Crown land, but the legislation also confers certain rights to enter, prospect and mine on private land.

There are various types of licences, leases and other mining interests. First, there are authorities to enter and authorities to prospect, search and explore the land. These do not authorize mining and permit the removal of only limited quantities of substances for testing purposes.

Second, there are certain mining claims or tenements which entitle mining to take place, and these are a conglomeration of associated rights and holdings such as rights to erect residences, water rights and rights to erect machinery. Some of these are referred to as authorized holdings (see Page 2.3). This group of interests usually depends on the person who seeks to acquire them by virtue of a "miner's" right.

Third, the most valuable mining interests are those mining leases which are granted for a substantial period of years and are subject to further renewal.

The Mining Act is administered by the Minister, and the State is divided into mining districts (see Fig. 2.1), whose chief officials are mining wardens and mining registrars who have predominantly administrative functions. These are the persons responsible for the granting and administration of mining rights and various mining interests.

MINING ACT (1904 – 1978)

This act was repealed (cancelled) on the 1 January, 1981. However certain tenements and holdings under it will continue until termination of their tenure as if the Act had not been repealed. A feature of this Act was the relatively wide range of entitlements under the Miner's Right and same 39 types of mining tenements and authorized holdings.

Specific Transitional Provisions

These relate to certain tenements and holdings under the old 1904 Act.

Tenements and Holdings Involved

1. Temporary reserves. These expired on 30.6.82.
2. Gold mining leases, coal mining leases or mineral leases become mining leases and continue to date of expiry.
3. Mineral claims or dredging claims continue in force for two years then expire. Substitute application for the land within such holdings can be made
4. Miners' homestead lease, residential lease, residence area, business area or garden area continue in force for five years then expire. Prior to expiry the holder, after written application, may be granted a title under the Land Act 1933.
5. Machinery areas, tailings areas, quarrying areas and water rights continue in force for three years then expire if no substitute application is made.
6. Licence to treat tailings granted under the repealed Act remain in force or can be renewed for yearly periods where the ground is held overall as a mining tenement.
7. A prospecting area in existence will remain in force until the expiry of its term.

See Table 2.1 Conversion Chart for tenements and holdings.
CHAPTER 2 MINING LAW

2.4 (By kind permission of the Hon. Minister for Mines W.A.)

Table 2.1 Conversion chart for tenements and holdings from old to new act

MINING ACT (1978 – 1981)

Basic terms:

- "Crown land" includes reserves for mining, common and public utility and leases for timber or pastoral purposes. It does not include private land or other reserved land.

- "Minerals" includes gravel, shale (including oil shale), sand, clay, limestone, rock ore evaporites but does not include gravel shale (not being oil shale), sand, clay, limestone or rock when on private land.

- "Private land" is any land alienated from the Crown.

Crown Ownership

All gold, silver and any other precious metal existing in its natural condition on or below the surface of any land in the state is the property of the Crown.

Except in the case of land alienated in fee simple before the 1 January, 1899 all other minerals are the property of the Crown.
Where the minerals are the property of the Crown a mining title must be obtained from the Mines Department before any mining operations may be undertaken.

When land is not the subject of a mining tenement, the owner (or lessee) of the land may use any mineral existing in its natural state on the land for any agricultural, pastoral, household, road-making or building purpose on that land.

**MINERAL FIELDS**

For the purposes of the Mining Act 1978 – 1981 the State is comprised of a number of various mineral fields, some divided into districts. (See Fig. 2.1)

The location of the various mining registrars' offices are also given in Table 2.1.

<table>
<thead>
<tr>
<th>MINING REGISTRAR</th>
<th>MINERAL FIELD</th>
<th>DISTRICT</th>
</tr>
</thead>
<tbody>
<tr>
<td>Broome</td>
<td>West Kimberley</td>
<td></td>
</tr>
<tr>
<td>Bridgetown</td>
<td>Greenbushes</td>
<td></td>
</tr>
<tr>
<td>Carnarvon</td>
<td>Ashburton</td>
<td></td>
</tr>
<tr>
<td>Collie</td>
<td>Collie River</td>
<td></td>
</tr>
<tr>
<td>Coolgardie</td>
<td>Coolgardie</td>
<td>Coolgardie</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Kununalling</td>
</tr>
<tr>
<td>Cue</td>
<td>Murchison</td>
<td>Cue</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Day Dawn</td>
</tr>
<tr>
<td>Kalgoorlie</td>
<td>Broad Arrow</td>
<td>Bulong</td>
</tr>
<tr>
<td></td>
<td>East Coolgardie</td>
<td>East Coolgardie</td>
</tr>
<tr>
<td></td>
<td>North East Coolgardie</td>
<td>Kanowma</td>
</tr>
<tr>
<td></td>
<td>North Coolgardie</td>
<td>Kurnai pi</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Menzies</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Ularring</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Yerilla</td>
</tr>
<tr>
<td>Kununurra</td>
<td>Kimberley</td>
<td></td>
</tr>
<tr>
<td>Leonora</td>
<td>East Murchison</td>
<td>Lawlers</td>
</tr>
<tr>
<td></td>
<td>Mt Margaret</td>
<td>Mt Malcolm</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Mt Margaret</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Mt Morgans</td>
</tr>
<tr>
<td>Marble Bar</td>
<td>North Coolgardie</td>
<td>Niagara</td>
</tr>
<tr>
<td></td>
<td>Pilbara</td>
<td>Marble Bar</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Nullagine</td>
</tr>
<tr>
<td>Meekatharra</td>
<td>Murchison</td>
<td>Meekatharra</td>
</tr>
<tr>
<td></td>
<td>Peak Hill</td>
<td></td>
</tr>
<tr>
<td></td>
<td>East Murchison</td>
<td>Wiluna</td>
</tr>
<tr>
<td>Mt Magnet</td>
<td>East Murchison</td>
<td>Black Range</td>
</tr>
<tr>
<td></td>
<td>Murchison</td>
<td>Mt Magnet</td>
</tr>
<tr>
<td></td>
<td>Yalgoo</td>
<td></td>
</tr>
<tr>
<td>Norseman</td>
<td>Dundas</td>
<td></td>
</tr>
<tr>
<td>Perth</td>
<td>South West</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Phillips River</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Warburton</td>
<td></td>
</tr>
<tr>
<td>Southern Cross</td>
<td>Yilgarn</td>
<td></td>
</tr>
</tbody>
</table>

(By kind permission of the Hon. Minister for Mines W.A.)

Table 2.2 Mining registrars offices
MINERS' RIGHTS

A Miner's Right which is without restriction as to term, costs $10 and allows the holder to prospect on Crown land, take and keep as his property samples and specimens of ore or material up to 20 kilograms.

All Miners' Rights issued under the Repealed Act on or after the 8 December, 1978 continue in force for the purposes of the Mining Act 1978 – 1981, and the holders of these Miners' Rights are not required to reapply for a new right under the current Act. See Fig. 2.2.

Entry onto certain classes of Crown land is restricted (e.g. within a 400 metre radius of a pastoral lessee's well) other than for the purposes of marking off a mining tenement, posting notices or maintaining boundary marks in relation to tenements.

LAND OPEN FOR MINING

The are three categories of land open to mining:

- Crown land
- Public reserves
- Private land

A holder of a Miner's Right may prospect, mark out and/or apply for a mining tenement in respect of Crown land not being a granted tenement.
Crown land may be exempted from mining or from any specified mining purpose. Likewise the land may be restricted to certain types of mining tenements.

Public reserves does not include reserves for mining, common and public utility.

Restrictions on marking out and/or applying for a mining tenement in respect to reserved land are as follows:

A. Class "A" Reserves and "National Parks"
   Ministerial consent is necessary prior to prospecting or marking out in the South West Land Division, Esperance and Ravensthorpe Municipal Districts.

B. State Forests or Timber Reserves
   Conditions and restrictions as prescribed by the Forests Act 1918, apply.

C. Other Reserved Land
   Entry permits may be required prior to entering various other reserves and the appropriate authority should be contacted, e.g. Aboriginal Reserves.

D. Private Land
   Private Land may only be marked off by virtue of a permit to enter private land. Generally a mining tenement giving a right to the surface (or to within a depth of 30 metres of the natural surface) may only be granted with the written consent of the owner and occupier of that land. Until compensation has been resolved no mining shall be commenced on the surface of private land.

MINING TENEMENTS

A mining tenement may include all three categories of land open for mining, and all mining tenements apart from exploration licences must be marked out. The procedures for marking out are well outlined in Department of Mines (W.A.) Pamphlet No. 3: MARKING OUT AND APPLYING FOR MINING TENEMENTS.

Types of Mining Tenements available are:

- Prospecting Licence
- Exploration Licence
- Mining Lease
- General Purpose Lease
- Miscellaneous Licence

DETAILS OF TENEMENTS

Prospecting Licence
1. Any person may be granted a Prospecting Licence.
2. Area permitted up to 200 hectares.
3. Limit of 10 Prospecting Licences per person without further approval but a maximum of 20 overall.
4. Valid for a period of two years, then the land is not to be available for a Prospecting Licence to the same person.
5. Land held under a Prospecting Licence is not surveyed unless under dispute.
6. The holder of the Prospecting Licence has priority to take a lease over that land held by virtue of the Licence.

7. Transfer of a Prospecting Licence is restricted during the first six months of its tenure.

**Exploration Licence**
1. Any person may be granted an Exploration Licence.
2. Area permitted is to be not less than ten (10) square kilometres and not more than two hundred (200) square kilometres.
3. Land held by an Exploration Licence is not surveyed unless under dispute.
4. Exploration Licence is valid for a period of five (5) years.
5. The applicant for an Exploration Licence must lodge a security with the Department of Mines, Perth.
6. Transfer of an Exploration Licence is restricted during the first year of its tenure.
7. Conditional reduction in area held by an Exploration Licence:
   - At the end of the fourth year not less than 3 (half) of the remaining area to be surrendered. The residual land held should be discrete rectangular areas.
   - Land taken over from the Exploration Licence as a lease or general purpose lease is deemed to have been surrendered.
8. The holder of the Exploration Licence has priority to take lease/s over the land under licence.
9. A Prospecting Licence may be taken out over an area held by another party (Exploration Licence) conditional that minerals searched for are different.

**Mining Lease**
1. May be granted to any person.
2. Any person may be granted more than one Mining Lease.
3. Area held by a Mining Lease shall not exceed 10 square kilometres (1000 hectares).
4. Survey fees and rental must be paid on application for a Mining Lease.
5. The period of tenure for a Mining Lease is 21 years and may be renewed.
6. Specific conditions are given for damage or injury to the land surface.
7. A Mining Lease allows the mining and disposal of minerals contained within the land held by the lessee.

**General Purpose Lease**
1. Any person may be granted a General Purpose Lease, but it is granted in relation to a Mining Lease.
2. Any person may be granted more than one General Purpose Lease.
3. Area of a General Purpose Lease not to exceed 250 hectares.
4. Types of General Purpose Leases:
   - (a) Machinery
   - (b) Mineral or tailings storage
   - (c) Usage related to mining operations
CHAPTER 2 MINING LAW

2.10

5. A General Purpose Lease remains in force until the surrender or expiry or forfeiture of the mining lease in relation to which the General Purpose Lease has been granted.

Miscellaneous Licence

1. May be granted to any person in relation to a Prospecting Licence, Exploration Licence or Mining Lease but must be directly connected with mining operations.

2. The term of such Miscellaneous Licence is until the surrender, forfeiture or expiry of the Prospecting Licence, Exploration Licence or Mining Lease.

3. Types of Miscellaneous Licences:
   (a) A Road Licence
   (b) A Tramway Licence
   (c) An Aerial Ropeway Licence
   (d) A Pipeline Licence
   (e) Tunnel Licence
   (f) A Bridge Licence
   (g) A Water Licence
   (h) A licence for any prescribed purpose.

4. Conditions applicable are specified by the warden as well as prescribed under regulations.

<table>
<thead>
<tr>
<th>TYPE</th>
<th>MAXIMUM AREA</th>
<th>TERM (YEARS)</th>
<th>FEES</th>
<th>SECURITIES</th>
<th>MINIMUM ANNUAL EXPENDITURE</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>200 ha</td>
<td>2</td>
<td>60 c per ha (Min $10)</td>
<td>*$500 for each Prospecting Licence. No security required for the first Prospecting Licence.</td>
<td>$40 per ha (Min. $20000)</td>
</tr>
<tr>
<td>Prospecting Licence</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>200 sq km (Note: min 10 sq km)</td>
<td>5</td>
<td>$150 $18 per sq km</td>
<td>*$5000 for each Exploration Licence. *Note: Where these tenements are situated upon or encroach on reserved lands, such further sum as the Minister may determine.</td>
<td>$300 per sq km (Min. $20 000)</td>
</tr>
<tr>
<td>Exploration Licence</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>1000 ha</td>
<td>21 renewable</td>
<td>$15 $6 per ha Up to $61730 depending on area</td>
<td>Such sum as the Minister may determine, but only in respect to tenements situated on or encroaching on reserved land.</td>
<td>$100 per ha (Min. $500 if $5 or less other wise $10 000)</td>
</tr>
<tr>
<td>Mining Lease</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>General Purpose Lease</td>
<td>250 ha</td>
<td>Linked to parent mining lease</td>
<td>$15 $6 per ha Up to $61730 depending on area</td>
<td>Such sum as the Minister may determine, but only in respect to tenements situated on or encroaching on reserved land.</td>
<td>$100 per ha (Min. $500 if $5 or less otherwise $10 000)</td>
</tr>
<tr>
<td>Miscellaneous Licence</td>
<td>N/A</td>
<td>Linked to parent tenement</td>
<td>$6 per ha</td>
<td></td>
<td>(By kind permission of the Hon. Minister for Mines W.A.)</td>
</tr>
</tbody>
</table>

Table 2.3 Summary of mining tenements showing areas etc
CHAPTER 3 – Mineral Deposits and Exploration

MINERALS, ROCKS, ORES AND NON-METALLIC MINERALS

A mineral is a naturally occurring inorganic element or compound. Many minerals exhibit distinct crystalline habits or appearance and may be found as well formed crystals or as crystalline aggregates. Some minerals are amorphous or non-crystalline. While some 2000 mineral species are known only a small proportion are frequently encountered. The application of modern mineral detection and identification techniques often reveal the presence of mineral species in deposits of apparent simple composition (minerology).

A rock is an aggregate of grains or crystals of one or more minerals. Sane rocks are predominantly one mineral (non-mineralic) such as sandstone, but in general, rocks contain a number of minerals. The identification and classification of a rock is made on the minerals present, their relative abundance, and other features of the rock such as texture and origin.

Ore is then an accumulation of minerals or deposits from which one or more minerals can be exploited at a profit. Consequently the term ore is applied to a mineral deposit that is valuable for its metal content. Non-metallic deposits are valued for the minerals or compounds present such as rock salt (halite).

ROCK TYPES

Rocks are grouped under three main categories:

(a) igneous rocks formed by crystallization of magma;
(b) sedimentary rocks formed by surface processes of weathering and deposition;
(c) metamorphic rocks, which may have been derived from (a) or (b) but have undergone physical and/or chemical changes from heat or pressure or both.

Further subdivisions are possible such as for igneous rocks which have coarse crystalline texture, formed at great depth and described as plutonic; those of intermediate depth are hypabyssal while surface extrusions are fine-grained volcanics. Sedimentary rocks are subdivided into clastic sediments and chemical precipitates.
Figure 3.1 Classification of igneous rocks
### STRUCTURAL FEATURES OF ROCKS

Structural features include the orientation of rock strata as well as types of deformation and discontinuity within them. Included in these are features related to grain, texture, bedding or layering, which, collectively, may be referred to as primary; other features are related to rock deformation through stress.

Sedimentary rocks are usually deposited as horizontal beds which vary considerably in thickness. Bedding planes indicate breaks or interruptions in the depositional process and may be evident by changes in grain size and minerology.

Rocks, which are metamorphic in origin have been subjected to great confining pressure and exhibit flowage and re-crystallization. Minor flowage may cause only localized thickening, thinning or distortion of beds or folds. Greater stress may produce a lineation or foliation with reorientation of mineral grains or even complete re-crystallization of the rock material, forming new mineral species or forming layers or bands of them.

### DIP AND STRIKE
The strike of a bed is the bearing of a horizontal line on the surface of the bed. The dip of a bed is the direction and angle of inclination, measured from the horizontal at right angles to the strike. (See Fig. 3.3).
CHAPTER 3 MINERAL DEPOSITS AND EXPLORATION

FOLDS
When subjected to stress such as lateral forces layered sedimentary rocks commonly bend or warp and form folds. A convex upward fold is an anticline; a convex downward one is a syncline. (See Fig. 3.4).

AXIAL PLANE
The plane bisecting the fold is the axial plane. If the axial plane tilts or dips then the fold is said to be asymmetrical. If both sides or limbs of the fold dip in the same direction, the fold is overturned. If the axial plane is horizontal or nearly so, the fold is recumbent. The line of intersection of the axial plane with any bed in a fold is the fold axis; this may be horizontal or inclined.

The angle of inclination of the fold axis, measured down-slope in a vertical plane is its plunge. A fold whose axis plunges in opposite directions at its two extremities is double plunging; an anticlinal fold without elongation of this nature is a dome; a similar synclinal warp is a basin.

JOINTS
A joint is a fracture with no appreciable relative movement in the fracture plane. Such cracks occur from tensile stresses such as contraction while cooling, release of confining pressures, and tension associated with directional compression and bending. Joints exhibit patterns referred to as sets, which exhibit similar orientation: where two or more sets occur in an area they are referred to as a system.

CLEAVAGE
The term cleavage describes a well developed closely spaced parting of rocks: a response to stresses. Cleavages may be parallel but are commonly inclined to the bedding. Spacing between cleavage planes ranges from 1 mm to several centimetres.

FAULTS
A fault is a fracture or break in the rock along which there has been appreciable movement. The surface of the fracture is the fault plane. There are several fault types which are distinguished by the orientation of the fault plane or the relative movement of the rock units on either side of it. If the fault plane is inclined or dips at an angle of less than 45° to the horizontal, the fault is low angle. Faults with a dip greater than 45° are high angle. Where the fault plane is not vertical the block overlying the plane is called the hanging-wall block; the underlying block is the footwall. When the hanging wall block has moved downward relative to the footwall block the fault is normal; if upward the fault is reverse. A reverse fault whose plane of dip is very low angle is usually termed a thrust where movement-displacement is large. Where a fault exhibits horizontal movement it is referred to as a strike slip fault.

Figure 3.3 Diagram showing dip and strike
Figure 3.4 Common geological fold structures

Block diagram of anticlinal and synclinal folds

Monocline passing into normal fault at depth

Figure 3.5 Geological fold structures
CHAPTER 3 MINERAL DEPOSITS AND EXPLORATION

3.6

Figure 3.6 Normal and reverse faults

NORMAL FAULT

Fault plane

Hanging wall

Fault movement

Footwall

GROUND SURFACE

REVERSE FAULT

Figure 3.7 Normal and reverse fault block diagram illustrations

Normal fault

Reverse Fault

Figure 3.7 Normal and reverse fault block diagram illustrations
UNCONFORMITIES
An unconformity is a plane that separates dissimilar rocks of different origin, orientation or age. Where the orientation of layered rocks on either side of the unconformity is distinctly different, it is referred to as an angular unconformity. A break between sedimentary layers such as a time gap may be termed a disconformity.

Figure 3.8 Structural features in rock masses
The earth, viewed from the standpoint of geological history, has been in a constant state of change. Perhaps most evident and important are the geological processes which have affected the outer layer or crust of the earth. These are of two types: those which build up and those which wear down the crust.

The surface of the earth as we see it has been sculptured, largely by atmospheric agents. These processes are weathering (mechanical and chemical breakdown of rocks), and erosion (the mechanical and chemical transport of rocks from one point to another). Rocks react differently on exposure to the atmosphere depending upon their mineral composition: there is a softening and breaking down of aggregation as well as partial dissolution of mineral constituents.

Once broken down the rock material is carried away by wind, water or gravity to be deposited at a lower elevation.

Rebuilding of the crust occurs through magmatic intrusions at the base of the crust or beneath the crust in the mantle. Additional quantities of magma occur along linear beds within the major ocean basins. Solidified magma intrusions within the crust may be recognized as dykes (planar bodies parallel to the beds), stacks (bodies basically cylindrical, largest dimension vertical) and batholiths (large rock masses of considerable lateral extent). Extensive igneous activity occurs where outpourings from volcanoes are seen.

Folding is another process of building up of the earth's crust. Compressional forces cause rocks to yield by crumpling or folding resulting in shortening and local thickening. The magnitude and scale of folding is extremely variable. Large depositional troughs or
geosynclines are those which are subject to extensive forces and resultant movement or orogenesis (mountain building) may result.

Constructive processes also include rock failure or faulting. Extensive folding is usually accompanied by rock failure in tension and overthrust faults which result in one mass overriding another.

**ROCK FORMING MINERALS**

These minerals are of common occurrence and are estimated to constitute some 99% by weight of the earth’s crust.

The eight elements involved in the minerals are oxygen, silica, aluminium, iron, calcium, magnesium, sodium and potassium. These form such minerals as silicates, oxides and carbonates.

**ECONOMIC ORE MINERALS**

A large number of economic ore minerals occur, pre-dominantly as compounds but some as elements. These include:

- **Elements:** Gold, silver, copper
- **Oxides:** Hematite Fe$_2$O$_3$
  - Cassiterite SnO$_2$
- **Oxysalts:** Magnetite Fe$_2$Fe$_3$O$_4$
- **Sulphides:** Galena PbS
  - Sphalerite ZnS

These are examples of metallic types of mineral deposits.

**CLASSIFICATION OF ECONOMIC MINERAL DEPOSITS**

Economic mineral deposits may be classified under three schemes:

(a) by composition or content;
(b) by form;
(c) by probable origin.

(a) Classification by composition, either chemical or mineral content, with emphasis on use distinguishes mineral deposits worked for metal content (e.g. iron) from those worked for other components (e.g. coal, building material). Where worked for metal content deposits are appropriately referred to as ore deposits, though it is common practice to extend ore to non-metals. Sub-categories of content are those of:

(i) metalliferous deposits;
(ii) industrial or non-metallic deposits;
(iii) coal;
(iv) petroleum and natural gas.
(b) Classification based on form. Fundamentally these are of three types.

First, veins, which fill fissures, fractures or faults, generally transverse to the structural features of rock, although they may be parallel to bedding and appear as open space fillings.

Second, the mineral bearing body may have the form which indicates that the economic mineral or its associated waste mineral (gangue) was deposited from solutions that simultaneously dissolved away the country rock and laid down the economic mineral and its gangue giving a replacement. Graduations between veins that are fillings and/or replacements are possible. Replacements such as chimneys or pipes may be irregular in form.

The third form is that of disseminations which consist of minute to large units of economic mineral with or without an associated gangue mineral, scattered irregularly through the country rock.

(c) Categories according to origin. The most common classification in use is that based on theory of origin of the deposit. There is often debate on likely origins of some deposits.

1. Magmatic segregates. These originate from the accumulation of mineral particles or crystals during cooling and solidifying of the molten rock. Many chromite, ilmenite and magnetite deposits originate in this way.

2. Pegmatites. These are vein-like or lenticular (dykes) masses often exhibiting coarse crystal-line intrusions of igneous rock in texture and composition. Accessory minerals found in pegmatites are the industrial minerals such as mica, feldspar, beryl and tourmaline.

3. Contact metamorphic deposits. These occur at or near contact between intrusive igneous rock and the surrounding rock and are accounted to have been produced from two dissimilar rocks, vapours or liquids.

4. Dynamometamorphic deposits. These result from intense deformation, usually compressional, resulting in re-crystallization of the deformed rocks. Deposits resulting are asbestos, marble and gneiss.

5. (i) Hydrothermal veins and replacements. These are formed by precipitation from hot vapours or solutions; an example is the tin veins of Cornwall (U.K.).

(ii) Precipitation from cooler and more liquid solutions. These account for porphyry coppers, lead-zinc-silver veins and replacements, gold-silver quartz veins and some mercury deposits.

6. Sedimentary deposits. These originate from weathering processes at or near the earth’s surface. These include evaporates, biochemical extraction and precipitation e.g. bog iron (siderite) Physical concentration of solid particles such as gold placers is in this category. Limestones and sandstones are also included. Coal is of sedimentary origin.

7. Secondary deposits . Many deposits may be of low concentration; later processes, known as enrichment, which includes oxidation, result in potential economic deposits. Oxidation of, say copper sulfide, results in oxides and carbonates; where these elements are soluble they may be transported to greater depths and precipitated to result in secondary sulfide enrichment. (See Fig. 3.10).

A primary deposit may contain such large volumes of gangue as to be not mineable, but leaching away of gangue and residual accumulation results in laterites such as nickel and bauxite.
EXAMPLES OF MINERAL DEPOSITS

*Placers:* Placers or alluvial deposits are those accumulations of metal or mineral particles that have high specific gravity, and do not readily dissolve into solution.

Minerals recovered from placers:

<table>
<thead>
<tr>
<th>Elements</th>
<th>Mineral Compounds</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diamonds</td>
<td>Cassiterite</td>
</tr>
<tr>
<td>Gold</td>
<td>Chromiote</td>
</tr>
<tr>
<td></td>
<td>Ilmenite</td>
</tr>
<tr>
<td></td>
<td>Rutile</td>
</tr>
</tbody>
</table>

*Metalliferous Deposits*

Aluminium is obtained from near surface laterites (bauxite) which are residual accumulations from chemical weathering.

Iron ore is predominantly produced from chemically precipitated sedimentary deposits.

Copper is produced from porphyry deposits (disseminated) of characteristically low grade.

Nickel occurs as a segregate from mafic intrusives such as massive pentlandite-chalcopyrite-phrrhotite ore bodies.
Non-metallic – Industrial Minerals and Rocks
Rocks, whether igneous, sedimentary or metamorphic, may be used in their natural state as building stone or as crushed rock (concrete aggregate), or for contained minerals such as clays, salt, gypsum, asbestos, talc and abrasive minerals.

Coal
Coal is a combustible material originating from the accumulation and preservation of vegetable matter in a suitable environment. This accumulated material is layered with mud and sand, forming sandstones, shales and coal. The factors involved in coal formation are biochemical action, i.e. peat alteration to low rank coals, while confining pressure causes an increase in the fixed carbon content and an increase in the rank of the coal. The occurrence of extensive thicknesses of coal indicates that there were once ideal conditions for plant growth and vegetation accumulation. For commercial use, coal is often washed to remove ash and shale.

EXPLORATION – GENERAL
Exploration refers to all the activities and evaluation required before a sound decision can be made about the size and potential production of a new mineral deposit. The purpose of exploration is the discovery and acquisition of mineral deposits suitable for economic extractive operations at the present time or in the future. A prime objective of mineral exploration is one of economics, that of finding and acquiring a maximum number of mineral deposits for a minimum expenditure of cost and time.

Successful exploration shows that a mineral deposit has been proven within specific parameters to be an ore body of economic potential. Development, the stage which follows exploration, involves opening an ore body to further define reserves (grade and quantity) and to develop the prospect to a condition where production can begin. The final stage in this sequence from discovery of a mineral deposit, proving as an ore body, and mine development is full scale production.

For mining operators, development, i.e. further exploratory work, proving and opening up of the deposit, as a costing area is carried forward once the mine is in production. This is an essential part of an operation, because a mine is a wasting or diminishing asset. To continue to operate, additional reserves need to be established and prepared for extraction.

Exploration involves a wide variety of techniques from traditional prospecting to those collectively referred to as geotechniques - the application of highly sophisticated and expensive instrumentation and techniques.

Originally, search for mineralization by prospectors could be stated in terms of the grasshopper approach of examining outcrops and gossans. This leads to the discovery of many significant mineral deposits by prospectors, but the technique though often intensive is also random. An area may be extensively prospected but only for a specific mineral and the occurrence of other minerals or structures of potential importance may not be noted.

The regional approach to exploration has been developed in the search for covered or blind mineral deposits and to cope with large prospect areas. This has led to the development and use of techniques such as geophysics, geochemistry and three-dimensional sampling. Regional search however, requires understanding of the nature and characteristics of known mineral deposits, Thus the type of detectable ‘anomaly’ of either mineralization or host rock can be interpreted and used as an indicator for the detection of other possible mineralization.
EXPLORATION – STAGES

There are two main phases of exploration: Reconnaissance and Target investigation. Each of these phases may be broken into two stages.

The Reconnaissance Phase incorporates regional appraisal, that is selecting areas favourable for the occurrence of mineralization. This preliminary stage is followed by a detailed reconnaissance of favourable regions which may be established within the target areas. Target areas exhibiting suitable characteristics will enter phase two stage one: a detailed surface appraisal of the target area. Those areas which pass the criteria of surface appraisal are subject to three-dimensional sampling by drilling. Mineral occurrences which successfully meet the parameters set for stage four, may then be acceptable for development as an economic mineral deposit, now known as an ore body.

The actual stages required will depend upon geological knowledge and mapping available of the district or area. Often where a mineral deposit outcrops and is sufficiently promising then stage four is started almost immediately.

The basic stage of Reconnaissance Exploration, that of regional search is a widely accepted and Established trend. Most mining companies, having established regional exploration offices, set exploration targets in accordance with corporate objectives. To this end the resources of exploration techniques of geophysical, geochemical and geological methods are matched according to the conceptual model of the mineral search.

Close study of metallogenic information is preliminary. Metallogenic provinces are areas of varying size containing deposits of more or less similar type and composition suspected of having the same or related origin. Metallogenic epochs are times when favourable conditions of origin existed.

Data compiled from analysis of regional research may be presented in the form of base maps and relevant overlays to such maps.

Such basic data would include:

(a) basic geology;
(b) location of mines and prospects;
(c) surface covet and ‘outcrop occurrences;
(d) details of intrusive rock type and occurrences;
(e) sedimentary rocks;
(f) mineral claim holdings.

Map presentation should be clear, accurate and uncrowded. A great detail of basic data may be available from maps produced by Government Departments.

EXPLORATION METHODS

Exploration utilizes a number of techniques or ‘tools’ which include geology, geophysics, geochemistry, photo-geology and remote sensing methods. Geology has prime place because exploration depends upon the selection of the area to be searched. This may be a geological concept, such as extension of geological structures in which mineralization occurs, or similar considerations.
**Traditional Prospecting** consists of surface prospecting usually covering an area using limited equipment such as shovel, pick and dolly pot for crushing rock samples. Gold bearing rock or alluvial soil could be examined by panning, winnowing or dry blowing.

**Geological Methods.** Geology is the science dealing with the history of the earth. In this science there are a number of divisions of specialization, such as economic geology, mining geology and exploration geology, which are applicable to the mining industry. Geology is a prime tool in exploration. Its application includes the setting of target concepts; collection of geological data from the stages of exploration, especially for map (graphic) presentation; interpretation of data; correlation of information from sources such as geophysical and geochemical surveys; formulating judgement on further work on a region or target area or rejecting an area rather than carrying it forward to another stage of exploration. Exploitation (mining of ore bodies) can only be made to the best advantage when geological structures and characteristics are known.

**Geological Mapping.** Sound geological mapping is a basic requirement of mineral exploration. This ensures accurate presentation of geological information for interpretation, and communication of results from geological surveys, geophysics, geochemistry, drilling and other investigations.

Government Agencies such as the Geological Survey of Western Australia, Commonwealth Departments such as the Department of Natural Resources - Division of National Mapping as well as the Bureau of Mineral Resources, produce geological maps of Australia on a variety of scales. Maps cover such items as geology, mineral deposits, ground water, mineral industry and radiometric surveys. Mineral lease and mineral claim information plans are available from the Mines Department. Such maps may serve as basic geological maps for regional geology. Detailed maps are often available for certain areas and districts, especially those being developed as mineral districts and fields.

Field mapping by geologists utilizes aerial photographs where possible. When photogeological mapping is complete, the outline of basic structure and the planning of field work may be done in the office. Good photo interpretation requires geological skill as the work is based upon visual recognition of photographic tone contrast, textures, characteristics, shapes and patterns. Ground checking should be done to confirm interpretations.

Colour aerial photography has assisted in improving definition and contrast. Recent use has been made of ERTS (Earth Resources Technology Satellites) imagery; these satellites are now referred to as Landsat.

For detailed field mapping, large scale surface maps may be prepared from field work using hand instruments such as a Brunton Compass, Abney level and tape survey. However, most detailed maps can be compiled from photogrammetric base methods with infield verification. Structure and mineralization are significant in mapping because exploration often takes place in old mining districts. The geologist, when compiling detailed surface maps, will pay attention to the technique of outcrop mapping, i.e. plotting and showing what is observed rather than inferred.

A number of geological guides associated with mineral occurrences have been utilized to assist in locating ore extensions to known mineral districts. These guides include wall-rock alteration recognized by experienced observers, leached outcrops or examination of surface gossans. The latter are perhaps one of the earliest recognizable guides to mineralization occurrence.

The occurrence of sulfide ore mineralization (which is unstable to surface weathering) leaves a residual capping of limonite (hydrous iron oxides). However there are true
gossans overlying mineralization of possible economic potential. False gossans occur on fault zones.

**GEOPHYSICAL SURVEYS**
Geophysics can be defined as the study of the physics of the earth as related to its physical properties, composition and structure. This may be interpreted directly or indirectly to economic mineral deposits. The methods used in the field of geophysics are: gravimetric, magnetic, electrical, seismic and radiometric. Areas of possible interest are those which are irregular in relation to their surroundings. Geophysical information can be of a complex nature and require extensive experience for interpretation. Information may be collected by airborne, surface or sub-surface methods. Computer analysis and processing of data has increased the use of geophysics for exploration.

**Gravity**
Changes in geological conditions as indicated by changes in gravity are of prime interest in exploration. Gravity meters measure relative acceleration of gravity in terms of milligals (metres per sec per sec) while normal earth gravity is of the order of 9.8 m/s². Observations, when corrected and adjusted, are plotted on a gravity map and from anomalies geological conditions of possible interest can be inferred.

<table>
<thead>
<tr>
<th>METHOD</th>
<th>PHYSICAL PROPERTY</th>
<th>FIELD</th>
</tr>
</thead>
<tbody>
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<td>Gravity</td>
<td>Density</td>
<td>Earth's field of gravity</td>
</tr>
<tr>
<td>Magnetic</td>
<td>Magnetic susceptibility remnant magnetism</td>
<td>Earth's magnetic field</td>
</tr>
<tr>
<td>Electrical</td>
<td>Electrical conductivity</td>
<td>Earth's natural telluric currents</td>
</tr>
<tr>
<td></td>
<td>Magnetic permeability</td>
<td>Artificially applied electric field</td>
</tr>
<tr>
<td>Seismic</td>
<td>Seismic wave velocity</td>
<td>Artificially created seismic waves</td>
</tr>
<tr>
<td>Radiometric</td>
<td>Radioactive decay</td>
<td>Natural radioactive radiation</td>
</tr>
</tbody>
</table>

*Table 3.1 Geophysical methods*
Magnetic
The earth’s magnetic field varies from 0.35 gauss at the equator to 0.65 gauss at the magnetic poles. Susceptible ferromagnetic rocks become magnetised in relation to the earth’s magnetic field; some rocks show magnetic orientation independent of the earth’s field. This causes local changes in the magnetic field which may be detected by instruments termed magnetometers that measure variations in the order of one-five thousandth of the earth’s field. Variations in intensity can then be plotted and interpreted as to the probable distribution of magnetic rocks in the earth’s crust.

Electrical Methods
A wide variety of electrical methods utilize a range of electromagnetic wave frequencies of the electromagnetic spectrum. These are dependent upon physical properties of earth materials such as electrical conductivity. The electrical properties of rock are expressed in terms of specific resistivity in ohm-meters. Resistivity (specific resistance) is numerically equal to the resistance measured across opposite faces of a 1 metre cube of material. Rocks are mainly insulatory; conductivity depends upon porosity, water content and salinity.

One technique used is the natural electrical-field method which measures what is known as telluric current.

Resistivity and Induced Polarization Methods employ high frequency current and a variety of electrode arrays, whereby a current is caused to flow into the ground and the resulting voltage is measured at other positions. The decay time/residual potential is measured on terminating the current. Using this information from ground traverses, the geometry resistivity and chargeability of the anomalous volume can be delineated.

Electromagnetic Methods
Pure electromagnetic methods energize the ground by means of alternating current flowing in a transmitter coil. The resulting signal indication of ground response is detected by a receiver coil. A variety of hybrid systems are based on this approach.
Seismic Method
This method depends upon the velocity of sound in earth material. A short pulse of seismic energy e.g. an explosive shot, is detected by a series of geophones at distant points. A permanent record is made with milli-second graduations; this allows determination of the time interval after the pulse generation. In complex rock masses, interpretation is difficult and the seismic method finds its use in suitable conditions such as in sediments for petroleum exploration. It has little success in locating sulfide ore bodies, but is of value to hydrological studies in determining water table depth.

Radiometric Methods
Natural radiation from elements such as uranium and thorium can be detected in rocks by radiometric means. Because of the particles emitted, only gamma will penetrate sufficient coverage to be useful in exploration. Generally, for detection, outcrop concentration is necessary. Detection is by means of a scintillation instrument: a gamma ray activates a crystal and causes a light flash (scintillates) which is photo multiplied to give a pulse proportional to the radiation. Alternatively, the number of counts may be indicated by a total count meter.

Remote Sensing
This approach is based on the properties of objects, above absolute zero temperature, radiating electromagnetic energy. Natural objects display a characteristic spectrum of emitted, absorbed and reflected radiation, though it is often masked by interference. Use of satellites to detect micro-wave, infra red, visible and ultra-violet portions of the spectrum is being attempted. These methods have potential for increasing information that can be deduced from aerial photographs and satellite imagery.

Geochemical Surveys
Geochemical surveys normally consist of systematic collection and chemical analysis of a range of naturally occurring minerals, particularly ore minerals. Materials selected for sampling include rocks, soils, stream sediments, surface or ground water, vegetation and soil air. Base metals are usually most readily detectable, while for other accompanying minerals, anomalous concentrations may be indicative of potential mineral concentration.

The method has won wide acceptance especially in geochemical reconnaissance for covered and tropical areas. It is particularly valuable for:

(a) reconnaissance of ore types only recently of economic importance
(b) reconnaissance of low grade deposits of, say, weathered rocks where mineralization is inconspicuous and unrecognized
(c) surveying mineralized districts to detect possible hidden occurrences
(d) ground verification of geophysical anomalies
(e) appraisal of geologically interesting features.

The information compiled is presented on geochemical maps, but these cannot be interpreted directly to mineralization occurrence nor can they be directly related to drill planning. The basic information is first correlated with geophysical and geological data and may then form a guide for three dimensional sampling.

Interpretation will depend upon the nature and origin of the material sampled. Soil samples may be of residual or glacial origin, while others may be of soil air. Stream material such as sediments is widely used. Water sampling while used is not readily
indicative generally because of the low solubility of minerals. The use of vegetation may be useful, not only in terms of geo-botanical species occurrence on mineralized areas, but also because of plant absorption of mineral: leaf and twig material can be sampled and analysed.

The key to geochemical analysis is the detection of trace elements. Concentrations of metals in soils and vegetation is in the range of 10 to 100 p p m (parts per million). There are two basic methods used for analysis: colourimetry and atomic absorption.

Colourimetric detection of trace elements depends upon conspicuous and quantitative (colour density) changes in colour of an indicator solution in reaction to an element. Poor reproducibility is one of the hazards and it is also subject to human error.

Atomic absorption techniques are based on the atomization of a solution, containing the unknown constituents in a gas flame and the measurement of the absorption spectrum of light passing through the flame. The characteristic wavelength identifies the element and the degree of absorption is related to element concentration. Ability to reproduce results is high once calibration checks are run. Often specialized laboratory firms handle geochemical analysis at regional centres.

TESTING OF MINERALIZATION/ORE DEPOSITS

During the last stage of exploration, common to all successful ventures, reliable estimates of reserves including qualitative and quantitative features of grade and tonnage must be developed with minimum work and minimum cost.

To this end, three dimensional sampling is used to obtain basic data for assessment. However once the fourth stage has been satisfied, the deposit may be classified as an ore deposit and can then be taken to the next stage, i.e. preparation for production. This stage is known as development which will provide the background information on which the venture may proceed.

METHODS OF TESTING

EXPLORATORY DRILLING

Drilling is directed towards determining the presence or absence of the mineral target and gaining a preliminary idea of the possible grade and size of mineralization. Indirectly it tests stratigraphy, structure, wall rock alteration, geochemical zoning and other possible characteristics and guides. This will often mean emphasis on the elimination of high risk areas, using drilling methods offering high penetration rates and low costs, without absolute attention on method and sampling precision. Once preliminary, drilling shows favourable results, a more detailed and accurate drill sampling may proceed, i.e. development drilling The three commonly used methods of drilling for mineralization assessment are: diamond, rotary and percussion. Selection of the type will depend upon factors such as geometry of target, type of sample required, depth of hole and location of the drill hole. One essential requirement of all drilling is the close supervision of drilling and sampling so as to ensure the accuracy and reliability of the information obtained.

DIAMOND DRILLING

The diamond drill core provides a useful type of drill sample, because visual inspection and testing is possible; this may be of particular value where mineralization is disseminated. Problems encountered include possible poor core recovery in narrow soft, friable mineralized zones. This may necessitate sludge sampling along with recovered core. Penetration rates are relatively slow with rates of 13 metres per 8 hour shift considered good. Costs vary and are generally in excess of $50 per metre. Diamond drilling offers
mobility and flexibility; a high proportion of drilling of this nature is done by contract firms.

**ROTOR DRILLING**

Rotary drilling has a high penetration rate, is simple in operation, and is cheaper than diamond drilling.

Chip material of 6 mm to 10 mm from holes of 110 mm diameter may allow some geological and mineralogic features to be recognized. However structural relationships such as fracturing and veining are not evident in the sample. Drill cuttings may be collected by cyclone and split at the drill site for assay and geological/mineralogical testing. The principal disadvantages of rotary drilling are: most equipment is restricted to vertical holes; casing is difficult to set and for certain purposes there is lack of source information from chip samples.

**PERCUSSION DRILLING**

Percussion drilling may be of surface drifter type using sectional drill steel or down-the-hole hammer. Air flushing is used where possible, with sample collection by cyclone. Where water flushing is used, sludge samples may be collected at the hole collar.

**PITS, SHAFTS, ADITS, COSTEANS**

Where surface outcrops or near surface deposits occur, test information on grade, structure characteristics and data on in site volume may be obtained by pits, shafts and costeans. Often material is unbroken and not reduced to fines as in drill cuttings. These methods provide useful checks against drilling as well as providing bulk sample for metallurgical testing of mineral assemblage and subsequent problems of treatment. Shallow pits may be excavated by backhoe, while bulldozing may be used for large exposures for mapping, as well as sampling. Prominent surface deposits may be amenable to tunnelling (adit) to obtain bulk sample material and to provide geological information on structure, zoning and cut-off.
### Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>ANGLE OF DIP</td>
<td>The angle at which strata or mineral deposits are inclined to the horizontal plane. In most localities, earth movements subsequent to deposition of the strata have caused them to be inclined or fitted.</td>
</tr>
<tr>
<td>APPARENT DIP</td>
<td>The inclination of a stratified deposit to the horizontal in any downward direction other than the direction of true dip. The apparent dip is always less than true dip.</td>
</tr>
<tr>
<td>ASSAY</td>
<td>In general, the determination of the quantity of a desired metal per unit mass of the material containing it. The term assay is usually restricted to materials containing precious metals.</td>
</tr>
<tr>
<td>COSTEAN</td>
<td>A trench cut across the conjectured line of outcrop of a seam or ore body to expose the full width.</td>
</tr>
<tr>
<td>DEVELOPMENT</td>
<td>To open up a coal seam or ore body by sinking shafts and driving drifts as well as installing the requisite equipment.</td>
</tr>
<tr>
<td>FOOTWALL</td>
<td>1. The lower wall of an inclined or horizontal fault.</td>
</tr>
<tr>
<td></td>
<td>2. The junction of the ore body and the country rock on the lower side of the lode, i.e. the wall upon which the ore body may be considered to be resting.</td>
</tr>
<tr>
<td>GOSSAN</td>
<td>The decomposed or weathered upper part of an ore deposit. In the gossan region the minerals of the ore body at depth have altered into oxy salts.</td>
</tr>
<tr>
<td>GRADE</td>
<td>The classification of an ore according to the desired or worthless material in it or according to value.</td>
</tr>
<tr>
<td>GROUND</td>
<td>A mineral deposit and the associated rocks.</td>
</tr>
<tr>
<td>HANGING WALL</td>
<td>1. The upper wall of an inclined or horizontal fault.</td>
</tr>
<tr>
<td></td>
<td>2. The junction of the ore body and the country rock on the upper side of a lode.</td>
</tr>
<tr>
<td>LODE</td>
<td>The well-defined occurrence of valuable mineral bearing material in situ. Used synonymously with ore body and to some extent with reef and vein.</td>
</tr>
<tr>
<td>ORE BODY</td>
<td>A mass of ore of any shape which may include low-grade and waste as well as valuable minerals but is separate in form and character from the surrounding country rock.</td>
</tr>
<tr>
<td>OUTCROP</td>
<td>That part of a rock stratum, vein or coal seam that appears at the surface. It may be plainly visible or almost obscured by superficial deposits.</td>
</tr>
<tr>
<td>STRIKE</td>
<td>The direction of a contour line of an inclined ore body.</td>
</tr>
<tr>
<td>YIELD</td>
<td>The amount of valuable metal obtained per tonne of ore.</td>
</tr>
</tbody>
</table>
CHAPTER 4 – Drilling Methods and Techniques

DRILLING

PURPOSE
Drilling is defined as the mechanical means of placing a directed hole in rock. In the mining industry the two most common reasons for drilling are exploration and blasting.

EXPLORATION
In exploration, drilling provides a means of obtaining samples of rock below the surface either in the form of rock chips, sludge or cores. The samples obtained may be used to yield the following information:

- geology of the mineralized zone
- quantitative data on grade and tonnes of material
- physical size and shape of the deposit
- mineralized and metallurgical characteristics of the ore
- physical characteristics of the ore
- information on other relevant factors such as ground conditions and presence of ground water.

Hole sizes and lengths vary widely in exploration and depend on the material and type of deposit being investigated.

BLASTHOLE DRILLING
For blasting purposes, holes are drilled in the rock in a predetermined pattern to allow explosives to be placed within the confines of the rock to be broken. The charged holes are then fired in a sequence which breaks the rock and allows rapid excavation of the fragmented rock.

Underground blast-hole drilling is either short hole (0 - 4 metres) or long hole (4 - 50 metres).

In short hole drilling the diameter varies from 25 mm to 70 mm and the hole may be drilled in directions that suit usage.

Vertical down holes are used in shaft sinking, winzing and benching operations, while vertical up holes are used in raising, shrink stripping or stoping. Horizontal holes are used for horizontal development work such as driving and cross cutting.

Long hole drilling (hole sizes vary from 50 mm to 75 mm) is used extensively for stoping methods such as open stoping and sub-level caving.

In surface blast hole drilling, drill hole diameters vary from 75 mm to 380 mm and hole length will be slightly in excess of the bench height of the open cut. Common bench heights are between 10 and 20 metres. Holes may be drilled vertically or inclined up to 30° off vertical.
MECHANICS OF DRILLING
The functional components of a drilling system consist of four points: drill, rod, bit and circulating fluid.

Drill
The Drill is the prime mover converting energy from its original form (fluid, electrical, pneumatic or combustion engine drive) into mechanical energy which activates the system.

Rod
The rod transmits the mechanical energy from the drill to the bit.

Bit
The bit applies the energy in the system by mechanically attacking the rock to achieve penetration.

Circulating Fluid
This fluid cleans the hole, controls dust, cools the bit and at times may be used to stabilize the walls of the hole. The medium used may be either compressed air, water or drilling mud.

A drilling system has to perform two operations to achieve penetration into rock: fracture of the material and the removal of fractured material (cuttings). Causing the rock to break during drilling is a matter of applying sufficient stress with a tool (bit) to exceed the strength of the rock. The resistance of the rock to penetration is termed drilling strength.

PENETRATION
Penetration can be achieved mechanically in two ways: by percussion or rotary action. These two actions may be used to classify various methods of drilling, though there are other variations and combinations.

<table>
<thead>
<tr>
<th>Types of Drilling</th>
<th>Mechanical Action</th>
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<tbody>
<tr>
<td>Percussion</td>
<td>Percussion</td>
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<tr>
<td>Rotary drag</td>
<td>Rotary</td>
</tr>
<tr>
<td>Rotary roller</td>
<td>Rotary</td>
</tr>
<tr>
<td>Rotary percussion</td>
<td>Rotary plus percussion</td>
</tr>
</tbody>
</table>

The bit used in percussion drilling is a chisel-shaped or button-studded tool which impacts the rock with a hammer like blow, then rebounds to strike the rock again a controlled indexed distance away (radially) achieved by rotation of the bit. Thus, the stress effective in breaking the rock is applied by the impact blow in an axial direction in a pulsating manner.

The two predominant mechanisms in percussion drilling, crushing and chipping, are evident in crater formation.
The sequence in crater formation is:

(a) The rock is elastically deformed with crushing of surface irregularities.
(b) Main sub-surface cracks form, radiating downward from stress concentrations at the edges of the bit, and enclosing a wedge of material which is crushed.
(c) Secondary cracks along shear trajectories to the surface form large fragments or chips.
(d) Broken particles are ejected by the rebound of the bit and the cleaning action of the circulation fluid, resulting in the formation of a crater.

The sequence is repeated with succeeding blows with rotation/indexing providing additional free faces which aid rock breakage and increase crater size.

In drag bit rotary drilling, e.g. diamond drilling, the drag action at the cutting surface is supplied by two forces: the thrust (a static load acting axially) and the torque (the force component of a rotational moment acting tangentially). The thrust is responsible for indentation and the torque for shearing or ploughing.

The mechanism of penetration is achieved as follows:

(a) as the cutting edge of the bit comes in contact with the rock elastic deformation occurs;
(b) the rock is crushed in the high-stress zone adjacent to the bit;
(c) cracks propagate along shear trajectories to the surface;
(d) the bit rebounds and moves forward to contact solid rock again, displacing the broken fragments.

There is fundamental similarity between the two basic drilling systems, i.e. percussion or rotary because, in each case rock fails under mechanical attack via crushing and chipping.

In roller bit rotary drilling the bit provides a hybrid action of percussion and rotary action from the cutting teeth.

In rotary-percussion drilling, percussion is superimposed on a rotary system and higher impact forces are achieved.

**CUTTINGS REMOVAL**

This second phase of drilling operation is important if good penetration rates are to be achieved. If cuttings are not quickly removed from the face of the bit and out of the hole, they recirculate to the bit face and further crushing occurs resulting in finer particles and slower penetration rates.

In both rotary and percussion drilling the common flushing mediums are:

- Compressed air.
- Water.
- Mud.

Each has its own application, though for high penetration compressed air is the most effective.
PERCUSSION AND ROTARY DRILLING MACHINES

PERCUSSION DRILLS (PNEUMATIC)
There are two basic types of pneumatic percussion drills: the piston drill and the hammer drill.

Piston Drill
In the piston drill the drill steel is attached to the piston and reciprocates back and forth with it.

Hammer Drill
In the hammer drill, the piston or hammer reciprocates in a cylinder and strikes the drill-steel anvil block on its forward stroke.

There are two basic operating principles in the action of percussion hammer drilling. First, the principle that makes the piston reciprocate in the cylinder and second, that which makes the drill steel and drill bit rotate.

Piston movement is effected by a self-acting valve that admits compressed air at the proper instant first at one end of the cylinder and then at the other end.

Rotation of the drill steel is accomplished by one of four methods:

- Automatic rifle bar rotation.
- Integral independent rotation.
- External independent rotation.
- Manual rotation (now obsolete).

AUTOMATIC RIFLE BAR ROTATION
Rifle bar rotation is the most common method of rotation. Rifle bar rotated drills include:

- hand held jack hammers;
- stopers;
- jack drills and jack legs;
- mounted drifters.

Hand Held Jack Hammers
These are used primarily for drilling downholes and are usually hand held by the operator. These machines may be classified into classes according to weight and power. A hand held machine is the sinker drill of approximately 30 - 35 kg. See Figs 4.1 and 4.2.

Stopers
These machines, used in mining operations for drilling vertical or near vertical holes, are equipped with a rigid air feed leg. See Figs 4.3 and 4.4.

Jack Drill and Jack Leg (Air Leg)
This machine is especially designed for use with an air leg feed. It is used for horizontal development such as driving and cross cutting as well as some stoping methods, such as shrink stoping. See Fig. 4.5.
1. Machine Controls  
2. Exhaust ports  
3. Air leg control air passages  
4. Drill steel retainer  
5. Chuck lines  
6. Machine back head  
7. Valve assembly  
8. Piston/hammer  
9. Rifle bar pawl rotation  

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Figure 4.1 Rock drill with rifle bar and integral air leg control

1. Control knob; simultaneous control of percussion mechanism and water flushing  
2. Silencing cylinder  
3. Piston  
4. Attachment for connection of pusher key  
5. Rotation mechanism of ratchet wheel type  
6. Flush pipe  
7. Air passage for chuck lubrication  
8. Chuck  
9. Drill steel holder  

(Reproduced by kind permission of Atlas Copco Australia Pty Limited)

Figure 4.2 Silenced rock drill without rifle bar mechanism
(Reproduced by kind permission of Atlas Copco Australia Pty. Limited)

Figure 4.3 Stoper machine with differing feed lengths
PNEUMATIC STOPER ROCK DRILL FOR OVERHEAD DRILLING.
(STOPING, RAISER, ROOF BOLTING)

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Figure 4.4
Drifters
These machines are classified according to cylinder bore size and cover the range from 65 mm to 150 mm diameters.

Drifters are built with various types of automatic or mechanical feed and use hollow drill steel and can be used either as wet drills or with compressed air for cuttings removal. Drills of 100 mm bore and over are usually equipped with reverse rotation for handling of sectional (jointed) drill steel; this allows easy uncoupling of drill steel.

Hole sizes produced by this type of drill are of the order of 65 mm to 150 mm and hole depths up to 35 metres. Penetration rate varies with rock type, for soft rock, e.g. limestone it may be of the order of 25 to 50 metres per hour; for hard rocks e.g. granite, from 10 to 20 metres per hour.
INTEGRAL INDEPENDENT ROTATION
Integral Independent Rotation Drills reciprocate the piston in the same manner as the hammer drill, but depend on a separately driven rotation mechanism to rotate the drill steel. Bore size is between 100 mm to 150 mm. See Fig. 4.6

This type of drill permits the coupling of drill rods without piston operation. The rotation motor is controlled with an adjustable valve: an air driven gear motor transmits rotational power to the chuck through a gear reduction.

EXTERNAL INDEPENDENT ROTATION
This type of drill, which is the down the hole drill is constructed without in-built rotation (rotation comes from an external source) and the piston strikes directly on the bit. Compressed air which is used to operate the drill and clean the hole is passed through the drill stem that connects the drill to the mounting. Because the hammer follows the bit in the hole there is no energy loss through long lengths of drill rods. Hole sizes range from 90 mm to 200 mm in diameter. Drilling speeds in deep holes are higher than those attainable with other types of pneumatic percussive drills. See Fig. 4.7.

Figure 4.6 Section through pneumatic valve-less drifter with independent rotation

1. Rotation motor.
2. Drive gears.
3. Piston.
5. Steel (shank) adapter.

Figure 4.6 Section through pneumatic valve-less drifter with independent rotation
Figure 4.7 Down the hole drill (150mm)
PERCUSSION DRILLS (HYDRAULIC)

Hydraulic percussion drills operate by the intermittent application of high pressure oil to a double-acting piston: the frequency of application is controlled by oil entry and discharge ports, either uncovered by the movement of the piston, or by the action of a sliding or rotating valve or by a combination of both. The weight, shape and travelling speed of the piston determine the mode of energy transmission from the piston to the drill rod and bit. See Fig. 4.8. Current developments incorporate a wide variety of designs and features.

The total energy from the prime source per metre of hole drilled hydraulically may be approximately one-third of that for pneumatic percussive drilling. Another feature is the use of high impact frequency and lower impact amplitude: the hydraulic drifter offers
higher penetration rates without reduction in drill steel or bit life. Drill steel rotation is usually achieved by an independent rotation motor.

Emphasis in application has been on high performance drive or tunnel headings using multi-boom purpose built jumbos. An area of potential application is for long hole drilling underground particularly where high penetration rates may be beneficial. Crawler mounted quarry rigs are also being developed.

**TYPES OF MOUNTINGS**

There are a variety of mountings used in percussion drilling: small drills use air legs; drifter types use mountings that can be either relatively immobile (such as bar and arm) or self-propelled or self-contained. Bar and arm mountings are being replaced with self-propelled rigs which usually contain more than one machine.

Underground self-propelled mountings can be dealt with under two sections:

- units for horizontal development work;
- units for stope production drilling.

**Horizontal Development Units**

One of the most common of these units is the diesel-powered hydraulic-boom jumbo. These units consist of two or three drifters mounted on horizontal hydraulic booms, which allow the drills to be positioned in a horizontal direction by the central control operator. Compressed air from the mine air supply reticulation is used to power drills, drive hydraulic pumps and generate power for lighting. Hole lengths are usually 3 to 4 metres. See Fig. 4.9.

**Stope Production Units**

Here the drifters are mounted on booms so that the machines can operate in a vertical position. The hole which depends upon the stoping method, may be of the order of 6 to 35 metres. One or two machines are mounted on each rig.

Self-propelled mountings are the most common for on-the-surface machine operation. The type used depends upon the hole size, the terrain to be negotiated and the depth of holes.

A versatile rig of this type is the self-propelled crawler, a mount which utilizes a drifter type drill of 100 to 150 mm. bore size and tows the air compressor. Drill steel change is manual, drill cradle length is usually 3 to 4 metres and holes of 15 to 35 metres may be handled. (See Fig. 4.10). Compressor size required ranges from 16.9 m³/min (600 c f m) to 33.9 m³/min (1200 c f m).

Larger crawler mountings are self-contained. The motive power may be either diesel or electric, and hole sizes vary from 90 mm to 200 mm in diameter. Down-the-hole hammers are used, which, with tower heights of 8 to 16 metres, avoids frequent rod changing and give a depth capability of up to 200 metres depending upon size. The size of the machine used is a direct function of the hole size desired. Where possible the tallest tower consistent with stability, is used; rod change and handling is usually hydraulic/mechanical.

These machines are usually equipped with dust collection systems as well as engine/compressor air intake protection both for personnel health and safety. Where frequent moves are necessary, rubber tyred mountings are used. Weight limitations per axle for on-highway trucks limits machine capability to hole sizes between 125 mm to 175 mm.
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Figure 4.9 Universal jumbo (boom machine for underground mechanised development)
CHAPTER 4 DRILLING METHODS AND TECHNIQUES

4.14

a. Cradle mounted drifter  
b. Derrick mounted drifter

Figure 4.10 Crawler mounted drilling rigs

Integral Drill Rods

Chisel-bit drill steel

For peak-efficiency drilling under varying conditions, different types of drill steel have been developed. The conventional integral steel with a chisel bit is the most common and the most widely used. It is the easiest pattern to regrind and if correctly used will give the best drilling economy.

Four-point steel

If the rock being drilled is soft or fissured there is a risk of getting stuck. That's where the four-point steel is used. Its bit shape makes it less likely to get wedged in. The four-point steel is also recommended for mechanised drilling, where the driller cannot keep the same close watch on every steel as when operating a single rock drill.

Small-gauge steel

Small-gauge steels with a ⅛” rod and bit diameters of 27–29 mm are used when small-bore holes are wanted but ⅛” steels cannot be used. For better sludge disposal, these steels have a flank width that is equal to the width across-flats of the rod.

Nomenclature

1. Striking surface
2. Shank
3. Collar
4. Rod
5. Flushing hole
6. Carbide insert
7. Cutting edge
8. Wing
9. Bit

B1, Bit thickness
B2, Insert height
B3, Width of cutting edge
D1, Bit diameter
D2, Insert height
D3, Effective drill-steel length
L, Shank length
R, Cutting edge radius
α, Clearance angle
β, Cutting edge angle

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Figure 4.11 Integral drill steels

4.14
TYPES OF BITS AND STEELS

There are three general classes of drill bits used in percussion drilling: cross bit, chisel bit and button bit.

Cross bits have a sintered carbide cutting edge with the cutting edges oriented at either 90° (cross design) or at angles near 20° (X design) to each other.

Chisel bits are used for integral drill steel as for jack hammer class of drills. They are not effective in the larger hole sizes, because the rotation is not sufficient for an index that will turn the chisel fast enough for quick penetration. Moreover, wear of bit gauge is excessive, so that the type of bit is restricted to sizes below 45 mm. See Fig. 4.11.

The button bit is a percussion bit with a flat drilling face mounted with carbide spheres. The design partially solves the problems of changing worn bits because it does not require regrinding. Buttons wear from normal drilling; matrix wear also occurs exposing more of the carbide buttons. See Fig. 4.12 Detachable Bits.

Steels

There are probably only three types of drill steel in use: integral drill steel, sectional drill steel and steel tubing used for down-the-hole drills.

Integral drill steel is hexagonal in section with internal water passage, with a tungsten carbide insert at the bit end and a forged collar at the shank end. The steel used is a strong wear-resistant alloy steel.
Sectional drill steel (extension drill steel) is used with cradle-mounted drifters using detachable bits and couplings. Sizes of drill rod and types of rod/coupling treads vary considerably. See Fig. 4.13.

**ROTARY DRILLS**

**Principle of Operation**

Rotary drills attack rock with energy (in the form of rotating action and thrust) supplied to the bit by a rotating drill string. Supplementary energy is applied in oil well drilling by the impingement of the circulation fluid, usually drilling mud, on the bottom, through jets in the bit. The jetting action of air used in mining has little significance in the removal of the material from the solid but must remove cuttings before they are reground.

Sufficient thrust must be provided so that stresses induced by the teeth are sufficient to overcome the compression strength of the rock. Higher thrust loads provide deeper tooth penetration and, within limits, greater rock removal efficiency and lower drilling costs.

Thrust force is obtained by the weight of tools above the bit, controlled by either hydraulic cylinders, cable or chain pull-down.

**TYPES OF DRILLS**

**Rotary Blasthole Drill**

These units consist of the following components:

- a rotating device which provides the rotation for the drill string. This is usually by top rotation with energy supplied by a hydraulic motor;
- a mechanism for applying feed pressures to the bit, usually in the form of a hydraulic motor driving a chain pull-down arrangement;
- a rod changing device which is usually an automatic bottom change type. The derrick or tower may be equipped with a carousel for rod storage and handling allowing one-man operation;
- a compressor for supplying air for hole flushing;
- hydraulic levelling jacks for levelling the unit;
- a dust collecting system for collecting chips and dust as they are flushed from the hole;
- a power source such as diesel engine or electric motor.

The rotary blasthole rig is either crawler mounted or truck mounted. See Fig. 4.14.
Figure 4.14 Rotary blasthole drill (diagrammatic) hole size 171 to 251 mm diameter

Figure 4.15 Rotary drill bit terminology
BITS
Two basic types of bits are used in rotary drilling: drag bits and roller bits.

Drag Bits
Drag bit bodies are a casting or forging in which the cutting blades can be replaced when dull. These bits, often hard-faced or with sintered tungsten carbide cutters brazed to them, are suited to soft formations where shock loading is low. The general size range of bits is between 150 and 200 mm diameter.

Roller Bits
Roller bits are made for four general types of drilling conditions i.e. for soft, medium, hard and very hard ground formations. The roller cutters consist of three cones toothed for either soft or hard formations; for very hard formations the cones have tungsten carbide inserts. For soft formations, the cones have larger more widely spaced teeth and different cone centreing geometry from that used on a bit for hard formations. Cones are mounted on bearings which must withstand the thrust of the drill. Gauge loss on the teeth is reduced by hard facing on the outer surface of the cones and the shirt of the bit body. See Fig. 4.15.

Air passing through the bit is usually diffused through one or more centre openings or through jet nozzles near the periphery of the bit between the cones. Part of the air circulated is diverted through passages to the bit bearings to cool and keep them clean.
Cone type rotary bits do not require sharpening. The design of each bit type is balanced for the formation it is to drill. This balance of design provides uniform wear throughout the bit so that discard occurs when teeth are used up, and gauge loss and bearings are at point of reject. See Fig. 4.16.

**AUGER DRILLS**

This type of drill is basically a variety of rotary drilling using a drag-type bit with material recovery/removal by auger flights which are a helix or screw-like thread on the drill rods.

Drilling resistance, with or without a control led rate of feed, prevents the thread penetrating in proportion to its speed. The material is plough-scraped by the bit and lifted by the auger flights in a screw action.

Application of augers is usual to soft formations such as soils and soft sediments; they are also used for mineral testing and blasthole drilling. Mountings may be trailer, truck or tractor for rough terrain or pit stopes. Hole sizes are of the order of 100 to 150 mm diameter and the depths 15 to 45 metres. See Fig. 4.17.

**DIAMOND DRILLING**

In diamond drilling, a ring like bit, armed with small diamonds, rotates and cuts out a cylindrical core of the rock through which it passes. The bit is mounted on the end of a shaft consisting of hollow flush jointed rods which convey torque and thrust as well as the flushing medium.

Diamond drill holes may be drilled from the surface or from underground at any angle of inclination. Major use is sampling ore bodies as well as testing geological structures and rock defects. See Fig. 4.18.

The principal feature is a core of rock that is produced and accumulates in a core barrel which can be removed either by removing the rod string from the hole or by wire line retrieval of the inner core barrel.

**DIAMOND DRILLING EQUIPMENT**

Diamond drilling mechanics perform two actions: rotation of drill rods and exertion of pressure on the bit so as to feed and advance it in to the rock.

The motive power may be diesel, petrol or electric, but increased use of hydraulics has extended the use of electric power especially for underground purposes. See Fig. 4.19.

The feed or downward pressure can be applied by two methods: screw feed, hydraulic feed.

Both types control the advance of the bit by different methods.

Screw feed heads are equipped with sets of feed gears on the spindle and nut. This type of feed mechanism is almost completely replaced by hydraulic feed units. See Fig. 4.20.
Figure 4.17 Trailer mounted Gemco 210b auger drill
Figure 4.18 Small size diamond drill rig for exploratory drilling

1. Feed frame and stand.
2. Rotary drive.
3. Rod holder.
4. Control panel.
5. Power pack unit (electric or compressed air).

(Reproduced by kind permission of Atlas Copco (Aust) Pty.Ltd.)

Figure 4.19 Small hydraulic diamond core drill
The hydraulic head machine has a feed mechanism consisting of hydraulic cylinders which regulate the pressure exerted on the bit. Monitoring of actual pressure can be done from a pressure gauge.

Where rocks are of uniform hardness screw feed machines driven by compressed air have been used extensively underground for small diameter holes.

**Drill Rods**
The drill rod is a hollow flush-jointed or coupled rod that is rotated in the diamond drill hole with the core barrel reaming shell and bit attached. In operation, the drill rods, with the drilling tool assembly at one end and water swivel and water hose at the top end, are held in a chuck at the bottom end of the feed spindle. Rotation of the spindle turns the rod assembly and feeds the bit into the rock. A wide variety of couplings and adaptors is used to connect rods to fittings such as casing, to fishing (recovery) for tools and other size rods. Rod lengths of 3.0 metres or 6 m with random sizes are available.

Wireline rods, which allow retrieval of the inner core barrel through the hollow rod centre have slightly larger dimensions than conventional rods. Couplings are usually flush coupled.

**Core Barrels**
The core barrel is a cylindrical chamber for receiving and retaining the core as drilling progresses. The tool assembly consists of bit, core lifts and reamer which is designed to maintain a nominal hole diameter so that loss in bit gauge with an old bit does not jam the next new replacement. For this reason reamers are usually run in a batch.
Core barrels can be either rigid inner tube or rotating with single, double or triple tube. The double tube rotating type offers core protection because the inner tube remains stationary as the core enters it: flushing fluid passes outside the tube to prevent friable core material being washed out.

The triple tube core barrel consists of non-rotating holding tube inside of which is a split inner tube. This allows soft or broken material to be cored successfully, but face or bottom discharge bits must be used to prevent the flushing medium from eroding the core.

![Figure 4.21 Diamond drill bits (functional elements)](image1)

![Figure 4.22 Structural elements of diamond bits (surface set)](image2)
There are two groups of diamond bits - surface set diamond bits and impregnated bits. The surface set bits have whole stones embedded in the surface of the crown. Stones of 10 to 80 per carat are embedded in the surface of the crown in such a way that overlapping of the stones is assured. The bordering material (matrix) holding the diamonds to the bit blank is resistant to abrasion. Diamonds will lose their effectiveness through polishing, breaking or burning, though a large percentage can be reclaimed for further use. See Figs 4.21 and 4.22.

Surface set bits are the ones most commonly used in drilling; they are suitable for soft to medium-hard drilling formations; their variables are (in terms of bit specification) bit size and type, matrix hardness, diamond size and quality, and type of core barrel.

Impregnated bits contain diamond grit - usually of material consisting of 80 - 1000 per carat - uniformly distributed throughout the bit crown. During drilling the crown wears down, exposing new diamonds to the limits of the depth of the impregnation, and the diamond crown is consumed. Diamond recovery in this case is often extremely low.

Impregnated bits are used for medium to very hard formations as well as fractured rock where drilling conditions are difficult. The variable bit specifications are: bit size and type, matrix hardness, diamond size, and type of core barrel.

Drilling Diamonds
Diamonds are used as the cutting element because of their extreme hardness; however they fracture easily and are sensitive to high temperatures. The diamonds used for drilling are industrial grade usually of natural origin: synthetic diamonds are used for grinding. Broadly classified they are either unbroken material of good shape and size, broken material (grit) used for impregnated bits, and special tough property types, such as carbonados and ballos which are scarce and expensive.

DIAMOND BIT PERFORMANCE
Bit and diamond wear depend upon inclination of the hole, the type of coolant used, and uncontrollable factors such as ground hardness, abrasive characteristics and bedding. Therefore correct selection of bit type and quality is important for satisfactory cost-performance.

Specific drilling operation procedures should be used to avoid bad practices such as excessive pressure and rotation, dropping bits on hole bottom etc. The pressure and rotation speed applied to the bit are important and should be determined by the size of the bit, type of bit (surface set or impregnated) pressure and volume of circulating medium and control of rate of deviation of the hole.

It is also necessary to make a clear distinction between rpm and actual lineal travel of the faces of the bit because of their direct relation to cutting speed. In theory, doubling rpm suggests double penetration, but progress is limited by flushing. Cutting speeds must be kept within the practical limits determined by drilling conditions.

CIRCULATING FLUID
The prime function of the circulation fluid is to cool the diamond particles and the matrix. Its secondary functions are to remove the cuttings as quickly as possible to reduce abrasion of both the diamond particle and the metallic matrix, and to act as a lubricant for the bit face.
The three most common forms of circulation fluid used are:

- water;
- water plus miscible (soluble) oil;
- mud.

Water is the cheapest and most widely used in ideal ground conditions; soluble oil/water emulsion is used instead of water when re-circulation is necessary. This allows increased drilling rates and bit performance. Drilling muds are used in caving ground to maintain stable hole walls. Numerous mud mixtures - freshwater, bentonite, sodium lignosulfonate, caustic soda, diesel oil - have been developed for certain conditions.

1. Locking latch device - holds core barrel in place.
2. Spring set water flow valve.
3. Inner barrel bearing support.
4. Adjustable core barrel locator.
5. Core barrel.
6. Overshot - harpoon barrel assembly retriever.
7. Reamer shell.
8. Drill bit.

Figure 4.23 Wireline core barrel
Wireline Drilling
Wireline drilling is the term applied to the diamond drill system which uses drill rods and core barrel, a system which allows the inner tube of the core barrel to be raised to the hole collar or surface and the empty barrel to be pumped back into place without removing the drill string or diamond tools from the hole. The retrieval system consists of an overshot assembly and a wireline winch. See Fig. 4.23.

This method offers better drill utilization because it decreases down time in pulling the drill stem. Improved core recover and lower drilling costs are other features of the system. The core size obtained is small per hole diameter compared with conventional systems because of the removal of core up the centre of the drill rods.

Fishing Tools
Fishing tools are used to retrieve lost equipment such as rods, bits and casing, and to avoid expensive abandoning of poorly completed holes.

DEVIATION OF DIAMOND DRILL HOLES
Drill holes deviate from their intended course. The extent of deviation is dependent upon hole depth, inclination and character of the material being drilled and the rate and core of drilling.

Little or no surface indication of hole deflection is given to the driller. It is therefore essential to survey holes where reliable data and information on, and interpretation of, drill core intersections are necessary as drilling proceeds.

Vertical or steeply inclined holes tend to deflect less in dip than flat holes. Holes drilled in stratified or schistose rock deflect more than those in homogeneous rock. Lateral deflection which must be determined at times also occurs.

Deflection through use of wedging systems, may be deliberately done to re-align a hole, make additional intersections of a target or clear obstructions in a hole. There are a number of wedging systems available: a common retrievable one is the Clappison which offers rapid operation without loss of depth in the hole already drilled; it also has an orienting device for accurate placement and hole deflection.

DIAMOND DRILL STANDARDS
A broad range of standards apply to the commercial range of diamond drill tools and equipment such as barrels, casing and drill rods.
There are two world-wide systems which provide for standard dimensions of drill rods, core barrels and drilling tools for mining and exploration:
1. The metric system usually associated with “Craelius” equipment
2. The imperial systems such as the D.C.D.M.A. (Diamond Drill Manufacturers Association) U.S.A. and C.D.D.A. (Canadian Diamond Drilling Association).

Briefly, nominal size standards are:
### Conventional Drill Rods

<table>
<thead>
<tr>
<th>Size (O.D.) in mm</th>
<th>Originally</th>
<th>E</th>
<th>A</th>
<th>B</th>
<th>N</th>
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</thead>
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<tr>
<td>33.3</td>
<td>41.3</td>
<td>48.4</td>
<td>60.3</td>
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New series

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<thead>
<tr>
<th>Size (O.D.)</th>
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<th>AW</th>
<th>BW</th>
<th>NW</th>
<th>HW</th>
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<td>34.9</td>
<td>44.4</td>
<td>54.0</td>
<td>66.7</td>
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### Wireline Q Series Drill Rods

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<th>NQ</th>
<th>HQ</th>
<th>PQ</th>
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<tr>
<td>O.D. (mm)</td>
<td>44.5</td>
<td>55.6</td>
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<td>88.9</td>
<td>114.3</td>
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### Flush Coupling Casing

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<th>Type</th>
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<th>AX</th>
<th>BX</th>
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<tr>
<td>Size O.D. (mm)</td>
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### Bit Sizes

#### Q Series Wireline

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<td>60.0</td>
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<tr>
<td>NQ</td>
<td>47.6</td>
<td>75.7</td>
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<tr>
<td>HQ</td>
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<td>PQ</td>
<td>84.9</td>
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</table>

#### Conventional Diamond Drill Bits

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<td>EXT</td>
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<td>37.7</td>
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<tr>
<td>AX</td>
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<td>59.9</td>
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<tr>
<td>NX</td>
<td>54.7</td>
<td>75.6</td>
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</table>

#### Conventional Bits D.C.D.N.A.

**W Group “M” Design**

<table>
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<th>Type</th>
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<th>Bit Diameter (mm)</th>
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<tr>
<td>EWM</td>
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<td>AWM</td>
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<tr>
<td>BWM</td>
<td>42.0</td>
<td>59.9</td>
</tr>
<tr>
<td>NWM</td>
<td>54.7</td>
<td>75.6</td>
</tr>
</tbody>
</table>

*Note: these are the equivalent in size of the AX sizing.*
APPLICATION OF DRILLING EQUIPMENT

The method and equipment used for blasthole and exploration drilling must aim at producing the desired results at minimum cost. Incentive schemes for performance, i.e. distances drilled, and cost, are useful but may have adverse effects on the desired fragmentation from blasting or the amount of information from an exploration hole.

These adverse effects arise from:

- blastholes drilled off-line or to the wrong depth;
- exploration holes with poor core recovery or ineffective survey control of hole deviation.

Drilling methods and equipment may be examined in four main areas of drilling activity:

- surface exploration;
- underground exploration;
- surface blasthole drilling;
- underground blasthole drilling.

SURFACE DRILLING – EXPLORATION

The four main methods of surface exploration drilling are: churn drilling, diamond drilling, percussion drilling and rotary drilling.

Churn Drilling

This method uses a falling heavy bit to pulverize drilled rock into broken-crushed material which is then removed by bailing. This method is suited to large holes and varying ground but is expensive because of low performance. It is restricted to vertical holes and is rarely used except for water boring.

Diamond Drilling

Diamond drilling in the majority of cases produces a better sample for analysis, visual appraisal and testing than any other type of drilling method. Poor core recovery samples; however wireline drilling, in particular, has revolutionized recovery of core in fractured, altered or incompetent rock formations. The most severe limitation to the use of diamond drilling is the cost, usually of the order of $50 per metre (depending on hole size, depth and location) which is about 5 times that of other methods. However, diamond drilling is used extensively in modern exploration because in many instances it is the only suitable method of obtaining the desired results. Most exploration drilling below 150 m - 200 m is put down by diamond drilling: holes in the excess of 3000 metres have been drilled. The most frequently used rig today is the self-contained mobile rig, truck-mounted using wireline equipment.

Percussion Drilling

Initially wet percussion long holes were used to obtain samples. This method is only reliable and good cutting recovery obtained, if the ground is relatively homogeneous and unfractured and the hole is completely flushed between sample hole sections. When compared with diamond drilling, percussion drill penetration rate is higher with a much lower cost. Limited depth and difficulty in obtaining a sample useful for geological and metallurgical testing is a major drawback. Dry drilling offers high penetration rates and good sample recovery, however fractured and wet ground poses problems of sample recovery.
Rotary Drilling
The rotary method may use either the drag type bit or a roller core at the end of the rotating drill column. Drill cuttings are removed from the hole by air, water, mud or by reverse circulation through a partial vacuum inside the drill column. Sample separation is by settling tanks for liquid or by cyclone for dry drilling. Complete flushing for sample recovery with checks on sample volume per distance drilled is less necessary if sampling is to be reliable.

Current drilling rigs available may be used as rotary or down-the-hole hammer percussion drilling so that drilling method can be matched to suit rock type and ground conditions.

UNDERGROUND EXPLORATION
The majority of underground exploratory drilling is done by diamond drilling. In most cases the drilling site is located close to the projected target so that a minimum of dead drilling is involved; therefore location of faults, shears and wall rock alteration detectable from core may be of considerable importance. In some instances percussion drilling is used with sludge sampling; dry drilling is generally not permitted because of the associated dust hazard.

SURFACE BLASTHOLE DRILLING
The selection of surface mining blasthole drilling equipment requires an evaluation of a number of components of the overall mining operation. Aspects such as crusher size, shovel size, truck capability, screen capacity and drilling equipment need to be interrelated because fragmentation results depend to a large degree on effective blasthole drilling. In general, the larger the equipment the lower the operating cost consistent with production requirements; consequently the use of large rotary drills is almost universal in large surface mines.

Matching of equipment relates largely to hole size: surface mounted drifters or wagons or crawlers are suitable for holes of between 50 mm and 100 mm; down-the-hole hammer rigs are used for holes of 70 mm to 175 mm and rotary rigs for those that range between 150 mm to 380 mm in diameter.

For secondary blasting of oversize holes the crawler mounted drifter machine may be the most useful.

Factors related to the choice of the drill to be used include availability, abrasiveness and fragmentation of the rock. Some rock may drill exceedingly well, but its abrasive nature may cause bit gauge wear that reduces useful life. Fragmentation is important because it is desirable to drill the largest possible hole and use a consistent drill pattern. When the rock does not break, i.e. fragment, satisfactorily it may be necessary to drill smaller holes and use a closer drill pattern.

UNDERGROUND BLASTHOLE DRILLING
The cost of labour, as in all drilling, necessitates equipment which reduces the labour involved while providing high productivity; consequently, the trend is towards equipment where one operator can manage two or more drills.

Percussion drilling, using a drifter machine, is the most common method of drilling blastholes in underground hard rock mines. The hydraulic boom is the mounting used for shallow holes of 3 to 10 metres: a more rigid mounting is necessary to avoid excessive deflection when drilling deeper holes.
Down-the-hole drills, (pneumatic percussion) are finding increased usage underground. Initially this was for special purpose large diameter holes e.g. cable holes, drainage, borehole rising and the like. However increased use is being made of large diameter holes. In long-hole sub-level stoping, hole sizes range from approximately 80 mm to 100 mm diameter.

Rotary drilling common in coal and soft rock mines is used underground for raise boring pilot holes and back reaming of large diameter holes for ventilation openings; these holes range from 750 m to approximately 2.5 metres in diameter.
CHAPTER 5 – Explosives

INTRODUCTION
An explosive is a solid or liquid substance or mixture of substances which, on the application of a suitable stimulus to a small portion of the mass, is converted in a very short interval of time into other more stable substances, largely or entirely gaseous, with the development of heat and high pressure.

Gunpowder or black powder is the oldest form of explosive. It was first used in munitions and in mining in the 17th Century although it was not in general use until the 1830’s when the safety fuse was introduced.

Guncotton and nitro-glycerine were developed in 1846 but were not used until Nobel developed the fulminate detonator in 1867. Nitro-glycerine was made safe to handle by mixing it with an absorbent, hence the development of dynamite. Later, Nobel developed Blasting Gelatine, the most powerful of the industrial explosives, which is a gelatinous mixture incorporating 92% nitro-glycerine and 8% collodion cotton.

Lower strength explosives were later developed by adding an admixture of sodium and ammonium nitrate to nitro-glycerine.

Developments of note are:
(a) Low freezing type "Polar" grades.
(b) Permitted explosives for use in coal mines.
(c) Electric instantaneous and delay detonators.
(d) Detonating cord.
(e) Ammonium nitrate carbonaceous blasting agents [Nitrocarbonitrate (NCN) blasting agents].
(f) Water gel and slurry explosives (metallized).

Developments from 1930 to 1950 witnessed the utilization of blasting agents consisting of ammonium nitrate carbonaceous mixtures and their phenomenal growth in use in recent years. In the 1970’s there was a further development of these mixtures in the form of metallized slurries.

NATURE OF EXPLOSIVES
The desirable basic properties of an explosive for mining use are:

1. Substance be sufficiently insensitive to be safe under all conditions of storage and handling.
2. On initiation, be capable of rapid chemical change, yielding gaseous products resulting from exothermic reaction with a consequent rise in temperature and an increase in volume in comparison with the original substance.
3. Exothermic reaction with an increase in pressure.
4. Substance should be simple, inexpensive and derived from readily available material.
5. Substance should possess:
• adequate strength or power;
• high velocity of detonation;
• a density suited to application;
• water resistance - in cartridge form;
• good fume characteristics and is not subject to freezing or exudation of nitroglycerine at working temperatures.

**PROPERTIES OF EXPLOSIVES**

**COMPOSITION**
Most high explosives consist of mechanical mixtures of two or more main explosive materials with or without combustible or oxidising compounds and other additives.

Nitroglycerine and ammonium nitrate are the two most important basic ingredients.

**STRENGTH**
The strength or power of a gelatinous explosive measure of the quantity of energy released on detonation, i.e. the ability of the explosive to do useful work. It is normal to differentiate between weight (mass) strength and bulk (volume) strength. The former is determined from the deflection of a freely suspended ballistic mortar in which small explosive charges are fired as compared with those caused by exploding equivalent weights of blasting gelatine. The bulk strength is density related and may be calculated from:

\[
\text{Bulk strength} = \frac{\text{Weight (mass)} \times \text{density}}{\text{Density of blasting gelatine}}
\]

\[
= \frac{\text{Weight (mass)} \times \text{strength} \times \text{density}}{	ext{Density of blasting gelatine}}
\]

(See Tables 5.1 and 5.2).

**RATE OF ENERGY RELEASE**
This is closely related to the quality of fragmentation and blasting efficiency achieved from an explosive; fundamentally it is the gas produced and bore hole pressure developed.

**TYPES OF EXPLOSIVES**
A broad classification of explosives provides for two groups: permitted, i.e. those which may be used in gaseous fiery coal mines; non-permitted, i.e. those which may not be so used.

However, there are five main types of explosives:
• Initiating explosives
• High explosives
• Deflagrating or low explosives
• Ammonium nitrate carbonaceous (AN-C) or Nitrocarbonate (NCN) blasting agents
• Metallized slurry blasting agents.
### Table 5.1 Strengths of commonly used explosives

<table>
<thead>
<tr>
<th>Explosive</th>
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<th>Bulk Strength</th>
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<td></td>
<td>%BG</td>
<td>%ANFO</td>
<td>%BG</td>
<td>%ANFO</td>
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<td>100</td>
<td>325</td>
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<td>AN Gelatine Dynamite '95'</td>
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<td>86</td>
<td>234</td>
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<tr>
<td>AN Gelatine Dynamite '75'</td>
<td>88</td>
<td>120</td>
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<td>215</td>
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<tr>
<td>AN Gelignite '65'</td>
<td>83</td>
<td>112</td>
<td>76</td>
<td>193</td>
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<td>EXACTEX</td>
<td>71</td>
<td>101</td>
<td>43</td>
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<td>PLASTERGEL</td>
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<td>103</td>
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<td>SEMIGEL No 2</td>
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<td>ANFO (gravity loaded)</td>
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<td>ANFO (pressure loaded)</td>
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**VELOCITY OF DETONATION (V.O.D.)**

The velocity of detonation of any explosive depends on the degree of confinement and diameter of cartridge under test. An increase in confinement and/or diameter will appreciably increase the velocity recorded. V.O.D. measurements of I.C.I explosives are usually made by firing test cartridges unconfined.

V.O.D. of different explosives should be compared by using measurements taken using cartridges of the same diameter fired in the same state of confinement.

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Table 5.1 Strengths of commonly used explosives
CHAPTER 5 EXPLOSIVES

5.4

Table 5.2 Explosives cartridge counts and dimensions (per 25 kg case)

<table>
<thead>
<tr>
<th>EXPLOSIVE</th>
<th>STOCK PRODUCTS</th>
<th>PRODUCTS AVAILABLE ON SPECIAL ORDER</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dia (mm)</td>
<td>Length (mm)</td>
<td>Weight of cartridge or cylinder</td>
</tr>
<tr>
<td>AN Gelatine Dynamite '95'</td>
<td>25</td>
<td>200</td>
</tr>
<tr>
<td>55</td>
<td>520</td>
<td>1.6 kg</td>
</tr>
<tr>
<td>AN Gelatine Dynamite '75'</td>
<td>55</td>
<td>520</td>
</tr>
<tr>
<td>AN Gignite '60'</td>
<td>22</td>
<td>200</td>
</tr>
<tr>
<td>25+</td>
<td>200</td>
<td>125 g</td>
</tr>
<tr>
<td>36</td>
<td>200</td>
<td>1.3 kg</td>
</tr>
<tr>
<td>60</td>
<td>520</td>
<td>2.5 kg</td>
</tr>
<tr>
<td>'ANZITE' Blue</td>
<td>65</td>
<td>475</td>
</tr>
<tr>
<td>(in plastic 'GELOK' cylinders)</td>
<td>65</td>
<td>475</td>
</tr>
<tr>
<td>EXACTEX</td>
<td>17</td>
<td>500</td>
</tr>
<tr>
<td>32</td>
<td>200</td>
<td>215 g</td>
</tr>
<tr>
<td>SEMIGEL No 2</td>
<td>25</td>
<td>150</td>
</tr>
<tr>
<td>AJAX</td>
<td>22</td>
<td>200</td>
</tr>
<tr>
<td>'DYNAGE'</td>
<td>190</td>
<td>210 g</td>
</tr>
<tr>
<td>MORCOTE</td>
<td>190</td>
<td>200 g</td>
</tr>
<tr>
<td>'HYDRONEX' (in film)</td>
<td>32</td>
<td>300</td>
</tr>
<tr>
<td>'MOLANITE' 95 (in film)</td>
<td>55</td>
<td>410</td>
</tr>
<tr>
<td>65</td>
<td>410</td>
<td>1.6 kg</td>
</tr>
<tr>
<td>'MOLANITE' 110 (paper wrapped) (in film)</td>
<td>25</td>
<td>200</td>
</tr>
<tr>
<td>32</td>
<td>300</td>
<td>165 g</td>
</tr>
<tr>
<td>(plastic cylinders)</td>
<td>45</td>
<td>410</td>
</tr>
<tr>
<td>(in film)</td>
<td>55</td>
<td>410</td>
</tr>
<tr>
<td>(in film)</td>
<td>65</td>
<td>410</td>
</tr>
<tr>
<td>(in film)</td>
<td>80</td>
<td>600</td>
</tr>
<tr>
<td>(in film)</td>
<td>90</td>
<td>600</td>
</tr>
<tr>
<td>MOLANITE PLASTERPAK</td>
<td>115</td>
<td>710</td>
</tr>
<tr>
<td>130</td>
<td>600</td>
<td>12.5 kg</td>
</tr>
<tr>
<td>(in film)</td>
<td>200</td>
<td>600</td>
</tr>
<tr>
<td>'METABEL': 750cm x 25mm x 3.2mm</td>
<td>2.5 kg</td>
<td>10</td>
</tr>
<tr>
<td>(1.0 kg/ah)</td>
<td>6.3 kg</td>
<td>4</td>
</tr>
</tbody>
</table>

(NOMINAL FIGURES) (NOMINAL FIGURES)

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INITIATING EXPLOSIVES
These produce intense shock and are suitable for initiating high explosive charges, e.g. in detonators.

HIGH EXPLOSIVES
Depending upon their composition High Explosives have a velocity of detonation of 1 500 - 7 600 metres per second producing large volumes of gases and considerable heat at high pressures. High explosives consist of chemical compounds which are converted into gases by initiation. Products should be gaseous otherwise smoke is produced. The high explosives Nitroglycerine and ammonium nitrate possess the following typical chemical reaction:

$$2C_3H_5(NO_3)_5 + NH_4NO_3 \rightarrow 6CO_2 + 7H_2O + 4N_2 + 0_2$$

(at the temperature of reaction all products are gaseous)

DEFLAGRATING OR LOW EXPLOSIVES
These were the earliest type of explosives developed, and are now used mainly as propellants in ammunition.
Ignition is by open flame, e.g. safety fuse, and the explosion is a rapid form of combustion in which the particles burn on their exposed surfaces. Low explosives, e.g. blasting powder, are black powder or gunpowder, with little resistance to moisture. The commonest form is a mechanical mixture of potassium or sodium nitrate with sulfur and finely ground charcoal. Low explosives are used in mining for dimensional stone quarrying.

AMMONIUM NITRATE CARBONACEOUS OR NITROCARBONITRATE BLASTING AGENT
Nitrocarbonitrate (NCN) blasting agents are mixtures of a fuel and an oxidizer, intended for blasting (but not otherwise classified as an explosive) and in which some of the ingredients are classified as an explosive, provided that the finished product, mixed and packaged, cannot be detonated by a number 8 blasting cap when unconfined.

The most common form is ANFO (Ammonium nitrate fuel oil). The ammonium nitrate is relatively inactive and when sensitized by mixing with fuel oil in the proportion 94:6 by weight, detonates at a velocity of 3 000 - 4 000 metres per second. The cost of this type of blasting agent is approximately 1/3 of NG based explosives. Reaction is:

$$3\text{NH}_4\text{NO}_3 + \text{CH}_2(x) \rightarrow 7\text{H}_2\text{O} + \text{C}_0_2 + 3\text{N}_2$$

Additional strength may be produced by incorporating aluminium powder in the mixture.

METALLIZED SLURRY BLASTING AGENTS
These explosives, which may be either water gel or slurry, are available cartridge or in bulk for on-site mixing for surface mining. Bulk strength is higher than ANFO with V.O.D. of 4 000 metres per second. They contain no nitroglycerine and their main property is their high water resistance.

MECHANICS OF DETONATION
Explosives which react slowly, or deflagrate, e.g. black powder, are termed Deflagrating Explosives. High explosives detonate when properly primed and an explosion is initiated in them. Detonation, characteristic of high explosives, is the process of propagating shock waves through the explosive with a chemical reaction that maintains the explosion.

For effective control of high explosives a charge in a bore hole is activated by an initiating explosive in the form of a detonator or booster, either alone or in conjunction with instantaneous detonating cord. The detonator which is usually incorporated in a cartridge of explosive to form a primer, may be thermally ignited by safety fuse if it is a plain detonator or by electricity if it is electric.

Thermal ignition of the initiating explosive in the detonator is known as Detonation. Shock waves initiate the HE charge. Detonation is reinforced and propagated throughout the column of the main charge by rapid combustion (chemical reaction). However, each type of explosive has a minimum diameter of column below which detonation cannot be consistently propagated; this is termed critical diameter and has been used as an indicator of performance sensitivity. In the same way, the term Minimum Booster, which is the threshold amount of booster required to propagate a detonation uniformly through a charge, is a measure of sensitivity.
COMPOSITION OF EXPLOSIVES

High Explosives

Most high explosives consist of mechanical mixtures of two or more explosive materials containing combustible or oxidizing compounds. Nitroglycerine, on the other hand, is an extremely powerful, oily yellow liquid high explosive made from the action of nitric acid on glycerine. It is poisonous: any contact with the skin or inhalation of the vapour produces fracture headache. Solidified nitroglycerine burns in small quantities, but detonates violently when critical mass is reached. The liquid has two problems: first, it freezes at -13°C; second, it is unstable, shock sensitive and difficult to handle safely.

Polar, non-freezing explosives, which have one-fifth of the NG replaced by nitroglycol (a substance that freezes at −22°C) have overcome the first problem. The second problem has been solved by mixing the NG with absorptive mediums such as flour or wood meal; e.g., blasting gelatin, which is nitroglycerine absorbed in soluble nitro-cotton. Other additives such as sodium-, and ammonium-nitrate are used to reduce the strength of nitroglycerine. Cooling salts, such as sodium chloride are used in permitted explosives.

Many NG explosives are distinguished by the prefix AN, which means they contain nitroglycerine and ammonium nitrate. You should not confuse an explosive with AN-C agents which are dealt with later. The AN is portion of the main explosive as well as being the oxidant. AN explosives are hydroscopic and require care in wrapping, packing and storage: cartridges are end waxed and packaged in a fibreboard box which has a polythene liner.

Non-Permitted High Explosives

Gelatinous Explosives: (non-permitted)

(i) AN Gelatine Dynamite ‘75’, which has high strength, water resistance, high density and V.O.D., plasticity, and is suitable for hard, tight blasting in underground development.

(ii) AN Gelignite ‘60’ is a weaker modification with similar properties to the above.

(iii) Plastergel is a high strength, high velocity AN gelatine containing ingredients that ensure propagation of detonation at high velocity.

Plastergel is suitable for blasting such as secondary blasting without drilling because of its V.O.D.

Semi-gelatinous Explosives (non-permitted)

Semigel No 2

This is, one of the semi-gelatinous non-permitted explosives. It is a cheaper explosive than AN60 with a lower-density, water resistance and velocity of detonation.

Non-NG Explosives (non-permitted)

Molonite

Molonite is a detonator-sensitive slurry explosive available in varying strengths in cartridge form. Formulated from non-explosive ingredients, it has an equivalent strength and V.O.D. to NG explosives and possesses good water resistance.

Permitted Explosives

These are flameless explosives designed for use in coal mines because they will not ignite gas accumulations.

Permitted explosives are available in gelatinous and non-gelatinous form.
Ajax is a typical gelatinous explosive that is water resistant with high density and strength suitable for coal mine development headings and breaking up hard coal.

Morco is a semi-gelatinous explosive with medium water resistance, it is low density and high strength and is suitable for cut coal conditions.

**AMMONIUM-NITRATE CARBONACEOUS AN-C OR NITROCARBONITRATE BLASTING AGENTS**

Although AN-C was patented in 1867 by two Swedish scientists, and AN was used extensively in explosives to reduce strength and cost, AN-C was not used extensively as an explosive until 1955. The initial commercial development stemmed from a mixture of AN-ammonium-nitrate, a relatively insensitive oxidizing agent - and a combustible substance or sensitising agent such as carbon black, coal, molasses or fuel oil.

The most common additive used is 6% by weight of fuel oil, but good mechanical mixing is necessary for an effective reaction. Mixing is difficult because oil and AN do not blend easily; however, surface active agents may assist in forming a mixture. Sometimes a dye is used with the oil to check for effective mixing. Ineffective mixing results in poor blasting and the formation of oxides of nitrogen - red-brown fumes. One of the disadvantages of ANFO is that it is not resistant to water, hence it suffers loss in efficiency if used under wet conditions. Plastic sausages or tubes may be used under some conditions for hole loading to prevent deterioration by ground water. Specially designed AN-slurry mixtures in either cartridge or bulk form have been designed to be used for blasting wet holes.

Results rely upon correct and thorough mixing of ingredients.

The mixtures provide three main advantages: economy, efficiency and safety. Ammonium-nitrate and fuel oil are themselves not classified as explosives; therefore they can be handled in bulk prior to mixing which makes their handling costs considerably less than conventional NG explosives. Controlled conditions ensure high gas generation and efficient fragmentation. The stability and low sensitivity of AN-FO mixtures makes them safe to handle. Mixtures can be managed by dissolving the ammonium-nitrate with water.

**ANFO PERFORMANCE**

This depends upon mixing, size, shape, grading, hardness, porosity and moisture resistance of particles of AN whether or not they have been treated with surface agents. Prilled AN i.e. AN in the form of small, roughly spherical prills lowers charging density but provides better oil absorption and improves initiation and blast performance.

Important parameters for efficient use of ANFO are:

- diameter of bore hole charge;
- loading density of charge;
- degree of confinement;
- absence of excess moisture;
- correct mixture of sensitiser and AN;
- strength, shape and position of primer or booster.
DETONATORS

High explosives and AN-C blasting agents are designed to be relatively stable for handling, safety, transport and storage - therefore, initiation of high explosive or blasting agents requires a powerful localized shock or detonation. This is accomplished by means of a detonator incorporating a cartridge of HE to form a primer or a prefabricated initiation unit booster.

TYPES OF DETONATORS AVAILABLE

No 8 plain (aluminium):

No 6 electric:

- instantaneous (copper or aluminium)
- half-second delay (aluminium or copper)
- short delay - Carrick (copper)

No 8 electric:

- instantaneous (aluminium)
- half-second delay (aluminium)
- short delay L series
- non-electric detonators

No 8 star electric:

- instantaneous (aluminium) (seismic blasting)
- half-second delay (aluminium)
- short delay (aluminium) L series
- embedded detonators (shaft sinking harnesses)

Note the numbering system relates to their respective strength, in terms of PETN base charge. The No 6, 8 and 8 star have base charge weights of 0.22, 0.45 and 0.80 g roughly with comparative strengths of 1, 2 and 4, See Table 5.3 Electric Detonator Identification.

All detonators consist of a metal tube of either copper or aluminium, closed at one end, into which is pressed a base charge of PETN (pentaerythritol-tetranitrate). On this is superimposed a mixture of sensitive explosive (usually ASA a lead azide aluminium powder mixture) to prime the base charge.

Plain Detonators

Plain detonators are the simplest type used for general purposes, especially where single charges are fired. Detonators are ignited by means of a safety fuse inserted into the open end and fixed into position by crimping the mouth of the tube to the fuse. After crimping, they may be dipped in a sealing compound for water-proofing and protection from the loss of sensitivity that usually occurs in damp storage conditions. Some bench crimping machines give a double roll crimp, obviating the need to dip seal the detonator end of the assembled rod.

No 6 detonators were in general use, but have been discontinued and No 8 plain detonators are now used. Detonators of the plain variety are packed in red cardboard boxes of 100. See Fig. 5.1 Plain Detonators.

Electric Detonators (Instantaneous)

These are similar in general construction to the plain detonator except that an electric fuse head is provided: the mouth of the detonator tube is sealed with a neoprene plug through which plastic-covered lead wires pass. See Fig. 5.2 Instantaneous Electric Detonators.
With the passage of an adequate electric current (300 milliamps), the bridge wire fuses, becomes incandescent and ignites the igniting compound and, in turn, the flashing compound and the priming charge. For practical purposes, these reactions are considered to be instantaneous. Plastic-covered lead wires of 1.4, 1.8, 3.0 and 3.6 metres are available: they are folded to prevent kinking when extended during placement and charging.

**Electric Detonators (Delay)**

Half-second delay aluminium electric detonators (generally used in underground development work) have a longer tube than plain electric detonators because a delay element lies between the fuse-head and primer charge. The delays are from 0 to 12 with a nominal delay of half a second (500 milliseconds) between each. The delay number is tagged on the lead wires.

<table>
<thead>
<tr>
<th>DETONATORS</th>
</tr>
</thead>
<tbody>
<tr>
<td>ELECTRIC DETONATOR IDENTIFICATION</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Tube</th>
<th>Base Mark</th>
<th>Description</th>
<th>Lead Wire Details</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Copper Wire Guage</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>PVC</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td>Half Second, No. 8 Star (Protected)</td>
<td>23 SWG</td>
</tr>
<tr>
<td>Al/Cu</td>
<td></td>
<td>Short Delay, No. 8 Star L Series, (Protected)</td>
<td>23 SWG</td>
</tr>
<tr>
<td>Cu</td>
<td>C3</td>
<td>Short Delay Carrick, No. 6 (Protected)</td>
<td>23 SWG</td>
</tr>
<tr>
<td>Cu</td>
<td></td>
<td>Instantaneous No. 6 (Non-Protected)</td>
<td>25 SWG</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td>Instantaneous No. 8 (Non-Protected)</td>
<td>25 SWG</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td>Instantaneous No. 8 Submarine &lt;=2 m (Protected)</td>
<td>25 SWG</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td>Instantaneous No. 8 Submarine &gt;2 m (Protected)</td>
<td>23 SWG</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td>Instantaneous No. 8 Star Seismic (Protected)</td>
<td>21 SWG</td>
</tr>
</tbody>
</table>

Notes:
(i) In the above table, detonators are referred to as No. 6, No. 8 and No. 8 Star. These detonator strength numbers relate to the weight of the base charge of pentaerythritol tetranitrate (PETN) as follows: No.6 ............ 0.22 g PETN  No.8 ............ 0.45 g PETN  No.8 Star ............ 0.80 g PETN
(ii) The term “Protected” refers to an insulating sleeve either fitted over the fusehead or interposed between the fusehead and the detonator tube wall plus a sleeve over the shorted ends of the lead wires. This electrical insulation obviates any build up of low energy static to discharge in the narrow neck of the detonator away from the fusehead. No protection is afforded against a high energy discharge such as a lightning strike.
(iii) The numbers in the column under ‘Base Mark’ indicate the detonator delay number in the delay series, eg H5 indicates delay No 5 Half Second Delay.
(iv) Detonators with copper shells, ie No 6 instantaneous, Carrick Short Delays and copper No 8 Star L Series are used mainly in coal mining.

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Table 5.3 Electric detonator identification

5.9
CHAPTER 5 EXPLOSIVES

5.10

Figure 5.1 Plain Detonators

Figure 5.2 Instantaneous electric detonator components

5.10
Short delay electric detonators generally used in multi-hole blasting are a development of half-second delay detonators: element is a very much shorter fast-burning pyrotechnic. Short delay detonators are available either as ‘L’ series with aluminium takes or Carrick Delays with a copper tube. Delay intervals are a nominal 25 milli-seconds between ‘L’ series detonators. See Figure 5.3 Typical Short Delay Electric Detonator and Table 5.4 Nominal Delay Times-Electric Detonators.
### 'L' SERIES SHORT DELAYS

<table>
<thead>
<tr>
<th>Delay No</th>
<th>Nominal Delay Time (ms)</th>
<th>Nominal Interval (ms)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>30</td>
<td>25</td>
</tr>
<tr>
<td>2</td>
<td>55</td>
<td>25</td>
</tr>
<tr>
<td>3</td>
<td>80</td>
<td>25</td>
</tr>
<tr>
<td>4</td>
<td>105</td>
<td>25</td>
</tr>
<tr>
<td>5</td>
<td>130</td>
<td>25</td>
</tr>
<tr>
<td>6</td>
<td>155</td>
<td>25</td>
</tr>
<tr>
<td>7</td>
<td>180</td>
<td>25</td>
</tr>
<tr>
<td>8</td>
<td>205</td>
<td>25</td>
</tr>
<tr>
<td>9</td>
<td>230</td>
<td>25</td>
</tr>
<tr>
<td>10</td>
<td>255</td>
<td>25</td>
</tr>
<tr>
<td>11</td>
<td>280</td>
<td>25</td>
</tr>
<tr>
<td>12</td>
<td>305</td>
<td>25</td>
</tr>
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<td>13</td>
<td>335</td>
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<td>14</td>
<td>365</td>
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<td>15</td>
<td>395</td>
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</tr>
<tr>
<td>16</td>
<td>425</td>
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</tr>
<tr>
<td>17</td>
<td>455</td>
<td>30</td>
</tr>
<tr>
<td>18</td>
<td>485</td>
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<td>19</td>
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<td>21</td>
<td>575</td>
<td>30</td>
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<td>22</td>
<td>605</td>
<td>30</td>
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<td>23</td>
<td>635</td>
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<td>24</td>
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<td>27</td>
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<td>28</td>
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<tr>
<td>29</td>
<td>815</td>
<td>30</td>
</tr>
<tr>
<td>30</td>
<td>845</td>
<td>30</td>
</tr>
</tbody>
</table>

Delays 0 to 18 are stock items in all States. Delays 19 to 30 are imported against firm order in regular shipments from Nobel’s Explosives Co., Scotland. Advice on delivery lead times is available from State Sales Offices.

### CARRICK DELAYS

<table>
<thead>
<tr>
<th>Delay No</th>
<th>Nominal Delay Time (millisecond)</th>
<th>Nominal Interval (millisecond)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>30</td>
<td>25</td>
</tr>
<tr>
<td>2</td>
<td>55</td>
<td>25</td>
</tr>
<tr>
<td>3</td>
<td>80</td>
<td>25</td>
</tr>
<tr>
<td>4</td>
<td>105</td>
<td>25</td>
</tr>
<tr>
<td>5</td>
<td>135</td>
<td>30</td>
</tr>
<tr>
<td>6</td>
<td>165</td>
<td>30</td>
</tr>
<tr>
<td>7</td>
<td>195</td>
<td>30</td>
</tr>
<tr>
<td>8</td>
<td>230</td>
<td>35</td>
</tr>
<tr>
<td>9</td>
<td>260</td>
<td>35</td>
</tr>
<tr>
<td>10</td>
<td>300</td>
<td>35</td>
</tr>
</tbody>
</table>

### HALF SECOND DELAYS

<table>
<thead>
<tr>
<th>Delay No</th>
<th>Nominal Delay Time (ms)</th>
<th>Nominal Interval (ms)</th>
</tr>
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<tbody>
<tr>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>1</td>
<td>500</td>
<td>500</td>
</tr>
<tr>
<td>2</td>
<td>1000</td>
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<td>3</td>
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<td>500</td>
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<tr>
<td>4</td>
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<tr>
<td>5</td>
<td>2500</td>
<td>500</td>
</tr>
<tr>
<td>6</td>
<td>3000</td>
<td>500</td>
</tr>
<tr>
<td>7</td>
<td>3500</td>
<td>500</td>
</tr>
<tr>
<td>8</td>
<td>4000</td>
<td>500</td>
</tr>
<tr>
<td>9</td>
<td>4500</td>
<td>500</td>
</tr>
<tr>
<td>10</td>
<td>5000</td>
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<tr>
<td>11</td>
<td>5500</td>
<td>500</td>
</tr>
<tr>
<td>12</td>
<td>6000</td>
<td>500</td>
</tr>
</tbody>
</table>

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Table 5.4 Nominal delay times – Electrical detonators
Table 5.5 Anoline and cordline delays (non-electric detonators)

<table>
<thead>
<tr>
<th>Delay No</th>
<th>'ANOLINE' &amp; 'CORDLINE' SHORT DELAYS</th>
<th>'ANOLINE' LONG DELAYS</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Nominal Delay Time (ms)</td>
<td>Nominal Interval (ms)</td>
</tr>
<tr>
<td>0</td>
<td>not made</td>
<td>—</td>
</tr>
<tr>
<td>1</td>
<td>30</td>
<td>30</td>
</tr>
<tr>
<td>2</td>
<td>50</td>
<td>25</td>
</tr>
<tr>
<td>3</td>
<td>75</td>
<td>25</td>
</tr>
<tr>
<td>4</td>
<td>100</td>
<td>25</td>
</tr>
<tr>
<td>5</td>
<td>128</td>
<td>28</td>
</tr>
<tr>
<td>6</td>
<td>157</td>
<td>29</td>
</tr>
<tr>
<td>7</td>
<td>190</td>
<td>33</td>
</tr>
<tr>
<td>8</td>
<td>230</td>
<td>40</td>
</tr>
<tr>
<td>9</td>
<td>280</td>
<td>50</td>
</tr>
<tr>
<td>10</td>
<td>340</td>
<td>60</td>
</tr>
<tr>
<td>11</td>
<td>410</td>
<td>70</td>
</tr>
<tr>
<td>12</td>
<td>490</td>
<td>80</td>
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<tr>
<td>13</td>
<td>570</td>
<td>80</td>
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<tr>
<td>14</td>
<td>650</td>
<td>80</td>
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<tr>
<td>15</td>
<td>725</td>
<td>75</td>
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<tr>
<td>16</td>
<td>800</td>
<td>75</td>
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<tr>
<td>17</td>
<td>875</td>
<td>75</td>
</tr>
<tr>
<td>18</td>
<td>950</td>
<td>75</td>
</tr>
<tr>
<td>19</td>
<td>1025</td>
<td>75</td>
</tr>
<tr>
<td>20</td>
<td>1125</td>
<td>100</td>
</tr>
</tbody>
</table>

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Non-Electric Detonators
A number of initiation systems exist which offer distinct advantages over electrical blasting systems such as the avoidance of electrical hazards and possible misfires where detonation circuit leakage occurs.

In general non-electric delay detonators are equipped with either lightweight detonating cord (0.8 to 1.15 g per metre) or small diameter shock tube coated internally with a reactive dust in place of electric lead wires with fuse-heads.

Two products of this nature are ANOLINE and CORDLINE DELAY DETONATORS. They allow down the hole priming with similar degree of firing time as electric detonators. The indicating trunk line is of detonating cord.

Anoline and Cordline delays are available as shown in Table 5.5.

Cordline is essentially Anoline reinforced to withstand abrasion in deep holes.
Another form of non-electric delay detonators is that of the 'NONEL' GT Detonators. These are high strength delay detonators which are initiated by a shock wave passing through Nonel plastic tubing 3.00 m diameter and 1.5 mm bore crimped with detonators. The shock wave travels at 1900 metres per second. Initiation is by special gun, plain or electric detonator or detonating cord.

A feature of the firing of Nonel is that it does not rupture the tube hence it does not side initiate N.G. explosives or create a measureable air blast. Nonel GT Detonator Delays are shown in Table 5.6.

‘Nonel’ GT Detonators

<table>
<thead>
<tr>
<th>Delay Nos</th>
<th>No of Delays</th>
<th>Delay Firing Times Milliseconds</th>
<th>Delay Interval Milliseconds</th>
</tr>
</thead>
<tbody>
<tr>
<td>3-20</td>
<td>18</td>
<td>75 – 500</td>
<td>25</td>
</tr>
<tr>
<td>24 28 32 36 40 44</td>
<td>6</td>
<td>600 – 1100</td>
<td>100</td>
</tr>
<tr>
<td>50 56 62 68 74 80</td>
<td>6</td>
<td>1250 – 2000</td>
<td>150</td>
</tr>
</tbody>
</table>

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Table 5.6 ‘Nonel’ GT detonator delays

'ANZOMEX' BOOSTERS

<table>
<thead>
<tr>
<th>Booster Size</th>
<th>Diameter (mm)</th>
<th>Length (mm)</th>
<th>Weight (g)</th>
<th>Quantity per case</th>
<th>No and size of holes per booster</th>
</tr>
</thead>
<tbody>
<tr>
<td>'ANZOMEX' A</td>
<td>26</td>
<td>35</td>
<td>25</td>
<td>800</td>
<td>1 x 8.0 mm dia tunnel</td>
</tr>
<tr>
<td>'ANZOMEX' D</td>
<td>45</td>
<td>55</td>
<td>139</td>
<td>180</td>
<td>2 x 6.5 mm dia tunnel</td>
</tr>
<tr>
<td>'ANZOMEX' M</td>
<td>45</td>
<td>100</td>
<td>250</td>
<td>100</td>
<td>2 x 6.5 mm dia tunnel</td>
</tr>
<tr>
<td>'ANZOMEX' P</td>
<td>53</td>
<td>118</td>
<td>400</td>
<td>60</td>
<td>1 x 8.1 mm dia tunnel</td>
</tr>
<tr>
<td>'ANZOMEX' Q</td>
<td>54</td>
<td>115</td>
<td>416</td>
<td>60</td>
<td>1 x 8.1 mm dia blind (100 mm deep)</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2 x 6.5 mm dia tunnel</td>
</tr>
</tbody>
</table>

DETONATING CORDS

<table>
<thead>
<tr>
<th>Product</th>
<th>Colour</th>
<th>Nominal Core Load g/m PETN</th>
<th>Tensile Strength kg</th>
<th>Velocity of Detonation km/s</th>
<th>External Diameter mm</th>
<th>Standard Pack and Weight</th>
</tr>
</thead>
<tbody>
<tr>
<td>Blue 'CORDTEX'</td>
<td>Blue</td>
<td>8</td>
<td>105 – 110</td>
<td>7.0</td>
<td>3.5 – 4.0</td>
<td>4 x 375 m reels 27 kg</td>
</tr>
<tr>
<td>REDCORD</td>
<td>Red</td>
<td>10</td>
<td>110 – 115</td>
<td>7.1 – 7.5</td>
<td>4.5 – 4.8</td>
<td>4 x 250 m reels 30 kg</td>
</tr>
<tr>
<td>FLEXICORD</td>
<td>White</td>
<td>10</td>
<td>125 – 130</td>
<td>7.1 – 7.5</td>
<td>4.5</td>
<td>3 x 334 m reels 26 kg</td>
</tr>
<tr>
<td>TUFFCord</td>
<td>Yellow</td>
<td>10</td>
<td>175 – 190</td>
<td>7.1 – 7.5</td>
<td>3 x 334 m reels 26 kg</td>
<td></td>
</tr>
<tr>
<td>'GEOFLEX'</td>
<td>Green</td>
<td>20</td>
<td>230 – 240</td>
<td>6.5 – 7.3</td>
<td>6.4</td>
<td>2 x 200 m reels 24 kg</td>
</tr>
<tr>
<td>'GEOFLEX'</td>
<td>Pink</td>
<td>40</td>
<td>260 – 265</td>
<td>6.1 – 6.8</td>
<td>4 x 100 m reels 29 kg</td>
<td></td>
</tr>
<tr>
<td>'SHEARCORD'</td>
<td>Orange</td>
<td>75</td>
<td>120</td>
<td>6.4</td>
<td>4 x 50 m reels 26 kg</td>
<td></td>
</tr>
</tbody>
</table>

REDCORD is also available as 1 x 250 m reel per carton, 6 kg gross weight, and in 50 m packs, 18 packs per case of 600 m, gross weight 21 kg.

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Table 5.7 Primers and initiators
DETONATING CORD

Cordtex (U.S.A. Primacord) is a detonating fuse made with a core of PETN wrapped in textile yarns and the whole sheathed in plastic. Its main characteristics are safe handling, high water resistance and a V.O.D. of 7 000 metres per second. There are a variety of grades available, but the standard most commonly used grade is equal to a number 8 detonator.

Detonating cord is used for the simultaneous firing of widely distributed charges and mass initiation of very large charges. When used in bore holes with NG explosives, the simultaneous initiation of explosives increases efficiency. Primer cartridges or boosters are needed to initiate the AN-FO in AN-FO columns (See Table 5.7). Detonating cord largely replaces detonators, except for the initiating detonator. When sequential firing is required, the detonating cord is used in conjunction with detonating relays which consist of two identical elements mounted inside are silent plastic shell. The colour of the shell denotes the delay period which may vary from a nominal 15 to 45 milliseconds (See Fig. 5.4 Detonating Delay Connector).

INITIATION OF EXPLOSIVES

GENERAL

A detonator is essential in the initiations of high explosives and AN-C or slurry blasting agents, but the actual method of affecting detonation may vary to a considerable degree. The selection of the particular system is governed by the nature of the work being done, although some choice is possible. The following alternative initiation systems will be considered:

a. firing with safety fuse and plain detonator
b. electric firing:
   i. Instantaneous
   ii. Delay
c. firing with Cordtex
Firing With Conventional Safety Fuse
This involves the initiation of high explosive by a plain detonator which has been ignited by a safety fuse.

The safety fuse consists of a train or core of blasting powder in cotton, jute yarn and waterproof compounds. It is used for igniting a detonator and the strength and construction is such that it cannot ignite and adjacent fuse. This type of safety fuse was invented by Brickford in 1831 and used with the plain detonator ie. capped form used for initiating explosives. It is coloured buff (dull yellow) and is resistant to oil from ammonium nitrate fuel oil explosives. It is available in 12 metres, 250 and 1000 metre lengths. Regulations specify limits of burning rates within limits of 90 to 110 seconds per lineal metre. Extensive tests are conducted on manufacture involving rough handling, burning tests, immersion, storage and flame spit emission.

When using safety fuse it is most important that the shortest fuse used for firing gives ample time for the shot firer to withdraw to safety: a minimum of more than one metre is specified by regulations.

PREPARATION OF SAFETY FUSE
Capping is the process of crimping a plain detonator to a length of safety fuse to produce what is commonly called a "rod". Crimping on safety fuse of plain detonators may be accomplished by a bench mounted detonator crimper or approved hand crimping pliers. (See Fig. 5.5). The first is used where large numbers of capped fuses (rods) are prepared. Hand crimpers are adequate where infrequent crimping is required. Specific safety precautions must be followed in handling and capping of fuses with plain detonators. (See Fig. 5.6).

Figure 5.5 Hand crimping pliers for plain detonators and bean hole connectors for PIC
METHODS OF IGNITING SAFETY FUSE

By Match
Only a single fuse may be lit by match in any one blast. The procedure adopted is to expose the central core of black powder of the safety fuse, place a match head against the core, then strike the matchbox striking surface across the match head. This will ignite the central core of the safety fuse.

Electric ignitors may also be used to ignite the safety fuse.

Fuse Igniters
These are available as a two-minute burning variety giving a fierce flame; regulations restrict their use to hand lighting of no more than four safety fuses.

Multiple Safety-Fuse Igniters
These are hollow cylindrical cartridge shells with a flammable base compound which allows multiples of eight live fuses to be ignited by one master. The safety fuses are packed tightly in the cartridge (blank fuses are used if the number is less than eight) and the master fuse is lit. When the master fuse burns down to the bottom of the cartridge it sets off a flash ignition of the base material which lights the safety fuses. The ignition of detonators in multiples of eight contributes to the safety and efficiency of blasting. See Fig. 5.7.

Plastic Igniter Cord
This is a method of igniting a number of safety fuses in a prescribed sequence especially when charges are too widely separated for the use of multiple fuse-igniter cartridges. P.I.C. is a cord-like fuse which burns uniformly and progressively with an intense flame.
Slotted connectors jointed to the P.I.C. trunk or branch lines are crimped to safety fuses for individual hole initiation. Joints in P.I.C. may also be made by knotting. P.I.C. is produced in two burning rates:

(a) P.I.C. Fast rate of 1/3 metre per second (300 mm per second) is colour-coded brown. This is useful for connecting secondary blasting in quarries to give both safety and economy. To avoid discontinuities because of fly rock cut-off, P.I.C. should not be longer than individual length of fuse; i.e. all fuses should be alight before the first charge detonates.

(b) P.I.C. slow rate, colour-coded green, has an approximate rate of 1/30 metre per second (30 m per second) and is used for firing underground.
ASSEMBLY OF A PRIMER CARTRIDGE

A primer is a cartridge of explosive which has been fitted with a detonating device used to detonate the remainder of a charge which does not contain any detonating device.

Procedure to prepare a safety fuse primer:

(i) Take a whole cartridge of explosive.

(ii) Skewer a 50 mm opening into an end of the cartridge with a wooden or brass skewer or make a hole diagonally in the side of the cartridge.

(iii) Insert the detonator end of the capped fuse into the hole until the detonator is completely buried.

An alternative to the above approaches is to open the wrapping at one end of the cartridge, skewer a hole and implant the detonator, restore the wrapper and tie it around the fuse with twine. The advantage of this approach is that the fuse is not likely to be displaced.

Essential requirements of the primer are:

• The detonator should not become dislodged from the cartridge.
• The detonator should be in the safest position in the cartridge.
• The primer should be capable of easy and safe loading into the required position in the hole without damage to the fuse.
• The detonator should be inserted deeply into the cartridge and lie along the long axis of the cartridge.
• The closed end of the detonator should point towards the bulk of the cartridge.

(See Figs 5.9 and 5.10)

Figure 5.10 Preparing a primer using a capped fuse
PREPARING A PRIMER USING DETONATING CORD
Detonating cord must be initiated by detonator and when initiated has the detonating properties of a detonator at all points along its length.

When detonating cord is to be used as a primer the cartridge should be prepared so that the cord cannot readily be pulled away from the charge. The following methods are suggested to ensure that there is sufficient contact between cord and cartridge.

(a) Diagonally skewer a hole through the cartridge and thread the detonating cord through it. Make another hole axially at the opposite end of the cartridge and anchor the cord end in this hole. Draw the cord tight so the primer is ready for priming.

(b) Make an axial hole through the centre of the cartridge and thread the detonating cord through and secure the end by knotting.

(c) Skewer an axial hole off-centre and thread through the cord, anchor the cord in the end of the cartridge. Draw the cord tight ready for primer placement. (See Fig. 5.11).

When several detonating cord primers (downlines) are to be fired simultaneously, it is necessary to joint the downlines to the trunk line so that they can be initiated by a plain or electric detonator fixed to the trunk line.

NOTE: Detonating cord explosion propagation is directionally sensitive. Junctions must be either gradual curves or tied knots at right angles, otherwise the downline may fail to be initiated from the mainline.

Connections of branch (down lines) to the can be made by the following methods:

(a) wiring or taping (restricted to initiation in one direction only);

(b) close-hitch joint;

(c) three-lap joint;

(d) clip joint; (See Fig. 5.12)
Figure 5.11 Three methods of preparing detonating cord primers
CHAPTER 5 EXPLOSIVES

5.22

Figure 5.12 Detonating cord connections

ELECTRIC SHOT FIRING

Electric blasting is in general use in Australia with instantaneous or delay detonators employing exploders (blasting machines) or mains power as the electrical impulse source: Cordtex may also be used for simultaneous firing of holes.

Care in all electrical firing is required so that the circuit resistance (because of loose and dirty connections) or current leakage can be minimized.

EXPLODERS

Exploders provide a simple method of electric firing but the number of holes or detonators is limited by the capacity of the machine and the length of the firing cable. Mains power firing using a voltage of 110-440 allows a larger number of holes to be fired but the system is more complicated. All exploders have removable firing keys which, together with the machines and circuits, ensure safety and efficiency. Exploders produce a low amperage high voltage output. Detonators are commonly connected in series.

Examples of exploders in use include:

Little Demon
This is a small hand-type generator capable of firing from 1 to 10 detonators. It is held in the hand and operated by giving the handle a quick twist.
ME-6 Exploder
This is essentially a battery capable of firing 6 electric detonators: a test for continuity is incorporated in the design. It is used essentially in coal mining. (See Fig. 5.13).

Rack Bar Type
These exploders, which are rated for 30 detonators, are essentially a generator (dynamo) where the armature is rotated by a rack-bar and pinion. At the end of the downward stroke an internal short circuit device opens and current is passed through the connected firing circuit.
5.24

Condenser Type
This type of exploder can be battery energised (e.g. Du Pont CD 32 Blasting Machine) or
dynamo energised (e.g. I.C.I. 'Beethoven' Dynamo Condenser Exploder) to charge
condensers which, when discharged, safely and reliably fire the rated capacity number of
detonators in a series circuit. The voltage build-up is high, of the order of 1500 volts; the
number of shots fired is usually of the order of 100 depending upon the output of the
machine. (See Fig. 5.14).

ELECTRIC PRIMER PREPARATION
The procedure for preparing electric primers is similar to that of capped fuses; half-
hitching the wires around the cartridge secures detonator and cartridge. The lead wires
should be left short-circuited and kept clear of all electrical circuits until loading and
charging is complete.
SHOT FIRING WITH EXPLODERS
All explosives should be equipped with removable firing keys which are to be kept in the possession of the shot firer. For single shots, the leading wires of the detonators are bared and connected to the end of the shot firing cable. On retreat, the other ends of the shot firing cable are connected to the terminals of the exploder, which is set up at a safe distance. When all personnel have been cleared from the area, the shot firer inserts the key and operates the exploder. Before firing an electric detonator circuit it can be proved by testing and the shot firing cable may be tested for continuity and insulation failure.

**Continuity Test**
The two wires of one end are twisted together. A circuit tester is connected to the wires at the other end which will indicate the passage of current and prove continuity. The electrical resistance is also determined.

**Insulation Test**
For this test the shot firing cable ends are left unconnected. The blasting circuit test meter, which is connected to one end should show no current flowing and that the resistance is infinite, i.e. an open circuit. Next the detonating circuit is connected to the shot firing cable and the complete circuit is tested from the firing position.

**Circuit Test Meter**
These meters are designed, and must be approved for detonation circuit testing so that under normal conditions of operation not more than 50 milliamps (mA) can be delivered to the circuit under test. However, when testing of faults must be conducted at the charged face, the maximum circuit-tester current approved is not more than 10 milliamps. A current flow of 300 mA may initiate the fuse head of an electric detonator. (See Figs 5.15 and 5.16).

The average resistance of electric detonators depends upon lead wire lengths: these may be seen in Table 5.8 which also shows the resistance of shot firing cables. The minimum current in a series circuit should be 1.5 amperes, while for parallel firing the minimum current of 0.75 amperes per detonator should be allowed.

**Power Mains Blasting**
When power mains are used, a much larger number of shots may be fired. For small rounds, simple series or parallel circuits are used, but where a large number of detonators are involved in one firing, a parallel-in-series (Fig. 5.20) or series-in-parallel circuit (Fig. 5.19) is used. Special precautions are required to prevent stray currents or accidental premature closing of the blasting circuit by unauthorized persons. Switch gear is usually locked in cubicles or cabinets. There are generally three components of mains firing:

- fuse box - which is an earthed metal box containing fuses and the main switch; it cannot be closed until the switch is in the off position;
- firing box - which contains a quick break switch, made so that the box cannot be closed unless the switch is in the off position. On firing the blasting switch is closed only momentarily; prolonged closure may result in flashover of the detonator fuse head to the tube, causing misfires.
Terminal current delivery 5 milliamps

Resistance Range:  
Low $\times$ 1 (0 – 30 ohms)  
High $\times$ 10 (0 – 300 ohms)

**Operation**
1. Connect blasting circuit to terminals.
2. Press push button switch and hold down. One light emitting diode will light (red).
3. Turn knob as indicated by arrow over light. If turned too far other diode will light.
4. The point where both lights are out - circuit resistance is read using scale multiplier as indicated by the range button.
5. If circuit resistance is normal it is then ready for firing.

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*Figure 5.15 Nobel Detameter (busting circuit tester)*

Terminal current delivery 12 milliamps

Resistance Range (Scales)  
Low 0 - 50 ohms  
High 10 - 1,000 ohms

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*Figure 5.16 Nobel blasting circuit tester (Blastometer) Mk 111*
5.27

Table 5.8 Resistance of electric detonators and connecting wires

- short circuit box - which is interposed between firing lines and the firing box. With the box closed the switch is in the shorted position so that premature blast due to stray currents in lines is prevented.

Once the round is connected and coupled to the firing lines the short circuit box is opened and continuity and resistance tests are made, the firing box is opened and connected to the fuse box. The shot may then be fired by momentarily closing the firing switch. The reverse procedure is then adopted to render the system safe for the next usage. The system uses a 440 voltage and extreme care is required because blasting lines are susceptible to stray electrical currents.

ELECTRICAL BLASTING CIRCUITS

(A) SERIES CIRCUITS

Simple series circuits are used almost exclusively for simple exploder machines. They are made by connecting one wire from each electric detonator to one wire of the next. Continuity in series firing can be accurately checked. A current of 1.5 amperes will ensure that each detonator functions adequately.

EXAMPLE

30 electric detonators with 4-metre lead wires, each with 2.0Ω (ohms) resistance, are connected to 200 metres of connecting wire (resistance 60 Ω/1000m) and 200 metres of firing cable (resistance 6 Ω/1000 twin wire cable).

Resistance of 30 electric detonators \( (30 \times 2) = 60 \, \Omega \)

Resistance of connecting wire \( = 12 \, \Omega \)

Resistance of firing cable \( = 12 \, \Omega \)

Total resistance: \( 84 \, \Omega \)

Voltage necessary \( = 1.5 \times 84 = 126 \) volts
Ohm's Law relationship

\[ V = i \times R \]

Where

- \( V \) = Voltage (volts)
- \( i \) = current (amperes)
- \( R \) = resistance (ohms)

The exploder suitable for this would be a condenser type with a rated capacity of 100 detonators. Series call for low current at high voltage. They are not commonly used with mains power current.

**B) PARALLEL CIRCUITS**

These are made by connecting the detonating wires to two common points (See Fig. 5.18). Each detonator provides an alternate path for the current. The complete circuit cannot be instantly tested for continuity: each detonator must be individually checked. In parallel firing, a minimum current of 0.75 amperes is usually allowed for each detonator. Total resistance is low because the total resistance equals the resistance of one detonator divided by the number of detonators.

**EXAMPLE**

30 detonators (resistance 2 \( \Omega \)) are connected in parallel. The firing cable is heavy duty (200 m, total resistance 3.6 \( \Omega \))

\[
\text{Resistance: } \frac{2.0}{30} = 0.06 \text{ ohm} \\
\text{Resistance of connecting wire} = 12.0 \Omega \\
\text{Resistance of firing cable} = \frac{3.6 \Omega}{15.66} \\
\]

Current required 0.75 \( \times \) 30 = 22.5 amps

Voltage required 22.5 \( \times \) 15.7 = 353 volts

**C) SERIES-IN-PARALLEL CIRCUIT**

This is arranged by connecting a number of series circuits in parallel, each circuit having approximately the same resistance (balanced). The resistance is given by the resistance of the series divided by the number of series in parallel. A current of 1.5 amperes should be provided for each series. (See Fig. 5.19).

**EXAMPLE**

30 electric detonators are connected in five series of six using 50 metres of connecting wire (3 \( \Omega \)) and 200 metres (3.6 \( \Omega \)) of firing cable. (Detonators 4-metre lead wires 2 \( \Omega \) resistance).

Current required 1.5 \( \times \) 5 = 7.5 amp

Resistance (2.0 \( \times \) 6) per series = 12 \( \Omega \)

Total resistance = \( \frac{12.0}{2} + 3 + 3.6 \) = 12.6 \( \Omega \)

Voltage required = 7.5 \( \times \) 12.6 = 94.5 volts
(D) PARALLEL-IN-SERIES
In this type of circuit, detonators are connected in parallel in a number of groups each in series, (See Fig. 5.20). The current to be provided depends upon the primary connections, hence 0.75 amps is the minimum requirement. Each group should have a similar resistance if misfires are to be avoided.

Additional parallel groups would add to the resistance and increase the voltage required. The more detonators in a group (parallel) the lower the resistance of the circuit.

**EXAMPLE**
30 detonators are connected in 5 parallel groups of 6 detonators (each 2 Ω) using 50 metres of connecting wire (3 Ω) and 200 metres of firing cable (3.6 Ω).

Resistance of each parallel group \( \frac{2}{6} \) = 0.3 Ω

Total resistance = 5 (0.30) + 3 + 3.6

series circuit

Current required: (0.75 x 6) = 4.5 amp

Voltage required: (8.1 x 4.5) = 36.5 volts

This is the most advantageous circuit for electrically firing a multiplicity of holes where they can be arranged in even groups.
Sequential Firing
The provision for charges to explode in sequence rather than simultaneously is important. This is achieved by:

1. Conventional Safety Fuse Firing
   - Firing one or two holes at a time.
   - Cutting fuses to different lengths or lighting in desired order.
   - Linking standard lengths with PIC in desired order.

2. Electric Detonators
   - Using half-second detonators.
   - Using short/millisecond delay detonators.

3. With detonating fuse using detonating relay connectors.
## Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>AN</td>
<td>Ammonium nitrate.</td>
</tr>
<tr>
<td>AN-C MIXTURE</td>
<td>Mixture of AN and a carbonaceous additive or sensitizer.</td>
</tr>
<tr>
<td>ANFO</td>
<td>Blasting agent consisting of AN and fuel oil.</td>
</tr>
<tr>
<td>BLASTING AGENT</td>
<td>Material or mixture consisting of fuel (combustible) and oxidizer, used as an explosive. Ingredients are not classified as explosives.</td>
</tr>
<tr>
<td>BLASTING POWDER</td>
<td>Low explosive consisting of potassium or sodium nitrate, charcoal and sulphur.</td>
</tr>
<tr>
<td>BOOSTER</td>
<td>An explosive of special character, used in small quantities to improve the performance of another explosive.</td>
</tr>
<tr>
<td>BRISANCE</td>
<td>Ability of an explosive to break (or shatter) rock by shock or impact, as distinct from gas pressure.</td>
</tr>
<tr>
<td>CAP</td>
<td>A detonator.</td>
</tr>
<tr>
<td>CAPPED FUSE</td>
<td>Safety fuse to which a plain detonator is crimped – &quot;a rod&quot;.</td>
</tr>
<tr>
<td>CARTRIDGE</td>
<td>Preformed unit of high explosive wrapped to a specific diameter and length. (plug or stick)</td>
</tr>
<tr>
<td>COOLING SALT</td>
<td>Either sodium chloride or sodium carbonate incorporated in HE to reduce heat of explosion as in permitted explosives.</td>
</tr>
<tr>
<td>DETONATING WAVE</td>
<td>Shock wave set up when a detonator is ignited.</td>
</tr>
<tr>
<td>DETONATOR</td>
<td>A cap or capsule of sensitive explosive material used to initiate a charge of high explosive.</td>
</tr>
<tr>
<td>FRACTEUR</td>
<td>A term used by miner or powder monkey for any high explosive.</td>
</tr>
<tr>
<td>GELATINOUS</td>
<td>(Plastic)</td>
</tr>
<tr>
<td></td>
<td>1. The property of types of high explosive to be pressed into different shapes, i.e. blasting gelatine, gelatine dynamite, gelignite.</td>
</tr>
<tr>
<td></td>
<td>2. Gelatinous explosive that is readily and efficiently placed or tamped into a hole.</td>
</tr>
<tr>
<td>GUNPOWDER</td>
<td>See Blasting Powder.</td>
</tr>
<tr>
<td>HE</td>
<td>High explosive.</td>
</tr>
<tr>
<td>IGNITE</td>
<td>To light or set alight. e.g. Safety fuse ignited by flame, plain detonator ignited by safety fuse, electric detonator ignited by current through bridge head.</td>
</tr>
<tr>
<td>Term</td>
<td>Definition</td>
</tr>
<tr>
<td>------------</td>
<td>-----------------------------------------------------------------------------</td>
</tr>
<tr>
<td>INITIATE</td>
<td>Act of detonating high explosive by means of a detonator or by detonating cord – (Cordtex).</td>
</tr>
<tr>
<td>LEAD WIRES</td>
<td>Insulated wires connected to the electrodes of an electric detonator.</td>
</tr>
<tr>
<td>NG</td>
<td>Nitroglycerine.</td>
</tr>
<tr>
<td>PERMITTED</td>
<td>Term covering an explosive that has passed British Government approval as suitable for use in gaseous coal mines.</td>
</tr>
<tr>
<td>POLAR</td>
<td>Term for low-freezing explosives.</td>
</tr>
<tr>
<td>PRI LLS</td>
<td>Cellular sub-globular AN particles formed by spraying AN solution against a stream of air.</td>
</tr>
<tr>
<td>PRIMER</td>
<td>A cartridge of HE incorporating a rod or electric detonator.</td>
</tr>
<tr>
<td>SEQUENTIAL FIRING</td>
<td>System of firing where holes of least resistance are detonated progressively.</td>
</tr>
<tr>
<td>STRENGTH</td>
<td>Explosive strength of unit weight or volume of HE compared with Blasting Gelatine in a ballistic mortar. Designation may be by 2 NG which is not a true measure of strength.</td>
</tr>
<tr>
<td>V.O.D.</td>
<td>Velocity of detonation: a measure of the rate of detonation wave travel through an explosive.</td>
</tr>
</tbody>
</table>
CHAPTER 6 – Blasting and Blasting Practices

GENERAL
Explosives are used in the mining industry for fracturing rocks ore and coal, either in the actual recovery of economic minerals or for development work in preparation for production, where no mechanical or hydraulic methods may be applied to advantage.

APPLICATION
In order to fracture rock (or to break ground) the explosive charge must be placed within the rock at some distance behind the face. For this purpose openings must be made into the rock either by drilling holes or by excavating chambers for large blasts.

The rock mass must have one or more free faces, i.e. it must be exposed or open on one or more planes. Drill holes are placed at right angles to the plane(s) of the face and the rock is blasted in the direction of the free face. This is necessary, first, because broken rock occupies a greater space than solid rock - See Table 6.1 - and must have room to expand (swell); second, it allows the rock to be broken in tension rather than in shear where there is no free face or tight conditions. (See Fig. 6.1).

Where a free face does not exist as in tunnels or mine development headings it must be provided by one of the following methods:

• by explosives in cut holes (centre, wedge, drag cuts);
• by explosives in cut holes in conjunction with free (uncharged) holes (as in burn cuts) by drilling a large diameter hole or slot;
• by a mechanical cut kerf (under cut) as for a coal face.
The figures represented are only averages and materials vary in weight, grain size, moisture content and degree of compaction from site to site. For specific characteristics, tests need to be made.

Note: Swell factor of material is an allowance for voids space between chunks of solid after blasting or excavation.

\[
\text{swell factor} = \frac{100}{100 + \% \text{ of swell}}
\]

<table>
<thead>
<tr>
<th>Material</th>
<th>lb/lcy (kg/lm³)</th>
<th>lb/bcy (kg/bm³)</th>
<th>Swell Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bauxite</td>
<td>2400 (1424)</td>
<td>3200 (1899)</td>
<td>.75</td>
</tr>
<tr>
<td>Clay dry</td>
<td>2500 (1483)</td>
<td>3100 (1839)</td>
<td>.81</td>
</tr>
<tr>
<td>wet</td>
<td>2800 (1661)</td>
<td>3500 (2076)</td>
<td>.80</td>
</tr>
<tr>
<td>Coal Bituminous</td>
<td>1600 (949)</td>
<td>2150 (1276)</td>
<td>.74</td>
</tr>
<tr>
<td>Copper Ore</td>
<td>2800 (1661)</td>
<td>3800 (1229)</td>
<td>.74</td>
</tr>
<tr>
<td>Granite</td>
<td>2800 (1661)</td>
<td>4600 (2729)</td>
<td>.61</td>
</tr>
<tr>
<td>Iron ore</td>
<td>3915 (2326)</td>
<td>6500 (3862)</td>
<td>.60</td>
</tr>
<tr>
<td>Hematite</td>
<td>5155 (3063)</td>
<td>8700 (5169)</td>
<td>.59</td>
</tr>
<tr>
<td>Limestone blasted</td>
<td>2600 (1543)</td>
<td>4400 (2611)</td>
<td>.59</td>
</tr>
<tr>
<td>Sand dry</td>
<td>2400 (1424)</td>
<td>2700 (1602)</td>
<td>.89</td>
</tr>
<tr>
<td>wet</td>
<td>3100 (1839)</td>
<td>3500 (2077)</td>
<td>.89</td>
</tr>
<tr>
<td>Trap rock</td>
<td>3350 (1990)</td>
<td>5000 (2970)</td>
<td>.67</td>
</tr>
</tbody>
</table>

Table 6.1 Material mass and swell factors

![Material swell (pictorial definition)](image)

2000 kg
Bank measure
1 cu metre

2000 kg
Loose measure
1.5 cu metres

Figure 6.1 Material swell (pictorial definition)
Aspects which have an effect on blasting operations are:

| ROCK: Presence of and extent of free faces | Strength of rock blasted |
| Structure of rock (jointed, massive or stratified) |
| EXPLOSIVE: Type, strength and nature | Loading density and confinement |
| DETONATION: Instantaneous or sequential |
| BLAST HOLE: Size, type and depth of hole | Amount of hole loaded |
| Burden |
| OPERATIONAL: Safety | Statutory regulations |
| Climatic conditions |
| Shape of excavation |
| Quantity of rock |
| Permissible concussion |
| Degree of fragmentation |
| Loading procedure and equipment |

Rock fragmentation factors, however, may be considered under three groups of parameters:

(a) explosive parameters, which include explosive density, velocity of detonation, gas volume and available energy.

(b) charge loading parameters, including charge diameter and length, stemming, coupling and type and point of initiation.

(c) rock parameters related to rock density, strength (compressive and tensile), texture and propagation velocity.

**CHOICE AND QUANTITY OF EXPLOSIVES**

In determining the type, strength, quantity and cartridge diameter of explosives to be used for a given purpose the most significant considerations are:

- Applicability to the particular job.
- Safety of life and property.
- Efficiency, usually stated in terms of explosives consumption, kg per tonne of material broken.
- Economy, taking the cost of drilling, charging, explosives and accessories expressed in terms of costs per tonne of ore or rock broken.

**FRAGMENTATION**

Fragmentation is a term used to describe the site distribution of rock boulders and particles produced when a solid mass of rock is broken by explosives.

The significance of fragmentation cannot be underestimated because, to a large degree it is the measure of the success of a blast: it influences the operational and maintenance costs of subsequent operations and equipment including such unit operations as excavating or
loading, haulage and crushing or size of reduction plant. Therefore both drilling and
blasting are clearly related to cost optimization of subsequent operations.

Poor fragmentation results in oversize or large boulders involving secondary breaking
costs to reduce them to a size which can be handled economically, safely and efficiently by
loading and haulage equipment, which in underground operations often includes hoisting
in inclined or vertical shafts.

Excessive production of fines or undersize rocks is also undesirable because it indicates
possible wastage of explosives; economic size reduction could be achieved by the correct
utilization of crushing installations. However, under certain circumstances, fragmentation
may be improved by the adoption of one or all of the following measures (these apply to
bench or quarry blasting):

1. Reduce hole depth; i.e. use shallow holes with improved distribution of explosive.
2. Reduce spacing between adjacent holes in a row.
3. Reduce burden distance.
4. Use an explosive with greater gas generation (heave) and less brisance.
5. Use short delay detonators.

BLASTING THEORIES
There are a number of theories regarding how explosives achieve their effectiveness in
fragmenting rock. Many of them involve empirical formulas which show certain blast
factors to be variable while others are fixed or static.

A theory of particular note is the Reflection Theory. The detonation of the explosive charge,
whether N-G or AN-C, creates a high gas pressure in the blast hole, which in turn
generates a compressive strain pulse in the surrounding rock. Crushing of the rock will
only take place in the vicinity of the hole because the strain pulse decays rapidly as it
travels outward in all directions from the hole. The bulk of the rock mass is broken up by a
tensile strain pulse, which is the reflection of the original compressive strain pulse from the
free surfaces. The rock is progressively broken up away from the free faces. Hence the
importance of providing free surfaces for high blasting efficiency either by ‘cut’ holes or
slotting (e.g. coal face) as well as allowing for broken rock to expand or swell on
fragmentation.
Figure 6.2 Open pit bench terms

1. Broken ore stocks.
2. Production face (working bench).
a. Dip of ore body.

Figure 6.3 Open pit bench terms

a. Bottom bench floor.
b. Toe of high wall.
c. High wall (bench face).
d. Bench crest.
e. Top of bench.
α. Angle of slope.
h. Bench height.
QUARRY BLASTING PRACTICE

Where possible, rock should be broken to the required size in the primary blast. The depth of the holes, the charge required, the burden distance, and hole spacing are closely related and careful control is essential to ensure blasting efficiency.

See: Fig. 6.2. Open Pit Multi-Bench Layout Cross Section which illustrates aspects of production faces.

Fig. 6.3. Open Pit Bench diagram that introduces fundamental terms used in quarrying.

Fig. 6.4. Diagrammatic Section Through a Quarry Bench illustrating terminology of quarry blasting.

The maximum amount of work in breaking out the face has to be done at the bottom of the bore hole where the charge must supply sufficient energy to shear the rock at the floor level, without leaving a toe, in addition to breaking it along the line of holes (to give an even, safe rock face) and displacing the mass of rock. Therefore inclined holes are used in preference to vertical holes, and sub-drilling is done below projected bench floor to ensure no remnant toe.
BURDEN DISTANCE

The burden distance is the distance between the bottom of the hole and the free face, and corresponds with the line of least resistance. Burden distance should be less than hole depth to prevent surface cratering. (See Fig. 6.5)

A blasting design formula for single holes gives burden in the following imperial equation. (Andersen's equation)

\[ B = C \sqrt{\frac{D}{L}} \]

where

- \( B \) = burden distance
- \( D \) = hole diameter, m
- \( L \) = length of hole, m
- \( C \) = constant determined empirically, 3.5. (This applies to NG based explosives and AN/FO) (See Table 6.2)

The formula derived by Andersen allows burden distance to be determined from readily available physical dimensions. The formula is modified for multi-shot firing where spacing affects the burden adopted.

Example of burden distance calculation for quarry blasting:

**Parameters**

Hole diameter of 100 mm, hole depth 10 metres.

Determine the single hole burden distance where \( C = 3.5 \)

\[ B = C \sqrt{\frac{D}{L}} \]

Hence \[ B = 3.5 \times \sqrt{\frac{100}{1000}} \times 10 \]

\[ B = 3.5 \times 1 \]

Burden distance is 3.5 metres

<table>
<thead>
<tr>
<th>Diameter of hole (mm)</th>
<th>LENGTH OF HOLE (m)</th>
<th>BURDEN (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>3</td>
<td>5</td>
</tr>
<tr>
<td>25</td>
<td>0.9</td>
<td>1.2</td>
</tr>
<tr>
<td>50</td>
<td>1.3</td>
<td>1.75</td>
</tr>
<tr>
<td>75</td>
<td>1.7</td>
<td>2.1</td>
</tr>
<tr>
<td>100</td>
<td>1.9</td>
<td>2.5</td>
</tr>
<tr>
<td>125</td>
<td>3.3</td>
<td>3.9</td>
</tr>
<tr>
<td>150</td>
<td>3.6</td>
<td>4.6</td>
</tr>
<tr>
<td>175</td>
<td>3.9</td>
<td>4.9</td>
</tr>
<tr>
<td>200</td>
<td>4.1</td>
<td>5.25</td>
</tr>
<tr>
<td>225</td>
<td>4.4</td>
<td>5.5</td>
</tr>
<tr>
<td>250</td>
<td>4.6</td>
<td>5.8</td>
</tr>
<tr>
<td>275</td>
<td>4.85</td>
<td>6.0</td>
</tr>
<tr>
<td>300</td>
<td>5.0</td>
<td>6.6</td>
</tr>
</tbody>
</table>

Table 6.2 Burden distances determined by using Andersen Equation (in metres, using \( c = 3.5 \)
### Table 6.3 Probable blasthole burden ranges for quarry blasting

<table>
<thead>
<tr>
<th>Blasthole Diameter (mm)</th>
<th>Range of Burden Distance (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>50 (2&quot;)</td>
<td>1.1 - 2.1 (3.5 - 7.1)</td>
</tr>
<tr>
<td>65 (2.5)</td>
<td>1.4 - 2.4 (4.5 - 8)</td>
</tr>
<tr>
<td>75 (3)</td>
<td>1.5 - 2.6 (5 - 8.5)</td>
</tr>
<tr>
<td>90 (3.5)</td>
<td>1.9 - 3.3 (6 - 11)</td>
</tr>
<tr>
<td>100 (4)</td>
<td>2.3 - 3.9 (7.5 - 13)</td>
</tr>
<tr>
<td>115 (4.5)</td>
<td>2.5 - 4.3 (8 - 14)</td>
</tr>
<tr>
<td>125 (5)</td>
<td>2.8 - 4.9 (9 - 16)</td>
</tr>
<tr>
<td>150 (6)</td>
<td>3.3 - 5.6 (11 - 18.5)</td>
</tr>
<tr>
<td>200 (8)</td>
<td>4.1 - 6.7 (13.5 - 22)</td>
</tr>
<tr>
<td>230 (9)</td>
<td>4.5 - 7.2 (15 - 24)</td>
</tr>
<tr>
<td>250 (9.8)</td>
<td>4.9 - 7.6 (16 - 25)</td>
</tr>
<tr>
<td>270 (10.6)</td>
<td>5.2 - 8.3 (17 - 27)</td>
</tr>
<tr>
<td>310 (12.25)</td>
<td>5.5 - 9.9 (18 - 32)</td>
</tr>
<tr>
<td>380 (15)</td>
<td>6.7 - 12.3 (22 - 40)</td>
</tr>
</tbody>
</table>

Note: Line of break shown for concentrated charge at hole bottom which results in high blasting efficiency, safe operation and cost reduction. In practice hole inclination is up to 60°.

Figure 6.5 Ideal inclination of quarry face and drill hole (inclination 45°) to prevent residual toe occurring
STEMMING
The uncharged section of the blasthole between the explosive charge and the borehole collar is usually back-filled with stemming material. The depth of stemming should not be less than the burden distance so as to minimise flyrock from cratering off the upper portion of the blast hole.

METHOD OF FIRING
Simultaneous firing charges in blastholes so they mutually assist one another is fundamental to surface mining operations. However restriction on the size of the simultaneous blast may apply because the ground vibration produced often results in damage to adjacent structures and consequent complaints. The principle of mutual assistance is retained by using short delay (milliseconds) firing methods which reduce or minimise blast vibration. Consequently, because vibration is related to the weight of explosive detonated in any single short delay period and not to the total weight of the blast, the overall site of the blast may be increased. The ground vibration is measured by instruments such as vibrographs or seismographs.

Advantages of delay blasting such as short delay electric detonators or detonating cord relay are:

1. Reduction in ground vibration.
2. Reduction in air concussion.
3. Reduction in overbreak.
4. Improved fragmentation.
5. Better control of flyrock.

CHARGE MASS CALCULATION
For successful blasting, the bore hole must be drilled to give an achievable burden distance. It is also necessary to obtain a loading density in the bottom portion of the blasthole.

The quantity of explosive, required for each blasthole may be determined by the following equation. Note that this applies to a specified explosive (powder) factor (kg/tonne).

Equation:
\[ Q = B \times S \times D \times d \times E \ (kg) \]
where
- \( Q \) = mass or quantity of explosive
- \( B \) = Burden distance (m)
- \( S \) = spacing (m)
- \( d \) = density factor of the rock (tonnes/m\(^3\)) (See Table 6.5)
- \( D \) = hole depth (face height)
- \( E \) = powder factor (kg/tonne)

Example of quantity of explosive required for a single blasthole.

Parameters:
- Hole diameter 100 mm; hole depth 10 metres;
- Column of explosive (section loaded) 6.5 m;
- Burden distance 3.5 m, powder factor 0.16;
- Hole spacing (1.5 x burden distance) 5.25 m;
- Rock density (granite) 2.7 tonnes/cubic metre

\[ Q = B \times S \times D \times d \times E \ (kg) \]
\[ = 3.5 \times 5.25 \times 10 \times 2.7 \times 0.16 \]
\[ = 79.4 \text{ kg of explosive} \]
6.10 A rough check is made on this result by using Loading Density Table 6.4. Explosive density is 1.45 gm/cm$^3$.

Explosive column mass per metre is approximately $\frac{12.0}{4.79} = 6.6$ metres

This agrees, roughly, with suggested loading of 6.5 metres.
Table 6.5 Rock densities per cubic metre

<table>
<thead>
<tr>
<th>Material</th>
<th>Specific Gravity</th>
<th>Solid kg/m³</th>
<th>Broken kg/m³</th>
</tr>
</thead>
<tbody>
<tr>
<td>Basalt</td>
<td>2.8-3.2</td>
<td>3040</td>
<td>2000</td>
</tr>
<tr>
<td>Coal — Anthracite</td>
<td>1.3-1.8</td>
<td>1600</td>
<td>1040</td>
</tr>
<tr>
<td>Coal — Bituminous</td>
<td>1.2-1.5</td>
<td>1360</td>
<td>880</td>
</tr>
<tr>
<td>Diabase</td>
<td>2.6-3.0</td>
<td>2800</td>
<td>1840</td>
</tr>
<tr>
<td>Diorite</td>
<td>2.8-3.0</td>
<td>2960</td>
<td>1920</td>
</tr>
<tr>
<td>Dolomite</td>
<td>2.8-2.9</td>
<td>2880</td>
<td>1840</td>
</tr>
<tr>
<td>Gneiss</td>
<td>2.6-2.9</td>
<td>2880</td>
<td>1840</td>
</tr>
<tr>
<td>Granite</td>
<td>2.6-2.9</td>
<td>2720</td>
<td>1760</td>
</tr>
<tr>
<td>Gypsum</td>
<td>2.3-3.3</td>
<td>2880</td>
<td>1840</td>
</tr>
<tr>
<td>Hematite</td>
<td>4.5-5.3</td>
<td>4880</td>
<td>3200</td>
</tr>
<tr>
<td>Limestone</td>
<td>2.4-2.9</td>
<td>2640</td>
<td>1680</td>
</tr>
<tr>
<td>Limonite</td>
<td>3.6-4.0</td>
<td>3760</td>
<td>2480</td>
</tr>
<tr>
<td>Magnesite</td>
<td>3.0-3.2</td>
<td>3200</td>
<td>2000</td>
</tr>
<tr>
<td>Magnetite</td>
<td>4.9-5.2</td>
<td>5040</td>
<td>3280</td>
</tr>
<tr>
<td>Marble</td>
<td>2.1-2.9</td>
<td>2480</td>
<td>1600</td>
</tr>
<tr>
<td>Mica-Schist</td>
<td>2.5-2.9</td>
<td>2720</td>
<td>1760</td>
</tr>
<tr>
<td>Porphyry</td>
<td>2.5-2.6</td>
<td>2560</td>
<td>1680</td>
</tr>
<tr>
<td>Quartzite</td>
<td>2.0-2.8</td>
<td>2560</td>
<td>1680</td>
</tr>
<tr>
<td>Salt-Rock</td>
<td>2.1-2.6</td>
<td>2320</td>
<td>1520</td>
</tr>
<tr>
<td>Sandstone</td>
<td>2.0-2.6</td>
<td>2400</td>
<td>1520</td>
</tr>
<tr>
<td>Shale</td>
<td>2.4-2.8</td>
<td>2560</td>
<td>1680</td>
</tr>
<tr>
<td>Silica Sand</td>
<td>2.2-2.8</td>
<td>2560</td>
<td>1680</td>
</tr>
<tr>
<td>Slate</td>
<td>2.5-2.8</td>
<td>2720</td>
<td>1760</td>
</tr>
<tr>
<td>Talc</td>
<td>2.6-2.8</td>
<td>2640</td>
<td>1760</td>
</tr>
<tr>
<td>Trap Rock</td>
<td>2.6-3.0</td>
<td>2800</td>
<td>1840</td>
</tr>
</tbody>
</table>

Figure 6.6 Drilling patterns using face holes (often referred to as Snake Holes) at toe of face
CHAPTER 6 BLASTING AND BLASTING PRACTICES

6.12 QUARRYING USING SMALL DIAMETER HOLES

Some small and medium-sized quarries are equipped with relatively small primary crushers and use small diameter holes drilled to the full depth of the quarry face to secure improved fragmentation and output. The height of quarry faces is restricted by legislation (usually a maximum of 20 metres). Small diameter holes (under 50 - 75 mm) are usually limited to depths of 5 to 6 metres. Faces higher than 6 m can be broken by using a combination of vertical, inclined and horizontal holes drilled by jack-hammers, air-leg machines and wagon drills. (See Fig. 6.6).

QUARRYING USING LARGE DIAMETER HOLES

Where an open pit or quarry is set up to handle large tonnages per day, loading arrangements at the face, and the size of the primary crusher usually permit the rock to be broken larger in the primary blast than it would be in a quarry with small equipment.

Larger diameter holes are drilled with percussion and rotary drilling machines: Percussion drilling is for harder rocks and rotary drilling for medium to hard formations. For soft material, auger-type rigs may be used. Inclined drilling, together with sub-drilling, is often used to reduce the possibility of a residual toe. Hole sizes range from 100 mm to 380 mm in diameter.

AN-C or slurry blasting agents are used as explosives, but large diameter holes are particularly suitable for ammonium-nitrate, fuel oil (AN-FO) explosives of low density. The charges are mechanically placed, i.e. pressure loaded or pumped, and priming is done by Cordtex and high explosive boosters. More than one primary charge is often used but the explosive, priming and loading pattern depends on hole size, wet or dry conditions, blast pattern and the fragmentation required.

BLASTING IN COAL MINES (UNDERGROUND)

Explosives used to break coal may be either permitted type (where coal is gaseous) or non-permitted (where no gas is encountered). However, low strength explosives are often preferred to avoid excessive breakage of coal.

Explosives used for blasting coal:
(a) hard coal requires a high strength, high density explosive;
(b) medium coal requires medium strength explosive;
(c) soft or friable coal requires a low strength, low V.O.D. explosive of low density.

COAL BLASTING

Two methods are used.

1. Blasting from the solid (grunching).

This method uses a number of holes which are drilled by a hand-held power auger in a fan or drag pattern. (See Fig. 6.7).

Underground mining at Collie uses a more refined version of the drag type cut, i.e. a full face of holes up to 27 – 32 in number. The explosive used is non-permitted Semigel because the seams are non-gaseous.

Shot hole patterns used in coal mining vary according to seam thickness, position of the cut, presence of partings on dirt bands, seam roof coal characteristics and the two methods of working i.e. long wall face or narrow workings (bord and pillar). Shot holes are usually single or double and roughly parallel, adjacent or offset.
2. Blasting cut coal.

The free face for the shot holes is provided by under-cutting a kerf usually at the bottom or near the bottom of the coal face. The cut is usually approximately 150mm wide and 1.25 to 2.5 metres into the face. (See Fig. 6.8).

Variations are adopted according to seam thickness and openings and vertical kerfs may be used.

**Figure 6.8 Blasting coal faces with cut kerf**

**CUT HOLES**

It is necessary to provide an artificial free face to blast into for economical blasting in tunnels, shafts and developmental headings in mines. This initial opening, termed a cut, is fired as a preliminary, often a part of, the main blast, particularly in hard rock conditions where ground is tight and not readily fragmented.
TYPES OF CUTS USED
The cut should possess properties consistent with the physical characteristics of the rock being blasted, such as smallest possible size, maximum safety, and minimum drilling and explosives consumption. Obviously it is necessary to ensure successful rock displacement and fragmentation rather than to fail to bring out a round.

Actual cuts adopted depend upon ground (rock) character, presence of joints and planes of weakness as well as size and type of drill holes, equipment heading size and the depth of round being pulled per blast.

A range of cut types may be identified:

1. Centre Cut.
   This cut also referred to as a pyramid or diamond cut, concentrates the explosive at the apex of the pyramid. It is effective but much flyrock results and explosive consumption is high (See Fig. 6.9).

2. The Wedge Cut.
   This variation uses a Vee or plough cut, where four or six holes converge. It is less effective in tough ground and requires a wedge angle not less than 60°. This limits the depth achievable per round. However, it may be suitable for some wide face headings. (See Fig. 6.11).

3. Drag Cut.
   This variation of the angled cut uses holes to drag from a free face. The variations are:
   (a) floor;
   (b) back (may result in back shatter);
   (c) to one side (fan);
   (d) to a convenient weakness e.g. shear plane.
   The drag cut is suited to shale or laminated rock, but is less effective than the centre cut in hard ground. It reduces concussion and fly impact in close proximity to installations. (See Fig. 6.13).

4. Burn Cut.
   This cut is suited to hard, brittle homogeneous rock. A wide variety of patterns may be adopted to suit heading sizes, equipment and ground characteristics.
   A significant advantage of the burn cut is that the holes are normal to the face allowing longer holes to be drilled and longer rounds to be pulled. (See Fig. 6.14).
   Burn cuts are designed to bring out the fragmented material; therefore certain holes are charged with explosives while others are left to serve as miniature free faces.
   Holes which are charged are fired in a specific sequence using ample delay between consecutive shots to allow fragmented material to clear or be expelled by the shot. (See Fig. 6.15).
5. **Large Hole Cut.**

This arrangement has evolved from the burn cut. Certain of the uncharged holes, usually one or two of the centre holes, are reamed out to provide a more effective miniature free face for the initial opening up of the burn cut.

This is typified by the Coromant cut which uses a template for hole location, and also provides two overlapping large diameter holes which are left uncharged. (See Fig. 6.16).

The cut holes in burn cuts are often drilled some 150 mm deeper than the remainder of the round. This ensures that the other holes in the round leave clear bottoms, without hole remnants or butts, for drilling of the next round. Normal rounds for development headings are 1.5 to 1.8 metres; however burn cuts and large diameter burns pull between 2.5 to 3 metres. The length of round will also be regulated to allow drilling, charging, firing and excavating to be done on a cyclical shift basis.
CHAPTER 6 BLASTING AND BLASTING PRACTICES

6.16

Figure 6.12 Typical vee or wedge cut

Figure 6.13 Drag cut in drive face (Floor)

Figure 6.14 Typical burn cut

Figure 6.15 Burn cut variations
BLASTING IN STOPING OPERATIONS

The extraction of ore from metalliferous mines is achieved by stoping operations. Ore is fragmented from the mass by drilling and blasting, but blasting in stoping operations must provide the required fragmentation without adversely affecting the stability of the walls. Therefore holes are placed so as not to shatter the walls, e.g. back-, floor-, foot- or long-wall, depending upon the incline of the ore body. Holes are spaced to suit burden distance (See page 6.10) corresponding to the physical dimensions of the holes drilled. Free faces always exist so that parallel holes to the free face are used to achieve blasting fragmentation.

CONVENTIONAL STOPING

Figure 6.17 Overhand Stoping with Horizontal Holes is the normal method for taking horizontal breasts or slices about 2.5 metres high progressively along a stope. (See Fig. 6.17).

Vertical up holes are avoided in stoping because ragged unstable backs are likely to occur. Instead, up holes inclined at about 60° to the horizontal are used especially for mechanized cut and fill stoping. (See Fig. 6.18).
CHAPTER 6 BLASTING AND BLASTING PRACTICES

Long hole stoping is used where massive wide ore bodies with good standing wall or country rock exist. Series of fan-like patterns of blast holes are drilled from sub-levels; these patterns are referred to as ring and have spacings up to 2.5 metres between them. (See Fig. 6.19).

Figure 6.19 Sub-level stoping using long hole drilling (ring drilling)

LONG HOLE SUB-LEVEL STOPING

Blasting in development headings includes shaft sinking, tunnelling, driving, crosscutting, rising and winzing. Because a free face does not exist cut holes are required and skill in drilling, loading up and blasting is necessary to obtain satisfactory results. The hole of least resistance is fired first so that additional free faces are established.

TUNNELS

Tunnel driving is common to construction projects in civil engineering work. Excavations are made in all types of ground conditions, each one requiring a different technique.

1. Soft unconsolidated ground requires the use of tunnelling shields and/or permanent support such as concrete or steel liners.
2. Soft to medium rock allows the use of tunneling machines such as moles.
3. Hard rock, in which conventional drill/blast/muck cycle is necessary.
Tunnels range from 3 to 15 metres in diameter and from metres to many kilometres in length. The large diameter headings are usually horseshoe shaped, with an arched roof and horizontal base or floor. They are used for railway and highway access in urban underground transit, mountainous terrain and under rivers; they are also used in mining and in hydro-electric aqueduct systems. They are drilled by jumbo rigs of various types. (See Figs 6.20 and 6.21).

Figure 6.20 Full heading using two large diameter burn holes (125 mm diameter) detonators (the numbers indicate the half-second delay used)

Figure 6.21 Large heading using a double vee cut. (vee fired using milli-second delays: remainder half-second delay detonators)

SHAFTS
Shafts for mine development are either vertical or inclined. Vertical shafts may be rectangular or circular, while inclined shafts are rectangular. Shaft sinking is often done by specialist contractors.

Centre or burn cuts are used where mechanized mucking out is used, which is almost universal. For rectangular shafts using hand mucking, bench cuts (a variation of a drag cut) are used.
Centre cuts produce considerable throw which may damage shaft supports. Misfires of the centre cut usually mean the loss of the round. Because shaft sinking is usually under wet conditions the occurrence of misfires is to be avoided. (See Figs 6.22 and 6.23).

Figure 6.22 Shaft sinking blasting, pyramid (centre) cut in a circular shaft. Centre cut fires in sequence using milli-second delays; remainder are second delay/detonators.

Figure 6.23 Shaft sinking blasting in rectangular shaft using vee cut which may result in poor fragmentation

Burn cuts, which result in less fly rock, are preferred to centre cuts. (See Fig. 6.24). Short delay blasting in any shaft or development heading round can give control of the shape and nature of the final muck pile.

The bench cut or sump cut method is favoured for rectangular shafts where hand mucking is used. A partial free face is used for the drag type round. The bench is high and dry and the sump allows dewatering.
A general aspect of shaft sinking is that a harness of electric detonators can be used. Shaft rounds with up to 150 electric delay detonators are possible with a factory made firing harness.

![Figure 6.24 Large diameter burn hole cut in shaft sinking round and fired with milli-second (short delay) detonators](image)

**DRIVES AND CROSSCUTS**

Drives and crosscuts are headings advanced horizontally along or across the strike of the country or ore body bend, to connect with vertical development. Cross cuts achieve more effective shearing of rock from blasting because they intersect the bedding planes.

Drag, wedge, centre, burn and large diameter centre cuts are used; the choice depending to some extent on local conditions.

Once the cut holes have been fired, holes in the remainder of the round are fired in a specific sequence. The holes in a round are referred to according to their position and purpose: i.e. cut casers, side holes, knee holes, shoulder holes, back holes and the lifters. (See Figs 6.25 and 6.26).

The lifters and back holes are angled down and out to maintain floor grade, arched back and overall section size. Where a drainage channel is used, one side is placed lower to accommodate this.
WINZES
These are small shafts, either vertical or inclined, sunk from one level to another. Hand mucking is generally used because of the small sections. Bench cuts are often used. (See Fig. 6.27).

RISES
These headings are advanced upwards and on blasting rock removal is by gravity. Adequate ventilation as well as access are apparent problems of this type of opening.

Burn cuts are the most common; the drilling is done from a staging. (See Fig. 6.28).

Mechanized raise access may be accomplished with raise climbers such as an Alimak unit. Another variation is the bore hole rise where a cage is suspended on a rope to enable drilling out and explosive loading.

Both rises and winzes, made by conventional methods, especially for ventilation and ore pass purposes, are being replaced by raise boring back reaming where cuttings are removed by gravity.
Figure 6.27 Sump cutting or benching for shafts and winzes

Figure 6.28 Conventional rise using a burn cut
## Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
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<tbody>
<tr>
<td>BLAST HOLE (BORE HOLE)</td>
<td>A hole drilled into rock to accommodate an explosive charge for blasting (breaking) rock or ore.</td>
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<td>BOTTOM PRIMING</td>
<td>A method in which the primer is placed near the bottom of the charge or hole.</td>
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<tr>
<td>BULLED HOLE</td>
<td>A shot hole which has been bulled or chambered by exploding a light charge of high explosive at the bottom. The chamber at the bottom can then accommodate a large quantity of explosive.</td>
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<tr>
<td>BURDEN</td>
<td>The volume of rock which lies within the zone of influence of a charge of explosive; the volume of rock to be broken by any hole or charge.</td>
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<tr>
<td>BURDEN DISTANCE</td>
<td>The distance between the main body of a charge and the nearest free face.</td>
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<tr>
<td>BUTT</td>
<td>The part of the hole that remains after the charge has been fired; it may contain unexploded charge, therefore drilling in both is extremely dangerous.</td>
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<tr>
<td>CHARGE</td>
<td>The explosive or blasting agent used in a blast hole.</td>
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<tr>
<td>COLLAR PRIMING</td>
<td>A method in which the primer is placed near the top, or at the collar end of the charge.</td>
</tr>
<tr>
<td>CUT</td>
<td>An artificial opening made in a face to provide a free face for blasting. It may be made mechanically as in coal mining or by explosives.</td>
</tr>
<tr>
<td>EASERS</td>
<td>The shot holes which break out to the free face formed by the cut holes.</td>
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<tr>
<td>KERF</td>
<td>A cut made in a coal face by a hand-pick or by a mechanical coal cutter to provide a free face for blasting.</td>
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<tr>
<td>LIFTER</td>
<td>The bottom holes of a round pattern for a drive or tunnel heading.</td>
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<tr>
<td>MISFIRE</td>
<td>A charge or part of a charge which for one of any number of reasons has not exploded. Specific safety procedures must be observed.</td>
</tr>
<tr>
<td>PATTERN</td>
<td>A dimensioned plan of holes to be drilled for blasting a face.</td>
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<tr>
<td>PULL</td>
<td>The quantity of rock excavated in a unit of advance in a heading, e.g. by drilling and blasting a round.</td>
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<tr>
<td>ROUND</td>
<td>The series of blast holes required to produce a unit of advance in a development heading or other face.</td>
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<tr>
<td>SEQUENTIAL FIRING</td>
<td>A system in which the holes with least resistance are detonated progressively, reducing the burden on each subsequent hole fired.</td>
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<tr>
<td>Term</td>
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<tr>
<td>SPACING</td>
<td>The distance between adjacent shot holes parallel to the free face.</td>
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<tr>
<td>STEMMING</td>
<td>Usually the sandy clay material prepared and wrapped in cartridge form and used for sealing a blast hole after the charge has been placed. The term may refer to any material used to seal a charge in a blast hole.</td>
</tr>
<tr>
<td>STOPE</td>
<td>The chamber or excavation from which ore is extracted underground.</td>
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<tr>
<td>TAMING</td>
<td>The act of charging or ramming a charge into a hole with the aid of a tamping stick.</td>
</tr>
<tr>
<td>TOE</td>
<td>A remnant of rock left unbroken at the foot of the quarry face by an unsatisfactory blast</td>
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CHAPTER 7 – Practical Use of Explosives

PLACEMENT OF EXPLOSIVE CHARGES

Explosives are used predominantly to break ground in mines, collieries, quarries, tunnels and other rock excavations. For most effective use, explosives are placed in bore holes which have been drilled in strategic positions within the rock mass.

Detonation of the placed explosive weakens the cohesive strength of the rock by brisance and creates a high gas pressure in the hole. A compressive strain pulse travels outward from the hole until it is reflected from a free face as a tensile strain pulse. The rock is fragmented under tension and displacement is assisted by gas pressure.

CHARGING SHORT HOLES USING CONVENTIONAL CARTRIDGED EXPLOSIVE

A number of operations which are adopted for preparing the blast hole and charging are listed:

1. The hole is flushed out by means of a compressed air blowpipe.
2. Explosive cartridges are inserted into the bore hole one by one and firmly squeezed into position with a wooden tamping stick in order to occupy the full cross section of hole.
3. The primer cartridge is placed last in the hole when conventional safety fuse and plain detonators are used. Care is taken to ensure that no damage occurs to the safety fuse protruding from the mouth of the hole.
4. The hole may be sealed using a stemming cartridge or material such as sandy clay. (See Fig. 7.1).
5. Where the charge is fired electrically, the electric primer is the first cartridge to be placed in the hole. Note that in all types of detonator, the base of the detonator points towards the bulk of the explosive charge. The lead wires of the detonator are held taut to one side of the hole while the remaining cartridges are inserted and tamped into position. (See Fig. 7.2).
CHAPTER 7 PRACTICAL USE OF EXPLOSIVES

7.2

Suggested Methods Using Safety Fuse rods

1. Single shot holes
2. Multiple holes or rotation firing
3. Multiple holes or rotation firing

Figure 7.1 Charged blast holes using plain detonators and safety fuse

Suggested Methods Using Electric Detonators

1. Using instantaneous electric detonators
2. For rational firing using delay electric detonators

Figure 7.2 Charged blast holes using electric detonators
PRECAUTIONS TO BE TAKEN DURING CHARGING

The main danger involved in charging a bore hole with explosive is that of premature explosion. The danger arises from a glancing blow of a film of explosive smeared on a hard surface. The following precautions may minimize this risk.

(a) Holes: clean out thoroughly prior to charging. Cartridges are then not ruptured.

(b) Cartridges must not be forced into the hole; the cartridge size should be smaller than the hole diameter but with tamping, should fill the hole cross section.

(c) Metal or metal tipped tamping rods are never to be used.

(d) The detonator must be secured to the primer cartridge.

(e) Cartridges must not be split or cut to aid tamping.

CHARGING BLAST HOLES USING PERMISSIBLE EXPLOSIVES

The procedure adopted in coal mines requiring permissible explosives is designed to avoid initiation of gas or dust. Primer placement is at the hole bottom, and is referred to as Indirect Initiation. (See Fig. 7.3). This involves placing the primer as the first cartridge into the bore hole with the detonator base pointing outwards. Advantages attributed to this approach are reduction in misfires and burning charges.

![Figure 7.3 Indirect priming or initiation for permissible explosives](image)

Where watergel and slurry type explosives in cartridge or sausage form are involved, pneumatic cartridge loaders may be used. Molanite (cartridge, small diameter, detonator sensitive, slurry explosive) can be charged with relative ease.

CHARGING SMALL DIAMETER BLAST HOLES USING AN/FO

Charging of AN/FO is done usually with pneumatic equipment allowing placement in any direction.

There are two basic types of pneumatic loading machines:

A. Ejector type, in which AN/FO is drawn from the bottom of an open-topped container and injected into the air stream in the placement hose and conveyed into the bore hole at high speed and high pressure. The principle of operation is a venturi action using air pressures of 450 to 750 kPa.

A well-known example is the 'Penberthy Anoloder' which will load AN/FO into small diameter holes from 25 mm upward: placement capacity varies from 2 to 7 kg per minute.
It consists of an aluminium hopper of 12.5 kg capacity, ejector assembly, and hand control valve with a length of 'Lo-stat' plastic loading hose. Construction is light and the machine has support legs and carrying strap. Operation involves the AN/FO being drawn from the hopper into the centre of an annular air jet within the ejector assembly then ejected into the bore hole through a length of semi-conductive plastic tube. Loading is controlled by the hand control.

A particular feature of the ejector type loader is that the jetting action of AN/FO into the borehole causes the prills to break and pack which increases the hole loading density as well as the sensitivity of the charge. See Figs 7.4 (a) and (b).

B. Pressure-ejector type, is a combination unit, in which AN/FO is injected into the airstream in the placement hose under 700 kPa pressure from an enclosed hopper pressurized to 170 to 200 kPa. Two compressed air connectors are necessary; the hopper is connected through a pressure reducing valve.

An example of this machine is the N.V.E. loader which loads AN/FO into holes up to 50 metres and at a rate of up to 45 kg per minute. (See Fig. 7.5 (a)).

![Diagram of Ejector Type AN/FO Loader](Reproduced by kind permission of Mining Supplies (Minsup) Pty Ltd, Adelaide)

*Figure 7.4 (a) Ejector Type AN/FO Loader (Minsup)*
Figure 7.4 (b) AN/FO Loader incorporating transport container/hopper

Figure 7.5 (a) Pressure - ejector loader (minsup)
SAFETY REQUIREMENTS FOR PNEUMATIC LOADING OF AN/FO

When AN/FO is charged pneumatically into bore holes static electricity may be generated. Therefore bottom priming using electric detonators may be hazardous where rock is of low conductivity and the relative humidity of the air is low. Protected detonators which have an insulating sheath fitted over the fusehead to reduce sensitivity to electrostatic discharge are therefore mandatory.

Suggested requirements for pneumatic loading are:

1. A conductive hose should be used.
2. The prill container and metal loading unit should be grounded with a metal cable, (See Fig. 7.5 (b)).

The suggested type of loading hose is that known as Lo-stat hose which is of semi conductive material readily identified by a yellow stripe along its length: body colour is black, diameters available are 16, 19 and 25 mm.

Policy on position of priming varies with state authorities, companies concerned, operating conditions and the type of explosive and detonators being used. When rock formations and conditions exist where cut-offs occur, i.e. a blast hole cuts off part of a neighbouring hole that is timed to explode later, then bottom priming is necessary. (See Fig. 7.6). This can be achieved effectively and safely using either bottom priming with electric detonation; for long deep holes, detonating cord can be used.
TAMPING
Placement of cartridges of explosives in small diameter underground blast holes is usually accomplished by using a tamping stick or rod. Wooden dowelling of approximately 20 mm diameter is common although approved non-metallic tamping rods may be used. Metal rods or those with metal ferrules should never be used.

Tamping of cartridge explosives, especially those of nitroglycerine composition, involves a certain risk or danger of premature explosion. Precautions, involving great care, are necessary. Procedures to be adopted are:

- Holes should be thoroughly cleared out and free of obstructions prior to charging.
- Cartridge diameters should be less than the minimum diameter of the hole.
- Cartridges need to be pressed firmly into place and enlarged by squeezing with the tamping rod so as to fill the cross section of the hole. Thumping or pounding is not recommended.
- Care should be taken not to dislodge the detonator or damage the fuse or the lead wires.
- Particular care should be exercised when loading and tamping soft plastic cartridges of slurries.

STEMMING OF EXPLOSIVE CHARGES
To localize the gases produced in an explosive reaction, the charge must be confined. An unconfined explosive charge, on initiation, results in excessive noise and concussion with little useful work being achieved. Charges placed in bore holes must be sealed or stemmed to prevent the gases from escaping prior to contributing to the fragmentation and throw of the rock. An effective stemming material is sandy clay which is usually formed into cartridges which are tamped into the hole above the charge.

Non-combustible material should be used, especially in coal mines. Paper or wooden spacers may burn or result in gaseous contaminants (smoke) unwanted underground. Water stemming bags have been used for quenching and cooling in coal mine blasting to minimize the risk of gas/coal dust explosions.

Where low charge mass is required in blasting, use of spacers of stemming material may be applicable: in surface quarry blasting this is referred to as deck loading. An alternative is to use low density explosives such as those with additives e.g. polystyrene beads. However while these are of practical relevance to controlled blasting, inherent disadvantages are toxic fume generation and the need for amendment to manufacturing licence if preparing the material on site.

USE OF AN/FO UNDERGROUND
AN/FO finds general application in underground blasting except for wet conditions. Hence it is not generally used for shaft sinking, winzing or under wet conditions.

When properly mixed and primed AN/FO possesses good fume characteristics and does not produce fracture headaches associated with nitroglycerine based explosives.

Pneumatic loading offers good rates of loading with the complete hole cross section being effectively filled. Fragmentation is generally improved because of better coupling and gas volume generation. Application is for short, small diameter holes, long holes and ring drilled holes in sub-level stoping.
In primary blasting AN/FO is taken underground in bags which can be fed to pressure-type pneumatic loaders for sub-level stope blasting, shrink, and cut and fill stoping.

For secondary blasting of oversize material small bags of AN/FO may be used for plasticing—clearing of chutes and passes.

For blasting in small development headings ejector type AN/FO loaders are commonly used.

**USE OF AN/FO FOR SURFACE BLASTING**

AN/FO is the most economical explosive for blasting of rock where conditions permit its use. However there are two disadvantages:

(a) low loading density when gravity fed into down holes;

(b) inherently low water resistance.

A partial remedy to the second problem is to use polythene sleeves. This is trade named 'lay-flat' obtainable in a range of hole diameters and two plastic wall thicknesses.

To overcome the effect of wet conditions and still use ammonium nitrate, use is made of AN slurry type blasting agents.

Their characteristics include higher density dependent upon compositional additives. Gelling agents mixed with the material on entry to the blast hole exclude water encountered in the hole, actually displacing water out of the hole, if full. Densities range from 1.05 to 1.3 g/cm³. The V.O.D. of slurries is higher than AN/FO ranging from 3350 to 5500 metres per second, depending upon composition, charge diameter, degree of confinement and density.

An example of a powerful metallized slurry manufactured at the blast hole site in a slurry mix truck is IREGEL 376 SD offering high strength, density of 1.25 gm/cm³ and high water resistance.

The density and energy release of AN/FO can be increased by the addition of aluminium powder, the high heat release being the main feature.

**CHARGING OF SLURRIES**

Where slurries are used for charging small diameter holes underground either in development headings and stopes, cartridged forms are used; these include products such as Molanite or Trovex.

For long holes and large diameter holes in excess of 130 mm such as surface mine/quarry blast holes slurries are charged in bulk.

Although slurries have a density greater than water, gravity placement by pouring into the blast hole is not always effective for displacement of water. Hole dewatering is generally carried out and the slurry mix pumped into the hole under pressure through a loading hose. The charging hose is lowered to the hole bottom, loading is commenced and the hose is withdrawn as the slurry progressively fills the hole.

**MIXING OF BLASTING AGENTS**

AN/FO is usually mixed on the job site in the following ways.

For small scale operations the correct amount of fuel oil is introduced into a bag of AN prills, and the contents are then later fed into a loading unit or poured into the blast hole.
For medium scale operations, mechanical mixing in batches in a concrete mixer lined with fibreglass is often used.

A mechanical mixer specially designed for AN/FO manufacture is that known as the COXON ANFO MIXER. Two models, operated by either hand or electricity, are available although other motive power sources are available.

**Production rates are:**

- for hand operation - up to 1 000 kg/hr
- for electric operation - up to 6 000 kg/hr

The unit consists of hopper and auger mixer. A pump, driven in relation to the auger speed, supplies the correct quantity of oil into the ammonium nitrate stream entering the auger.

For mines with large AN/FO consumption, bulk mixing plants are established nearby. Pre-mixed quantities are bagged and despatched underground or into loading trucks for quarry blast hole loading.

On site mixers using AN/FO mixer trucks are available in sizes ranging from 3 to 11 tonnes in capacity. These are known as Nobel-Amerind Mixer Trucks. They transport AN prills and fuel oil separately, mix them at the site and deliver the mixed AN/FO to the bore hole.

Mixing of AN and fuel oil constitutes the manufacture of an explosive, therefore licensing under the control of Chief Inspector of Explosives, Explosives Branch - of the Mines Department for Western Australia, is required.

**BLASTING AGENTS – PRIMING**

The low sensitivity of AN/FO (and slurries) requires care with priming for satisfactory initiation. For small diameter holes underground up to 75 mm in diameter, a single cartridge of AN Gelignite “60” with a No 6 detonator, is a sufficient primer where the AN/FO is well mixed, and in a continuously packed column. For large holes in quarries priming requires particular care especially where holes are wet or where decked charges are used. The V.O.D. of the primer should be greater than the inherent V.O.D. of the charge. Anzomex cast boosters, which have a velocity of detonation of 7000 m/sec are used. Down lines of detonating cord are generally used.

Primer position is important with bottom priming yielding good results when the primer is either above or below the bench floor level. Two primers on the down line located above and below the bench level can provide excellent initiation.

**MISFIRES**

A charge that has not exploded or has only partly exploded is referred to as a misfire and requires extreme caution.

The first precaution is to allow a safe interval of time (specified in Statutory Regulations) between the report of a misfire and the approach to the face.

Prior to removing as well as during mucking out of broken rock from the face, care should be taken to identify unexploded cartridges and/or detonators and handle them as circumstances demand.
The face requires careful examinations for remaining butts which may contain remnants of a charge. Butts should never be drilled.

Misfires can be caused by defective explosives (storage deterioration), damage to components while charging, water entering the detonator, discontinuity of the electrical circuit, insufficient current in a circuit, and hole cut offs.

Capped safety fuse may result in misfires due to damp/wet conditions in storage. Storage in plastic bags assists in overcoming this problem.

Poor cutting, crimping and waterproofing during capping may also cause misfires. Bottom priming using safety fuse and plain detonators may result in the kinked section of the fuse breaking the powder train.

Electric detonators may be responsible for misfires where internal breakages of detonator leads and of firing cables occur. Adequate testing of circuit components is essential to eliminate discontinuities.

Exploders may also give rise to misfiring and regular testing and servicing is essential. The exploder capacity should not be exceeded.

Misfires involving nitroglycerine based explosives should be refired.

AN/FO misfires can be resolved by sluicing out with water or other approved means. However the presence of a primer may require refiring.

**EXPLOSIVE PACKAGING**

High explosives are usually packed in fibreboard cases of 25 kg capacity containing the cartridges in a sealed plastic bag. Each case contains a number of wrapped cartridges depending upon the diameter, length and density of the particular explosive. Cases are marked on each end with the explosive name, the cartridge diameter, date and batch symbol. Ammonium nitrate prills are packed in 30 kg bags and 160 kg drums.

**EXPLOSIVE STORAGE**

Regulations governing the safe handling and storage of explosives are the concern of the Chief Inspector of Explosives. In Western Australia the Explosives and Dangerous Goods Act and Regulations specify the requirements for explosives and blasting agents. Actual standards of recommended practice can be seen in Australian Standard 2187 Part 1 SAA Explosive Code, Storage, (1979) Part 2 Use of Explosives (1983).

Storage and handling of explosives and blasting agents on mine sites must meet the standards set down by the Chief Inspector of Explosives. Magazines for surface storage require licensing where more than 250 kg of explosive or blasting agents are stored. For lesser amounts magazines have to be of approved construction.

Storage of explosives in magazines must provide security as well as prevent the chance of premature explosion. Storage is aimed at preventing detonation. Under-ground storage and handling is specified under the Mines Regulation Act and Regulations.

High explosives must be stored separately from detonators. Because explosives deteriorate with age, reasonably rapid turnover is required. Any deteriorated stocks will need to be disposed of in a manner approved for the material in question.
DISPOSAL OF UNWANTED OR DETERIORATED BLASTING MATERIALS

Explosives, blasting agents and detonators may need to be disposed of due to deterioration or because they are unwanted material. Disposal may be accomplished by:

- detonation;
- burning;
- immersion, or sea burial.

The method adopted will depend on circumstances and the type of material. Advice of explosive field representative, inspector of mines or police should be sought as to the safest method.

---

<table>
<thead>
<tr>
<th>EXPLOSIVE</th>
<th>STOCK PRODUCTS</th>
<th>PRODUCTS AVAILABLE ON SPECIAL ORDER</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Diam (mm)</td>
<td>Length (mm)</td>
</tr>
<tr>
<td>AN Gelatine Dynamite '95</td>
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<td>200</td>
</tr>
<tr>
<td></td>
<td>55</td>
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<td>560</td>
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<tr>
<td></td>
<td>65</td>
<td>520</td>
</tr>
<tr>
<td>'ANZITE' Blue</td>
<td>(in plastic 'GEOLOK' cylinders)</td>
<td>90</td>
</tr>
<tr>
<td></td>
<td>65</td>
<td></td>
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<td>PLASTERGEL</td>
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<tr>
<td>SEMIGEL No 2</td>
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<tr>
<td>'MORCOL'</td>
<td>32</td>
<td>190</td>
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<tr>
<td>'HYDROMEX' (in film)</td>
<td>55</td>
<td>410</td>
</tr>
<tr>
<td>'MOLANITE' 95 (in film)</td>
<td>65</td>
<td>410</td>
</tr>
<tr>
<td>'MOLANITE' 110 (paper wrapped)</td>
<td>25</td>
<td>200</td>
</tr>
<tr>
<td></td>
<td>25</td>
<td>300</td>
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<td>'MOLANITE' PLASTERPAK</td>
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<td>(in film)</td>
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<tr>
<td>(in film)</td>
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<tr>
<td>'METABEL' 750 cm x 250 mm x</td>
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<td></td>
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<tr>
<td>2.2 mm</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 7.1 Explosives - cartridge counts & dimensions (per 25 kg case)
CHAPTER 7 PRACTICAL USE OF EXPLOSIVES

POINTS ON EXPLOSIVE USAGE

The shot firers should be trained, preferably non-smokers and possess habits which exhibit self-discipline.

In mining operations miners collect and store their own stocks of explosive (working party magazines), charge and fire holes.

In other situations if possible, it is preferable to employ shot firers for explosive usage. For explosive usage other than mining a person may possess a Shot firer’s Permit obtainable from the Explosives Branch of the Mines Department of Western Australia.

Key points which are relevant to shot firers of all categories are:

- Avoid smoking.
- Handle explosive materials with care.
- Keep blasting materials dry.
- Use safety fuse of sufficient length.
- Don’t use force to implant the detonator into a cartridge.
- Don’t screw or force safety fuse into a plain detonator initiating composition.
- Safety fuse should have a square clean end for insertion into the detonator.
- Crimp a detonator away from the base charge; use approved crimping tools not your teeth.
- Avoid kinking or bending of safety fuse.
- During tamping, squeeze cartridges securely into place. Don’t pound.
- Use only wooden or approved tamping rods, never metal tubes or rods.
- Avoid kinking of lead wires for electric detonators.
- When inserting the electric primer cartridge hold wires taut to avoid wire kinking but don’t jerk to straighten.
- Carefully check all connections for a round prior to connecting a firing cable or connection. Maintain personal possession of the exploder key.
- Return to a face after a blast only when:
  (a) exploder is disconnected and key removed;
  (b) an adequate period of time has elapsed (dependent upon type of detonator initiation used).
- Withdraw all personnel to a safe place prior to firing.
- Exercise extreme caution with misfires.
- Report all misfires especially to the oncoming shift, or leave instructions for the cross shift.
- Store all blasting material in the appropriate receptacle and place; never discard or leave about.
- Do not load a bulled or chambered hole unless it is cool.
- Be conversant with and observe relevant safety code requirements for your job site.
EXPLOSIVE ACCIDENTS

Extreme care must be exercised in the handling of explosives.

Most explosive accidents are caused by negligent disregard of the Statutory Regulations for safe practice:

- Safety fuse too short or delay in fuses.
- Drilling into explosives – drilling in butts.
- Premature firing of electric detonated rounds.
- Returning too soon after a blast (danger of delayed shots).
- Personnel entering blasting area if entry unguarded.
- Unsafe practices in handling, transport and storage.
- Lack of caution in dealing with misfires.
- Smoking when handling explosives.

SPECIAL BLASTING TECHNIQUES

CHAMBERING DRILLING HOLES

The bottom of small bore holes may be enlarged by exploding a small charge electrically. No stemming other than water should be used. The pulverized rock from the vicinity of the hole is expelled by gases escaping from the blast. Sufficient time should be allowed prior to loading the main charge of explosive, or the hole should be cooled with water.

COYOTE OR TUNNEL BLASTING

Where very large quantities of rocks are to be blasted and free faces exist such as spur ridge or side of a hill, larger quantities of explosive can be placed within excavated chambers than in bore holes. The explosives are packed into the chambers in the cases and the passage ways back-filled. Charges are electrically detonated with detonating cord as trunk lines.

Conditions Applicable For This Approach

- Quarries and heavy hillside cuts where conventional drilling methods are impracticable because of high costs. An example of a coyote blast underwater was the removal of Ripple Rock from Seymour Narrows, British Columbia in 1958. The blast, one of the largest brown non-nuclear charges fired was (2756 000 lbs), approximately 1.23 mega tonnes.
- Rock types where the desired degree of fragmentation can be obtained simply by dislodging and heaving out a large mass of material.
- The production of large rock pieces suitable for breakwaters. (See Fig. 7.7).
SECONDARY BLASTING

Where rock is blasted from a face, except underground development headings, some may break into pieces too large to be handled by loading equipment, trucks, chute openings, and crushers. The equipment in use necessarily limits the size of material that can be handled safely and economically. Oversize must be reduced, usually by explosives. This procedure is referred to as secondary blasting and is achieved by popping or plastering.

POPPING

Short, small diameter holes (30 - 40 mm diameter, 100 - 150 mm deep) are drilled into the boulder. A light charge, usually a high strength gelignite, is used to crack the boulder. A very large rock may require several such holes. The advantage of this method is that a small quantity of explosive is required; the aim is to break the rock without undue scattering. The main disadvantages are fly rock scatter and the equipment required.

PLASTERING

This method does not involve drilling a hole but requires a much larger explosive charge. The charge is placed on top of the boulder if possible in a depression and covered with mud or clay. Plastergel is commonly used, however small packs of AN/FO are also used. Advantages of the method are no boring, little flyrock and relative speed of preparation. Disadvantages are that approximately four times as much explosive is used and noise and concussion from relatively unconfined blasts can be a nuisance.

Because of the disadvantages associated with the use of explosive for secondary blasting and breaking other methods have won acceptance. These approaches may allow operations - such as loading and trucking to proceed without interruption. For small quantities of oversize on grizzlies hand spalling may be practical. At the quarry face drop balling may be useful, or alternatively oversize may be removed to an area set aside for this task. In drop balling a cast steel ball is dropped by a crane onto the oversize rock; the ball’s mass is between 1.5 to 2.0 tonnes. The cost advantage of such methods is that clearing of the work site is not necessary while reduction is being carried out.

OTHER METHODS

These include wedging devices where a hole is drilled and a wedge is inserted to shatter the rock. Impact breakers, especially hydraulic-powered, mounted on back-hoe type booms are finding wider application especially at fixed grizzlies and at crusher feeder openings.
OVERBREAK CONTROL

In civil engineering tunnelling projects particular emphasis is placed on gauge line near the perimeter of the excavation. Contacts may involve a penalty clause for material inside this which may need stripping; likewise where an excavation is to be concrete lined overbreak may also incur penalties. It is generally important then, that the walls of the excavation be smooth and as close to the nominal gauge line as possible.

Another significant aspect is the effect of shattering and resultant instability of the remaining rock mass after blasting.

In mining these aspects contribute to reduction in overbreak, better stability, and lower dilution of ore broken especially in underground headings and stopes.

A number of techniques, all of which involve the drilling of a line of holes along or around the perimeter gauge line, in order to provide a line of weakness, or fracture plane, along which the rock will separate when the main round is fired, are mentioned. They aim to finish with a relatively smooth rock surface.

The methods are: line drilling, requiring considerable close speed drill holes; cushion blasting, where perimeter holes are charged but decoupled; smooth wall blasting, using decoupled spaced charges for perimeter control in development/tunnel headings; and presplitting involving creating a shear zone prior to any production blasting.

WESTERN AUSTRALIAN STATUTORY AUTHORITIES FOR EXPLOSIVES

The Chief Inspector of Explosives, Department of Mines, administer the Explosives and Dangerous Goods Act 1961-1967 and Regulations through which is controlled the importation, manufacture, conveyance, storage, sale and, at places other than mine sites, the use of explosives throughout Western Australia.

Explosives may be purchased only by a person authorized to possess explosives viz:

(i) the Owner of a mine or authorised agent of the Owner;
(ii) the holder of a Magazine Licence;
(iii) the holder of a Shot firer’s Permit;
(iv) the holder of a Permit to Purchase Explosives.

The State Mining Engineer, Department of Mines, administers the Mines Regulation Act 1946 - 1974 and Regulations through which is controlled the use of explosives in mines and quarries in Western Australia.

MINES REGULATIONS

Part 7 - Explosives and Blasting Agents - of the Mines Regulation Act and Regulations states the most important statutory requirement with which personnel associated with mining should be conversant. Coverage of the most significant aspects is presented under a number of group headings. References listed are the relevant sections of the Regulations.

Preliminary coverage of regulation requirement concentrates on general aspects and underground mining requirements.
BLASTING AGENTS
These are defined as material whose ingredients are not classified as explosives and when prepared cannot be detonated when tested in a manner laid down by the Chief inspector of Explosives. (ref. 7.1). Blasting agents are to be stored, transported, handled and used as an explosive. (ref. 7.3) Where a blasting agent is used on a mine site, it must be manufactured under licence from the Chief Inspector of Explosives. (ref 7.2 & 7.38 (a)). Blasting agents are not to be mixed underground. (ref. 7.38 (b)).

MAGAZINES AND STORAGE
All explosives and blasting agents are to be stored in a main magazine or if at the work place, in a working party's magazine. (ref. 7.4).

Surface magazines which stock more than 250 kg of explosive or blasting agent have to be constructed in accordance with the Explosives and Dangerous Goods Act, 1961 and are to be licensed by the Chief Inspector of Explosives. (ref. 7.5 (i)). Magazines storing 250 kg or less of explosives or blasting agent need only be of a construction satisfactory to an Inspector. (ref. 7.5 (2)). Each main magazine is to be under the control of an appointed person who is responsible for the surface storage and security of the stored explosive material. (ref. 7.7).

DETONATOR MAGAZINES
Detonators are to be stored in a separate magazine where practicable, with no detonators or detonator accessories being stored in a main magazine unless they are contained and separated and stored so they will not detonate other explosive or blasting agents. (ref. 7.19).

Detonator (plain) capping stations for the capping of safety fuse to form rods is to be done above ground and not in a magazine. Only approved crimping appliances to be used. (ref. 7.10).

UNDERGROUND MAGAZINES
Main underground magazines are to be situated in upper level of mine and must be connected to the surface by an independent air pass. The entry and distance of the magazine chamber should be greater than 20 metres and include a 90 degree angle. Quantity stored is to be one week's supply for the mine with additional carryover of up to 500 kg. Approved ventilation and security must be provided. (ref. 7.6).

Working party's magazines underground are to be clear of travelling ways, and all explosive, blasting agents, detonating fuse and detonators are to be kept in separate containers. The quantity permitted is for two normal day's work. (ref. 7.10).

DETERIORATED EXPLOSIVE
Old and deteriorated explosives are to be removed from magazines and destroyed. (ref. 7.12). No explosive, blasting agent, detonating cord or detonators are to be disposed of unless by an approved method. (ref. 7.13).

EXPLOSIVE USERS
A person is not to charge or fire an explosive or blasting agent in a mine unless he has satisfied the mine manager, foreman or supervisor by a practical test that he is competent to do so. The minimum age for persons to handle, charge or fire explosives and blasting agents is 18 years. (ref. 7.15).
HANDLING OF EXPLOSIVE
Explosives and blasting agents are to be stored and transported in approved containers. Separate containers are to be provided for explosive, blasting agent, fuses and detonating accessories. Rods are to be stored separately from explosives. (ref. 7.17).

SAFETY FUSE
Safety fuse must pass a burning rate test and exhibit burning rates of between 90 and 110 seconds per lineal metre. (ref. 7.22). The minimum length of safety fuse to be used for firing any charge is one metre, or long enough to allow the person firing the charge to retreat to safety.

A single safety fuse may be lit by a match; however for more than one fuse an approved lighter is required. A maximum of four fuses may be lit by hand. Multiple igniting cartridge ignitor cord or another approved method may be used. (ref. 7.34).
CHAPTER 7 PRACTICAL USE OF EXPLOSIVES

DETONATING CORD
Detonating cord is classified as an explosive and is to be handled and kept as an explosive. Detonators and detonating cord are to be kept separate in storage, and in transport.

DRILLING PRECAUTIONS
Drilling should not be carried out in any face or bench underground that has not been washed down and butts washed clean and examined for misfires. No drill hole is to be collared in a butt hole. (ref. 7.24).

CHARGING OPERATIONS
Holes prior to charging are to be blown out - or cleaned. Tamping rods used are to be either wood or of an approved non-metallic material. Nitroglycerine explosives are to be charged into a hole only in cartridge form; the hole diameter should not impede placement of the cartridge. (ref. 7.26).

Where ammonium nitrate blasting agent is charged by pneumatic means, then the loader and charging hose are to be suitably earthed. The loading hose of pneumatic loading devices is to be of approved semi-conductive characteristics. Protective type electric detonators are to be used. (ref. 7.39)

FIRING TIMES
Firing underground is to be done during the period of fifteen minutes before the recognised crib time and/or the end of the shift. No firing is to be done unless workmen have been removed to the ventilation intake side of places where firing is to be conducted. Provided consent is given by the Assistant Underground Manager, Foreman or Shift Boss, firing may be undertaken at other times for the following: freeing obstructions from chutes, ore passes or mill holes, and making working places safe, or firing misfires.

Shaft sinking blasting may be conducted at non-prescribed times provided it is safe to do so. Intention of unusually large blasting underground is to be communicated to the Senior Inspector, consent in writing being required where non-prescribed blasting time is contemplated. (ref. 7.30)

WARNINGS OF FIRING
For underground blasting a person intending to fire a charge must give warning to persons in adjacent workings prior to firing; entry to blast site is guarded or firing notices displayed. (ref. 7.28).

AFTER BLASTING
After a blast a person shall not recommence work in the work place until an inspection has been made for misfires by the shot firer or a competent person. (ref. 7.40).

ELECTRIC FIRING
Where electric firing is conducted in a mine only competent persons who have been instructed in such work, and authorised by the Manager in writing shall be permitted to charge or fire these shots. A record of authorized persons will be maintained on a register.

Electrical firing equipment such as circuit testers, exploders, switches, and other approaches are to be kept in good working order. Circuit testing devices for continuity testing or resistance testing must be of approved design. The locating of testing and circuit tester output is specified.
Electric detonators taken underground are to be of an approved type and the lead wires are to be short circuited. Firing cables are to be protected and short circuited when not in use. (ref. 7.35).

**ELECTRIC EXPLODERS**
An electric exploder is permitted to be used for firing single electric detonators, electric detonators in series, but not series/parallel circuits unless authorized by the Manager. The exploder should have adequate capacity, and be in charge of the shot firer who has the firing key or handle in his possession. (ref. 7.37).

**MISFIRES**
Where a misfire has occurred or is assumed to have occurred or where a hole has damaged safety fuse, detonating cord or detonator lead wires exposed to it shall be treated as a misfire. This includes cut-off holes, and butts unless shown not to contain explosive or blasting agent. (ref. 7.41)

When a misfire has occurred or is suspected to have occurred underground, the miner or shot firer should report the misfire to the Underground Manager, Foreman or Shift Boss, or if at the change of shift, to the oncoming shift.

Drilling is prohibited in the face until the misfire has been made safe or refired. (ref. 7.42).

The re-approach time for a charge which has misfired is stated as: for safety fuse - an elapse of half an hour from lighting the fuse; for electric firing - a time of five minutes from removing the electric power source and short circuiting the firing cable. (ref. 7.44). A record of misfires is to be maintained in a record book.

**Misfires – Remedial Action**
Nitroglycerine explosives which have misfired are to be reprimed and fired. Blasting agent in a misfired state may be washed out by water and if a primer remains it is to be refired, (ref. 7.45).

Misfire containing blasting agent and safety fuse which has been reprimed and fired shall not be approached until one hour after firing.

**SURFACE OPERATIONS AND QUARRIES**

**WORKING PARTY MAGAZINES**
For quarrying operations the quantity of explosive, blasting agent, detonating cord and detonators to be stored in a working party’s magazine should not exceed one day’s work requirement. The type and location of such magazines and containers are to be approved by the Inspector. (ref. 7.11).

**EXPLOSIVE HANDLING**
The quantity of explosive or blasting agent taken into a quarry face shall not exceed that amount estimated for immediate use. (ref. 7.17(2)b).

**DRILLING PRECAUTIONS**
Drilling is not to be carried out in a quarry face or bench until it has been examined for misfires. Except for clearing a misfire after repriming and refiring, drill holes should never be drilled within 6 metres of a hole containing explosive or blasting agent unless otherwise authorized. (ref. 7.25).
CHAPTER 7 PRACTICAL USE OF EXPLOSIVES

7.20

CHARGING OPERATIONS
Nitroglycerine explosives shall be charged into holes in complete cartridges. In down holes, explosives in cartridge form are to be lowered into position, not dropped. The minimum distance from a hole being charged for operational equipment is six metres excluding charging or loading vehicles e.g. mixer truck. (ref. 7.27).

For charging purposes, where other methods are not practicable, ammonium nitrate blasting agent may be poured into a hole. (ref. 7.39 (2)).

FIRING TIMES
Blasting in quarries is restricted to between 7.00 a.m. and 6.00 p.m. unless written permission from the Inspector is obtained. In built up areas blasting times may be specified by the Inspector. Blasting to remove obstructions from crushers as refiring misfires may be carried out in consent with the quarry manager at other than the prescribed times. However at all times workman and public safety must be maintained. (ref. 7.31).

FIRING WARNINGS
Firing of an explosive charge or blasting agent requires that proper warning be given and all entry points be guarded or firing warning notices erected. Where blasting constitutes a public nuisance or danger, audible warnings of blasting may be required. (ref. 7.29).

FLY ROCK
Where debris or fly rock from surface blasting could endanger any person or property the manager has to ensure that precautions are taken to minimize such dangers, (ref. 7.33).

MISFIRES
Where a misfire has occurred or is suspected to have occurred in a quarry or surface mining operation no work shall be done until it has been inspected and corrective action taken. (ref. 7.43).

When repriming and refiring of a misfire has failed to explode the explosive or blasting agent the rock around the misfire is to be cleared off and the hole in question located. The remaining portion of the hole shall be removed either by digging it out or by drilling and firing one or more holes adjacent to it.

BULLED OR CHAMBERED HOLES
A hole shall not be recharged after bulling unless one hour has elapsed since firing, or water has been used to cool the hole.
### Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>BACK- BREAK</strong></td>
<td>Ground broken beyond the last or outer row of holes.</td>
</tr>
<tr>
<td><strong>BULLED HOLE</strong></td>
<td>A shothole which has been bulled or chambered by exploding a light charge of high explosive in the bottom; a hole with a chamber at the bottom to accommodate a large quantity of explosive.</td>
</tr>
<tr>
<td><strong>BUTT</strong></td>
<td>The remaining part of a hole after the charge has been fired; a hole which was not fully blown out. As it is likely to contain an unexploded charge it is dangerous to drill in a butt.</td>
</tr>
<tr>
<td><strong>COLLAR PRIMING</strong></td>
<td>A method in which the primer is placed near the top, or at the collar end of the charge.</td>
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<tr>
<td><strong>METALLIZED</strong></td>
<td>In respect of blasting agents, sensitized or boosted with metal powder or granules (usually aluminium or ferro-silicon) to yield more energy.</td>
</tr>
<tr>
<td><strong>MISFIRE</strong></td>
<td>A charge or part of a charge, which for one of any number of reasons has not exploded. They are usually difficult and dangerous to resolve. Misfires must be treated with respect.</td>
</tr>
<tr>
<td><strong>MUD CAPPING</strong></td>
<td>A method of breaking a boulder with explosive without drilling a hole. The charge is set in a depression on the boulder and covered with a thick layer of wet clay to confine it (plastering).</td>
</tr>
<tr>
<td><strong>PRIMER</strong></td>
<td>A cartridge of HE (or a prefabricated precast booster) incorporating the detonating device. This is the key element of a charge of explosives.</td>
</tr>
<tr>
<td><strong>PRILLS</strong></td>
<td>Cellular sub-globular particles of AN formed by spraying concentrated AN solution against a stream of air.</td>
</tr>
<tr>
<td><strong>TAMPING</strong></td>
<td>The act of charging or tamping a charge into a hole, with the aid of a tamping stick.</td>
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</table>
SURFACE MINING

A surface mine is an open-air excavation for removing minerals. Surface mining can be employed to extract metallic or non-metallic minerals from any near-surface deposit of any rock type. Near surface generally implies depths less than 150 metres. The size of the deposit may range from a few tonnes to 100 million tonnes. The larger deposits are usually of considerable area. Surface mining methods may be tabulated into three basic types: placer, open pit and glory hole, or combinations of these.

A. Placer Mining
   (a) Planning and Sluicing.
   (b) Hydraulicking.
   (c) Dredging.

B. Open Pit
   (a) Single-bench.
   (b) Multiple-bench.
   (c) Strip mining.
   (d) Quarry mining.

C. Glory Hole

Surface mining offers a wide flexibility in production, which includes the ability to mine selectively and the potential for 100% extraction of ore within the pit limits. Fewer men are required, because mechanization permits high unit production and greater safety than underground mining.

Problems in surface mining include adverse weather in some locations, and undesirable environmental problems, such as surface scarring, dust, noise and blasting vibrations as well as waste disposal.

FACTORS AFFECTING SELECTION OF A MINING METHOD

CLASSIFICATION OF MINERAL DEPOSITS AND ROCK MINERALS
After a mineral deposit has been discovered, delineated and evaluated, the next step is the selection of a mining method that is physically, economically and environmentally adaptable to recovering the mineral from the deposit. The term "mineral deposit" is used to denote a concentration of mineral (including rock and coal). Many of the critical design criteria must be determined using information obtained from an examination of exploratory drill core specimens.

The major factors influencing the choice and characteristics of surface mining are:

1. the thickness of overburden and physical proper-ties of capping and enclosing country rock;
2. the thickness, shape, configuration and structure of the mineral deposit;
3. its mode of occurrence (position with regard to the ground surface, angle of dip);
4. hydro-geological conditions of mining (these should be known in order to:
• choose a rational method of dewatering the deposit;
• estimate its cost;
• select the correct high wall angles;
• work out measures aimed at preventing landslides and water in-rushes,

5. feasible technical facilities for surface mining work (i.e. the types of energy and equipment, mainly drilling, loading and transport equipment);

6. climatic conditions prevalent in the area of mining operations. (These exert a definite effect in working soft ground which requires no drilling and blasting);

7. environmental factors, such as the preservation of the surface overlying the mine, and the prevention of air and water pollutants.

Of these factors, the physical characteristics of the deposit and properties of the mineral and surrounding rock limit the methods that can be employed to mine it.

**TYPES OF MINERAL DEPOSITS**

Some six types of mineral deposit are recognised from their physical characteristics.

1. **MASSIVE**
   A deposit of considerable lateral and vertical extent in which mineralization (ore) is relatively uniformly distributed, e.g. disseminated copper (porphyry copper deposits).

2. **BEDDED OR TABULAR**
   A mineral deposit that parallels the stratification, most often sedimentary rock, usually laterally extensive, and of limited thickness. Most coal measures and some evaporated mineral deposits e.g. potash, fall into this class.

3. **NARROW VEIN**
   A zone or belt of mineralization (ore) typically long, narrow (less than 3 metres), often dipping steeply, and usually lying within boundaries separating it from neighbouring rock. Gold and other metallic minerals occur in narrow veins.

4. **WIDE VEINS**
   Veins wider than 3 metres but with similar characteristics to narrow vein.

5. **LENTICULAR OR POCKET**
   Isolated ore body or an enrichment of limited vertical and horizontal extent in a massive, bedded or vein deposit. Lead, zinc and iron ore often occur in this manner.

6. **PLACER**
   A surface or near surface deposit, usually tabular and of considerable extent, containing mineral particles especially heavy minerals, e.g. gold, platinum in detritus material e.g. sand and gravel.

   Associated with mineral deposits are a variety of different rock types (materials). These may be examined under headings as follows:

**A. Detritus**
   Sediments and other accumulations of solid particles, produced by mechanical and/or chemical disintegration of rocks, which are often relatively unconsolidated, i.e. having low cohesive strength and low compressive (bearing) strength, e.g. soil, sand, gravel, alluvium.
B. Jointed and Fractured - (Unbonded)
A rock mass which occurs at the surface or near a fault zone in which the joints or fractures contain weathered material e.g. altered or decomposed products such as clays.

C. Jointed and Fractured with Partial Bonding
Here fractured material is partially re-cemented.

D. Laminar: Thin Bedded
A rock mass generally of sedimentary origin with lamina thickness of up to 1/3 of a metre. Partings between the lamina are partially bonded, e.g. bedded shales and sandstones associated with coal seams and basins.

E. Laminar: Thick Bedded
A rock mass with laminar thickness greater than 1/3 of a metre.

F. Massive
Rocks which are relatively unjointed and fractured in mass or where fracture or jointing occurs, but are bonded by recementation. Breccias and conglomerates may fall into this group. Overall the rock mass exhibits a strong cohesive strength.

SURFACE MINING METHODS

PLACER MINING
Placer mining is affected by the concentration of minerals from detrital materials by selective settling in running water. A prime requirement is that the material be in or near water and on or near the ground surface.

PANNING AND SLUICING
Panning was used in traditional prospecting of gold mining in placer deposits where water was plentiful and ore, e.g. gold, silver precious stones, was concentrated in layers or pockets. Panning can only be used when the ore or valuable mineral is heavier than the gangue (waste rock), and so production is very limited. Panning is useful as a sampling method, and is used for prospecting/exploration purposes in tracing placer deposits to the vein or ore source (loaming).

Sluicing used in early gold production days has been replaced largely by higher production methods.

Water and a trough-shaped box (sluice box) are used to separate the ore and waste. Ground slope is necessary for the water to carry material through the sluice box.

HYDRAULIC MINING
This involves larger placer deposits that generally contain gravel and boulders. Large quantities of water under pressure are directed through pipes and nozzles (giants) to disintegrate the deposit. The system may involve a ground sluice where face material is washed through the sluice box. Alternatively, the sand, gravel and valuable mineral are picked up by a gravel pump and pumped to a sluice or separating plant. Handling of solids in suspension in pipes is referred to as hydraulic (pipeline) transport.

Face heights handled may vary from 5 to 20 metres though heights up to 50 metres have been handled using remote controlled monitors. Bedrock slope needs to be greater than 2% and up to 5% for coarse material. Production is limited by availability of water including adequate head (pressure), deposit thickness, boulder size, and bed-rock slope. (See Fig.8.1). Ground sluicing is shown in (a) pumping to elevated sluices in (b) and hydraulic monitor or giant in (c).
DREDGING
Dredging is the underwater excavation of a placer deposit of detritus type rock material. For this approach the deposit is usually low grade, large in area and in thickness. Dredging may be in old river beds or in active river courses. Dredging may also be done off-shore under suitable conditions.

Dredging recovery is high where the bedrock is hard and flat and bottom loss is minimal.

Dredges used are fundamentally of two types:

A. Bucket ladder type which consists of a ladder like truss to which is attached an endless chain with buckets. Basically a continuous large-volume digging machine usually incorporating gravity concentration facilities eg. jigs, a tail stacker that allows waste discharge, the dredge is a floating plant mounted on a barge-like pontoon. (See Fig. 8.2).

B. Suction cutter dredges are basically floating pontoons with a pump mounted on board which excavates the material by suction and transports its to a store-based or floating concentrating plant. This suction pipe may be equipped with a cutter head for improving excavation of material. Beach sand deposits of rutile, ilmenite and zircon are frequently handled by this method.

(a) Gravity Sluicing
1. Waste material
2. Sluices
3. Alluvial (placer) deposit
4. Sluice channel

(b) Hydraulic elevator
5. Pipe line from gravel pump
Figure 8.1 Hydraulic mining of placers (Alluvials)

Figure 8.2 Bucket ladder dredge (California type)
### Table 8.1 Factors influencing selection of mining method

<table>
<thead>
<tr>
<th>MINING METHOD</th>
<th>BUCKET DREDGING</th>
<th>SUCTION DREDGING</th>
<th>DRAG LINE DREDGING</th>
<th>SLUICING</th>
<th>DRY MINING</th>
</tr>
</thead>
<tbody>
<tr>
<td>Minimum m³ to</td>
<td>20 000 000</td>
<td>250 000</td>
<td>1 000 000</td>
<td>100 000</td>
<td>1 000 000</td>
</tr>
<tr>
<td>be excavated</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>in average</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>production</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Nature of wash</td>
<td>Reasonably free with absence of hard pinnacles and bars</td>
<td>Unconsolidated with short size range</td>
<td>As for bucket dredging</td>
<td>As for bucket dredging</td>
<td>Capable of being broken up with hydraulic jets</td>
</tr>
<tr>
<td>Nature of over-</td>
<td>Relatively free with absence of large boulders</td>
<td>Unconsolidated</td>
<td>As for bucket dredging</td>
<td>As for bucket dredging</td>
<td>Capable of being broken up with hydraulic jets</td>
</tr>
<tr>
<td>burden</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Type of mineral</td>
<td>All types</td>
<td>Lighter heavies such as rutile, zircon</td>
<td>As for bucket dredging</td>
<td>As for bucket dredging</td>
<td>All types</td>
</tr>
<tr>
<td>Wetness of ground</td>
<td>Wet or dry, providing a dredge be established and maintained</td>
<td>All types</td>
<td>All types</td>
<td>As for bucket dredging</td>
<td>All types</td>
</tr>
<tr>
<td>Water require-</td>
<td>Large</td>
<td>Large</td>
<td>Large</td>
<td>Very large</td>
<td>Any degree of slope</td>
</tr>
<tr>
<td>ments</td>
<td>Medium</td>
<td>Medium</td>
<td>Medium</td>
<td>Very large</td>
<td>Any degree of slope</td>
</tr>
<tr>
<td>Slope of bottom</td>
<td>Relatively flat</td>
<td>Relatively flat</td>
<td>Relatively flat</td>
<td>Any degree of slope</td>
<td>Any degree of slope</td>
</tr>
</tbody>
</table>

The bucket ladder dredge can be used in water depth of 4 to 30 metres and has been used for tin mining to depths of 48 metres.

Suction dredges operate in pond depths of up to 9 metres; at greater depths, suction pump lift must be assisted by jetting the intake.

Problems associated with dredging are, having sufficient water in the pond to float the dredge and having sufficient clear water to beneficiate the material being dug. Because dredging is usually on a large scale, tailings disposal and restoration of the land and water are of major concern. Table 8.1 indicates factors involved in method selection for mining placer deposits.
OPEN PIT MINING

Open pit mining, also referred to as open cut mining, is employed to exploit mineral deposits in any rock type lying on or near the surface. The (pit) excavation is open to the sky and the weather. These methods are best suited to ore bodies of substantial horizontal dimensions which permit relatively high rates of production at low cost.

Although stripping and quarrying are forms of open pit mining, strip mining usually refers to the mining of coal (also opencast) and quarry mining is related to the production of non-metallic materials such as dimension stone, rock aggregates etc.

Factors that determine the pit layout include orientation of the deposit, stripping ratio, weather conditions, rate of production required and the equipment available. At some depth for every type of deposit there is likely to be a choice usually based on economic factors of mining either by surface or underground methods. Some operations involve both methods or transfer from one to the other and a certain depth is attained.

**CHOICE OF METHOD**

In selecting the method of mining it is necessary to compare the economic efficiencies of open and underground work, except when the advantages of one of the methods are quite obvious. The basic characteristic used in the economic evaluation of open pit mining is the stripping ratio, which generally means the volume of excavated ground per unit of the ore or mineral.

![Diagram](image)

1. overburden cover
2. waste (country rock)
3. ore body.

*Figure 8.3 Open pit section – overburden to ore relationship*

In open pit operation, mining includes the cost of removing the waste overburden and the waste in the slopes of the pit. (See Fig. 8.3).

The ratio of waste to ore is therefore the controlling factor in comparative cost of mining an ore body by open pit versus underground methods. This involves projected underground
mining costs and open pit mining costs for ore and waste: a ratio is then derived which should not be exceeded by surface mining.

Actual inpit design requires the establishment of a break-even stripping ratio which usually utilizes the re-coverable value of ore, production cost and the stripping of waste: a minimum profit is often incorporated. It is important to recognize that stripping ratios are not the overall average ratio but those of the pit sections. (See Fig. 8.3).

If each section was to match the break-even stripping ratio without the provision of a minimum profit then the operation would produce no profit margin.

**FORMULA BREAK-EVEN STRIP RATIO**

 Recoverable-even stripping ratio = \[
\frac{\text{Recoverable value/tonne ore} - \text{production cost/tonne ore} - \text{stripping cost/tonne waste}}{\text{stripping cost/tonne waste}}
\]

**EXAMPLE**

 Recoverable value per tonne of ore: $6.80
 Production cost per tonne of ore: $4.30
 Stripping cost per tonne of waste: $0.65

 Breakeven strip ratio = \[
\frac{6.80 - 4.30}{0.65} = 3.85:1
\]

Stripping ratios vary from low ratios (1:l) under adverse conditions to high ratios (20:l) in particularly favourable conditions. For hard rock, the ratio tends to be low and for soft rocks, high, especially where direct overcasting of waste can be done by the excavator. The lower stripping ratios for metalliferous ore deposits by comparison with the mining of coal beds are due to the difference in density of coal and ore, as well as the more difficult conditions in the exploitation of hard rock deposits (drilling, blasting and transportation of waste over considerable distances). The nature and extent of mechanization has a considerable bearing on the break-even stripping ratio.

Four types of open pit mining are readily identified:

1. single-bench;
2. multiple-bench;
3. strip;
4. quarry.

**SINGLE BENCH**

In an open pit mine a bench forms a single level of operation above which the material (mineral and waste) is excavated from the bench face. (See Figs 8.4 and 8.5).

Single-bench open-pit mining can be employed to mine any comparatively shallow mineral deposit in any rock type: thus quarry and strip mining may be single-bench operations.

The maximum stable bench height and bench slope depend upon the rock type that forms the bench. Bench heights are specified by Mines Regulation (17.13)(i) as not to exceed 20 metres unless a higher face is allowed by the District Mines Inspector. For sand pits the maximum height is either the vertical reach of the excavator at the working face or 10
metres whichever is less. In some instances, bench face heights up to 60 metres have been used. Obviously face stability is critical otherwise excavation equipment is likely to be in danger should the face collapse.

Typical single-bench operations include deposits of sand, gravel, coal seams with limited overburden (usually referred to as strip mining) and near surface exposures of dimension stone and aggregate stone (quarry mining).

Production by single-bench operations is limited only by the capacity of the equipment that can be employed in the pits and by the number of areas along the face that can be excavated simultaneously.

![Figure 8.4 Single bench open pit mine](image1)

![Figure 8.5 Single-bench in flat surface deposit using](image2)
8.10

Figure 8.6 Multiple-bench open-pit mine

1. Inclined ramps.
2. Bench faces

Figure 8.7 Bench access (Spiral route layout)
MULTIPLE-BENCH

Multiple-benches can normally be employed in any massive thick-bedded, wide-vein or tabular deposit that lies at or extends to a depth greater than that suitable for single-bench mining. The rock material may be of any type strong enough to permit the development of benches of economic height and may consist of unconsolidated material to hard rock material. (See Fig. 8.6).

Where pit depth is in excess of 8 to 15 metres, more than one bench is usually necessary.

With more than one bench, the bench width will vary according to the size of the excavation (loading) and haulage equipment as well as the rock material in the bench face. Widths may be from 6 to 20 metres. Benches are normally used as roadways, either forming a spiral to the bottom of the pit or with ramps between horizontal benches. Bench widths or berms are also designed to provide protection for men and materials from small slope failures (slides). (See Fig. 8.7).

Bench slopes are usually steeper than the pit slope, because rock can maintain a nearly vertical wall for short heights. (See Figs 8.8 and 8.9).

Pit slopes vary from 20' to 70' from the horizontal. During the final mining cycle before abandonment the pit slope may be steepened to increase recovery.

Environmental aspects related to multiple bench mining are those of waste disposal, noise, dust, blasting vibrations and land restoration. Water may or may not be an environmental problem.
**STRIP MINING**

Strip mining is the term applied mainly to the mining of near surface coal seams. However other mineral deposits with low cohesive strengths can also be mined by this method. Most stripping involves bedded sedimentary formations; blasting may or may not be required depending upon the type of overburden.

Coal seams mined by stripping range from approximately one metre to 10 metres or more in thickness: thicker and multiple seam deposits are usually mined by benching. With current equipment, overburden to coal ratios of as high as 30:1 and with depths up to 50 metres, have been stripped where overburden conditions have been favourable. High stripping ratios depend partly on mining cost and equipment efficiencies which vary widely between operations and in most instances substantially control minable coal and over-burden stripping ratios.

Strip mining is usually accomplished by removing the overburden and coal from a strip across one dimension of the deposit. A parallel strip is then excavated in the opposite direction and the overburden or waste rock is placed into the strip previously mined. The cycle is repeated. (See Figs 8.10 and 8.11).

Stripping equipment is usually large and may discharge the waste material direct to the previously mined strip (if done by dragline this is termed casting) Draglines used for this duty have bucket capacities from 5 to 200 cubic metres. Bucket wheel excavators are also used, and stripping shovels have been used in the U.S.A. but are not common elsewhere. Mining of the coal seam is accomplished using shovel and truck combinations.

In strip mining maintenance of the sidewall is not as critical as for multiple-bench pit operations; however high waste piles do present slope failure problems.

---

**Figure 8.10 Strip mine**
Figure 8.11 Overcasting (strip mining) using direct casting dragline

**QUARRY MINING**

Quarrying is the term used to describe the surface mining of rock, such as marble, granite, limestone, shale etc, that are valuable for either their mechanical or chemical properties. In this type of mining, the deposit usually is either massive, bedded or penticular, and is suitable for bench mining. Most quarries are in sedimentary rock (limestone). However metamorphic rock (marble) and igneous (granite) rocks are mined.

There are two basic types of quarry: dimension stone and broken stone, i.e. aggregate and chemical limestone.

Dimension stone quarries normally have benches with vertical faces and the overall pit slope is steep. (See Fig. 8.12).

Figure 8.12 Dimension stone quarry (slate, diorite, marble)
The rock, therefore, must have bonding across any fracture or joint planes, to produce a relatively high cohesive strength. The stone is generally broken loose by some manner of cutting instead of blasting. This is done to preserve its shape and strength. Bench heights up to 60 metres are used where cut blocks are lifted from the work face. Production is normally very selective and in limited quantities.

Quarry mining of aggregate or chemical stone is usually done by blasting to fragment the rock: the degree of fragmentation depends upon the product desired. Rock strengths vary from low to high according to the rock type and condition and production is less selective and at higher rates than for dimension stone quarries. Overburden removal is generally necessary. (See Fig. 8.13).

**GLORY HOLE**

Glory hole mining implies an open excavation from which ore is removed by gravity through an ore pass connecting to underground haulageways. (See Figs 8.14 and 8.15).

A glory hole is usually described as an operation where ore around the ore pass is blasted so that it falls into the pass by gravity, resulting in a funnel-shaped configuration. Modern open pits may haul to an ore pass connected to skip loading facilities (shaft hoisting) at or below the pit limits. These methods are however strictly not glory holing but possess similarity in utilizing underground transport methods. (See Figs 8.16 and 8.17).

The glory hole method may be used to mine almost any type of deposit, though rock material should not have a tendency to pack in drawpoints.
Figure 8.14 Glory hole mining

Figure 8.15 Opening up bottom horizon of open pit by underground workings

Figure 8.16 Open pit adit haulage
### Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>ALLUVIAL DEPOSIT OR PLACER DEPOSIT</strong></td>
<td>Earth, sand, gravel, or other rock or mineral materials transported and laid down by flowing water.</td>
</tr>
<tr>
<td><strong>ANGLE OF REPOSE</strong></td>
<td>The maximum slope at which a heap of any loose or fragmented solid material will stand without sliding or come to rest when poured or dumped in a pile or on a slope.</td>
</tr>
<tr>
<td><strong>BANK; BENCH FACE</strong></td>
<td>Specifically, usually a steep sloping mass of any earthy or rock material rising above the digging level from which the soil or rock is to be dug from its natural or blasted position in an open pit mine or quarry.</td>
</tr>
<tr>
<td><strong>BANK HEIGHT; BENCH HEIGHT; DIGGING HEIGHT</strong></td>
<td>The vertical height of a bank measured between its highest point or crest and its toe at the digging level or bench.</td>
</tr>
<tr>
<td><strong>BANK SLOPE</strong></td>
<td>The angle, measured in degrees of deviation from the horizontal, at which the cork or rock material will stand in an excavated, terrace-like cut in an open-pit mine or quarry.</td>
</tr>
<tr>
<td><strong>BENCH</strong></td>
<td>A ledge, which, in open-pit mines and quarries, forms a single level of operation above which mineral or waste materials are excavated from a continuous bank or bench face. The material or waste is removed in successive layers, each of which is a bench, several of which may be in operation simultaneously in different parts of, and at different elevations in an open pit mine or quarry.</td>
</tr>
<tr>
<td><strong>BERM</strong></td>
<td>A horizontal shelf or ledge built in to an embankment or sloping wall of an open pit or quarry to break the continuity of an otherwise long slope for the purpose of strengthening and increasing the stability of the slope or to catch or arrest slope slough material. A berm may also be used as a haulage road or serve as a bench above which material is excavated from a bank or bench face.</td>
</tr>
<tr>
<td>Term</td>
<td>Definition</td>
</tr>
<tr>
<td>------------------</td>
<td>--------------------------------------------------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>BURDEN</td>
<td>1. The distance between the explosive charge and the free face of the material to be blasted (burden distance).</td>
</tr>
<tr>
<td></td>
<td>2. Barren or non-ore material that overlies and must be removed to gain access to minable grade material. Frequently called over-burden or cover.</td>
</tr>
<tr>
<td>DROP CUT</td>
<td>The initial cut made in the floor of an open cut pit or quarry for the purpose of developing a bench at a level below the floor.</td>
</tr>
<tr>
<td>GLORY HOLE</td>
<td>A pit from which or in which material is dumped and fed by gravity to an underground haulage system.</td>
</tr>
<tr>
<td>GROUND SLUICING</td>
<td>To strip ground down slope by means of a directed stream of water to placer material and transport it to a riffled trough in which valuable mineral is recovered.</td>
</tr>
<tr>
<td>HAUL ROAD</td>
<td>A road built to carry heavily loaded trucks at a good speed. The grade is limited on this type of road and usually kept to less than 17 percent of climb in direction of load movement.</td>
</tr>
<tr>
<td>HIGH WALL</td>
<td>The unexcavated face of exposed over-burden and coal or ore in an opencast mine or the face or bank on the uphill side of a contour strip mine excavation.</td>
</tr>
<tr>
<td>HYDRAULIC MONITOR; GIANT MONITOR</td>
<td>A device for directing a high pressure jet of water in hydraulicking. It is essentially a swivel mounted, counter-weighted nozzle attached to a tripod or other type of stand and so designed that one man can easily control and direct the vertical and lateral movements of the nozzle.</td>
</tr>
<tr>
<td>HYDRAULICKING</td>
<td>Excavating alluvial or other mineral deposits by means of high-pressure water jets.</td>
</tr>
<tr>
<td>OPEN PIT MINE; OPENCAST MINE; OPEN CUT MINE; STRIP MINE</td>
<td>A mine working or excavating open to the surface.</td>
</tr>
<tr>
<td>PIT LIMITS</td>
<td>The vertical and lateral extent to which the mining of a mineral deposit by open pitting may be economically carried. Cost of removing overburden or waste materials versus the minable value of the ore so exposed is usually the factor controlling the limits of a pit.</td>
</tr>
<tr>
<td>PIT SLOPE</td>
<td>The angle at which the wall of an open pit or cut stands as measured along an imaginary plane extended along the crests of the berms or from the slope crest to its toe.</td>
</tr>
<tr>
<td>PLACER DEPOSIT</td>
<td>Unconsolidated deposits of detrital material containing valuable mineral.</td>
</tr>
<tr>
<td>QUARRY</td>
<td>An open or surface working usually for the extraction of building stone, slate, aggregate, limestone etc.</td>
</tr>
<tr>
<td>SLOPE</td>
<td>The inclined surface of a hill, or any part of the surface of the earth; the inclination of a roadway.</td>
</tr>
<tr>
<td>SLOPE STABILITY</td>
<td>The resistance of any inclined surface, e.g. the wall of an open pit or cut, to fracture by sliding or collapsing.</td>
</tr>
<tr>
<td>Term</td>
<td>Definition</td>
</tr>
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<td>----------------------</td>
<td>-------------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>SLUICE BOX</td>
<td>A long inclined trough or launder containing riffles in the bottom that provide a lodging place for heavy minerals in ore concentration. The material to be concentrated is carried down through the trough on a current of water. Sluice boxes are widely used in placer operations for separating minerals, such as gold, platinum, or other heavy minerals from alluviums.</td>
</tr>
<tr>
<td>SLUICING</td>
<td>A separation of minerals in a flowing stream of water.</td>
</tr>
<tr>
<td>SPOIL; WASTE</td>
<td>The overburden or non-ore material removed in gaining access to the ore or mineral material in surface mining.</td>
</tr>
<tr>
<td>STRIPPING</td>
<td>The removal of earth or non-ore rock materials, as required, to gain access to the ore or mineral materials worked; the process of removing overburden or waste material in a surface mining operation.</td>
</tr>
<tr>
<td>STRIPPING RATIO</td>
<td>The unit amount of spoil or waste that must be removed to gain access to a similar unit amount of ore or mineral material.</td>
</tr>
<tr>
<td>TOE</td>
<td>The base of a bank, bench or slope in a quarry or open-pit mine.</td>
</tr>
</tbody>
</table>
CHAPTER 9 – Mining Method – Underground Metalliferous

If the depth of an ore deposit is such that removal of the overburden makes surface mining unprofitable, underground methods need to be considered. The problem of recovering the mineral from such a deposit is reduced to selecting or developing a mining system that will exclude other options on a safety and profit basis and at the same time provide adequate ground support to protect working areas and if necessary to preserve the surface.

BASIC MINING CHARACTERISTICS

In establishing factors influencing the choice of mining method and the working of an ore deposit close attention to the physical characteristics of ore body, the ore and the enclosing country rock is necessary.

CLASSIFICATION OF ORE BODIES ACCORDING TO DIP ANGLE

The angle of dip is related to the method of moving the broken ore in the working openings (stopes).

<table>
<thead>
<tr>
<th>Angle of Dip</th>
<th>Classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>0°</td>
<td>Horizontal</td>
</tr>
<tr>
<td>0 – 3°</td>
<td>Conventional horizontal</td>
</tr>
<tr>
<td>3 – 30°</td>
<td>Gentle sloping</td>
</tr>
<tr>
<td>30 – 45°</td>
<td>Inclined</td>
</tr>
<tr>
<td>45 – 90°</td>
<td>Steep ore bodies</td>
</tr>
</tbody>
</table>

CLASSIFICATION OF ORE BODIES BY THICKNESS

<table>
<thead>
<tr>
<th>Thickness (Metres)</th>
<th>Classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 – 0.7</td>
<td>Very thin/narrow</td>
</tr>
<tr>
<td>0.7 – 2 m</td>
<td>Thin</td>
</tr>
<tr>
<td>2 – 5</td>
<td>Medium-thick</td>
</tr>
<tr>
<td>5 – 15/20</td>
<td>Thick</td>
</tr>
<tr>
<td>15/20 – over</td>
<td>Very thick/wide</td>
</tr>
</tbody>
</table>

The classification of ore bodies according to their thickness is based on the most frequently employed method of excavating development (preliminary access) openings, or in the direction of stoping. However it is generally taken to be the distance between the upper and lower surface of the ore deposit. (See Fig. 9.1).

The upper surface is referred to as the hanging wall side, the lower surface as the footwall side. In very thin ore bodies stoping and development headings require excavation of the wall rock (often referred to as ripping). In medium-thick and thick ore bodies development headings may be within the ore body. Often drives are on the footwall/country rock contact in order to delineate the ore body.
STRENGTH OF ORES AND HOST ROCKS
The hardness of ores and enclosing country rocks may be classified according to factors such as minability. Such simple groups consist of loose, soft, friable, strong and very strong. Actual rock compressive strengths can be used for a more comprehensive classification. A key feature of rock strength is the presence of in situ defects such as fractures and joints. Another classification known as rock quality designation (RQD) uses the fracture frequency in a drill core to assess the quality of the rock. The RQD is the percentage of core recovered in pieces longer than a fixed based length of core.

Figure 9.1 Ore body configurations
<table>
<thead>
<tr>
<th>Rqd Percent</th>
<th>Description of Rock Quality</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 – 25</td>
<td>Very poor</td>
</tr>
<tr>
<td>25 – 50</td>
<td>Poor</td>
</tr>
<tr>
<td>50 – 75</td>
<td>Fair</td>
</tr>
<tr>
<td>75 – 90</td>
<td>Good</td>
</tr>
<tr>
<td>90 – 100</td>
<td>Excellent</td>
</tr>
</tbody>
</table>

STABILITY OF ORE AND ENCLOSING COUNTRY ROCKS

Stability means the ability of rocks not to collapse when exposed from below and on the sides. The stability of rocks greatly influences the choice of a mining system: ores and enclosing country rock can be classified according to properties of stability.

1. Soft and very unstable ground not allowing even the slightest exposure in the roof and walls of a mine opening. Tunnelling shields are necessary, soft ground includes loose, free-running friable ground.

2. Unstable ground permitting very limited exposure of the roof and walls of an opening immediately adjoining the work face. Ground support is necessary immediately after excavation.

3. Medium-stability rocks allowing limited exposures of the roof and walls of an opening. Artificial support will not be required immediately but is necessary with time but not always over the entire exposure

4. Stable rocks permitting extensive exposures of the roof and particularly the walls; support required at individual positions or at regular intervals.

5. Very stable rocks allowing very large exposures of the roof (back) and walls without any artificial support.

Even though these classifications are not quantitative they are a practical approach to the choice of a mining method taking into account the stability of the ground.

BASIC MINING CYCLE

The basic mining cycle, both open pit and underground for conventional mining is the same: first establish access, then extract on a cyclic drill/blast/load operation, and then transport.
(Reproduced by kind permission of Atlas Copco Australia Pty Ltd, Perth)

Fig. 9.2 Mine layout and Access
In addition there is the necessary provision of auxiliary services including:

- water/pumping;
- ventilation;
- materials/personnel transportation;
- maintenance;
- control systems;

Mine access is provided from shafts, declines or adits depending upon the angle of inclination of the opening. Development work involves the driving of drives, cross cuts and other openings to delineate the ore body and to substantiate drill hole intersection results (ore reserves). Stope preparation is the preparing of stopes ready for extraction. (See Fig. 9.2).

**STOPING**

Stoping is the actual extraction operation. Stoping method and mining method can be treated roughly as synonymous terms.

There are three basic principles on which a method is selected.

1. Safety.
2. Efficiency (maximum extraction).
3. Economy (lowest cost/maximum profit).

Two and three are closely related and can be conflicting.

Factors affecting the choice of stoping method may be grouped as follows:

1. Spatial characteristics of the ore body:
   - Size, shape (width/thickness, vertical extent, regularity).
   - Attitude - dip, strike, pitch.
   - Depth.

2. Physical, chemical and mechanical properties of the ore and country rock.
   - Ore and wall rock strength - compressive, tensile, shear.
   - Planes of weakness - joints, bedding planes, shear, faults.
   - Liability of degradation, oxidation.
   - Mineralogy.

3. Groundwater and hydraulic conditions.
   - Hazard or nuisance water.

4. Economic factors.
   - Grade of ore, value of mineral, distribution of values.
   - Comparative mining and processing costs
   - Desired production rates
   - Geographical considerations.
   - Cost of labour and materials.
   - Effect of dilution and loss
   - Availability of labour and materials.
   - Infrastructure (essential services), housing

5. Environmental factors.
Geographical considerations.
Preservation of the surface.
Prevention of air and water Pollution
Influence of depth (problem of depth - pressure and rock stress).

The choices available when considering the factors are numerous; they range from each end of the spectrum:

- High production/low grade/low cost/dilution - loss/ low efficiency.
- Low production/high grade/high cost/clean/high efficiency.

The second choice falls within what is termed selective mining; however for conservation of resources efficient and effective extraction should be aimed at.

**MINING METHOD CLASSIFICATION**

Mining methods are classified according to rock strength (ground support method); however there are many variants.

**CONVENTIONS**

Mining may be referred to by the vertical direction of extraction.

Overhand stoping is that which proceeds upward; underhaul stoping that which proceeds downward such as by benching. Breast stoping is where mining proceeds at or near the horizontal.

Conventional mining is generally taken to involve non-mechanized drilling using air-leg, hand-held pneumatic rock drills. Installation of support timber etc is largely done by hand.

Mechanized mining is that which utilizes mechanized drilling rigs and direct load-out of broken material. Installation of rock support is handled largely by drill platform rigs. Broadly mechanized stopes enable wheeled transport entry to facilitate production such as mobile explosive (AN/FO blasting agent) pressure loaders.

Underground mining also falls within two broad groupings: entry mining and non-entry mining. The second refers to methods which remove material in either gaseous, liquid or solid form from beneath the earth’s crust without providing access or entry for personnel.

**SELF-SUPPORTED OPENINGS**

Self-supported openings are those in which the super-incumbent (overlying) load is carried by the sidewalls and pillars. The span (minimum wall to wall, wall to pillar or pillar to pillar dimension) that will stand unsupported may range from virtually zero for closely jointed or thinly laminated rock materials with little or no cohesive strength across joints or partings to more than 35 metres for massive ore bodies and rock.

1. **SELF-SUPPORTING OPENINGS (NATURAL)**

   A. Open Stope Mining
   - isolated openings;
   - sub-level stoping;
   - longhole stoping.

   B. Pillored Open Stopes
• random pillars;
• regular pillars.

2. **ARTIFICIALLY SUPPORTED STOPES (SUPPORTED OPENINGS)**
   A. Shrinkage Stoping (broken ore)
   B. Cut and Fill (waste filled)
   C. Stull Stoping
   D. Square-set Stoping
   E. Longwall Mining

3. **CAVING METHODS (STRESS RELIEF)**
   A. Caving (ore broken by induced collapse)
   • sub-level caving;
   • block caving.
   B. Top Slicing

**OPEN STOPE MINING**

An open stope is an underground opening from which ore is removed without use of material for wall or roof support.

**ISOLATED OPENINGS**

These are essentially unpillored - unsupported underground openings which have ore extracted from isolated pockets, lenses and shoots of ore. (See Fig. 9.3).

**SUB-LEVEL STOPING**

Sub-level stoping is generally employed in steeply dipping narrow, wide-vein and bedded deposits. The rock material in the hanging wall and footwall and the ore should be relatively competent.

Two basic configurations are possible: longitudinal and transverse. (See Fig. 9.4).

In both stope configurations ore is mined from sub-levels by benching and flows by gravity to the drawpoints. Longitudinal sub-level stoping is used for comparatively narrow, steeply dipping deposits. The stopes run parallel to the strike of the deposit. Stope widths up to 20 metres have been mined in this manner. Pillars required are rib and floor pillars and extraction is of the order of 75%. Dilution may occur from footwall or hanging wall, however caving of this nature is to be avoided.

For very thick deposits transverse sub-level stoping is used: the stopes are perpendicular to the strike of the deposit, but recovery is less due to separating rib pillars.
LONGHOLE STOPING
This is, fundamentally, an adoption of sub-level stoping often referred to as blast hole stoping. The drilling pattern used is a series of ring like fans. Heavy drifters drilling holes up to 50 metres are used and the method achieves continuous high production. (See Figs 9.5 and 9.6).

Figure 9.3 Isolated openings without pillars

Figure 9.4 Schematic sub level stoping

(Reproduced by kind permission of John Wiley and Sons Inc. N.S.W.)
Figure 9.5 Longhole sub-level stoping
PILLARED OPEN STOPES

Generally a mineral deposit of considerable extent such as a narrow or wide-vein deposit or a large pocket or lense of ore, cannot be mined as a single unsupported open stope. To maintain stability, support is required within the limits of the deposit, and if this support is affected by leaving areas of unexcavated ore or waste, the system of mining is referred to as pillared open stoping.

RANDOM PILLARS

This approach is used in mining:

- Pockets and lenses of ore where ore grade and thickness is variable. Pillars are left in lower grade sections where possible.
- Inclined deposits less than 45° which are relatively narrow.

The span permissible between random pillars varies with the quality of the roof rock. (See Fig. 9.7).

Extraction and recovery from random pillar open stopes is of the order of 60 to 80 percent.
Figure 9.7 Open stope with random pillars

Figure 9.8 Open stoping with regular pillars

Figure 9.9 Open stoping with regular pillars in an inclined deposit
OPEN STOPING WITH REGULAR PILLARS
Generally in bedded deposits in which grade and ore thickness are relatively uniform, regular pillar systems are used. (See Fig. 9.8).

A fundamental variation of this method is Room and Pillar Mining as used for coal mining. Extraction recovery using this method varies considerably generally being between 50 and 75% but averaging around 60%. (See Fig. 9.9).

ARTIFICIALLY SUPPORTED STOPES
A supported opening is one in which a significant part of the incumbent load is carried on an artificial support system. Straight gravity loading of overlying cover imposes a pressure of approximately 25 kPa per vertical metre of depth. This means that at a depth of some 150 metres the support system would need to have a capacity of 350 tonnes per sq metre for total support assuming the rock surrounding the rock opening carried none of the load. Support systems with this capacity are generally impractical except in final backfill situations. Use of light supports however will only carry a small percentage of such a load. Hydraulic props and close-spaced packs may have support capabilities of around 30 to 50 percent of the above load. Backfill has the greatest ability to support once subsidence and compaction have occurred.

SHRINKAGE STOPING
Shrinkage stoping is used to mine narrow or wide veins which are steep dipping. This mining method is basically an overhand stoping system in which a portion of the broken ore accumulates until the stope is completed. When solid rock is broken by blasting, the broken fragments occupy a larger volume; this is known as swell and is often of the order of 30 to 50 percent. (See Fig. 9.10).

FRAGMENTATION
This increase in bulk of broken ore is shrunk off periodically through chutes or drawpoints so as to maintain a working floor for additional mining. The ore must be strong enough to stand unsupported across the width of the stope and when broken should not pack to the degree that it cannot be drawn off. The hanging wall and footwall rock need to be relatively competent to prevent fracture and dilution of the ore.

During the period the stope is being mined both the hanging and footwall rock are supported to some degree by the ore in the stope. When all the ore within the stope is mined, the remaining ore is drawn off. Mining recovery of the order of 75 to 85% is achievable. (See Fig. 9.11).

![Solid: 1 cu metre](image)  ![Blasted: 1.5 cu metre](image)

*Figure 9.10 Rock swell on fragmentation*
CUT AND FILL STOPING

Cut and fill stoping is suited to deposits which have relatively steep dip, i.e. greater than the angle of repose of the broken ore. The ore is usually massive with the hanging wall of a type which will not stand for a long period without support.

Ore is broken by overhand mining and removed from the stope through ore passes. After the broken ore is removed, the stope is filled with waste to within working distance of the back and the mining cycle is repeated. The waste fill may be broken rock, sand and/or gravel, soil or classified mill tailings. Pillars generally can be recovered allowing almost total extraction/recovery of ore. (See Fig. 9.12).

The method is adaptable to mechanization and often referred to as (MICAF) Mechanized Cut and Fill Stoping. The main features are the use of Load Haul Dump (L.H.D.) diesel loaders, drill jumbos and ore passes in the country rock footwall instead of in the fill. (See Fig. 9.13).
STULL STOPING
Stull stoping is a method which employs systematic or random timbering (stulls) placed between the footwall and hanging walls of the stope opening. The deposit may be flat to steeply dipping and usually 3.6 metres or less in thickness. The stulls provide the only artificial support and usually depend on the hanging wall and footwall being relatively competent. For openings greater than 3.6 metres other support systems are necessary. (See Fig. 9.14).

SQUARE SET STOPING
Square set stoping may be employed where the ore deposit is structurally weak and where faulting and fracturing of the surrounding rock also makes it weak. The method is adaptable to deposits with irregular boundaries. Where other methods are inadequate this method is possible and provides high recovery of ore. However it incurs a high cost in labour and materials. Fill may be used to further support the timber sets. (See Fig. 9.15).

Figure 9.12 Cut and fill stoping
Figure 9.13 Diagrammatic layout of a Micaf stope (Mt Isa)

Figure 9.14 Stull stoping

Figure 9.15 Square set stoping

(Reproduced by kind permission of The Society of Mining Engineers, Colorado)
LONGWALL MINING
Longwall mining in its original concept is employed primarily to extract coal, although with modifications the method is used to extract metallic minerals e.g. uranium and gold. The method is applicable to deposits ranging in thickness from 0.9 metres to 2.5 metres and dipping less than 12°. The rock material overlying the deposit should be thin bedded, relatively incompetent and cave freely and completely behind the prop line. The rock floor should be competent to support the prop loads. A massive system of props is used at the face and working areas. This method receives extensive coverage under Mining Methods for bedded deposits.

CAVING METHODS
The general requirement of this method is massive type mineral deposits of large horizontal area, such as thick beds, or masses. The ore should be weak, or if hard, fractured with weak bonding. Overburden may be firm rock or drift material but most cave and follow the ore down as it is removed. The method is applicable to large low grade ore deposits. Because the cave progresses to the surface, the method can only be employed where such disturbance can be tolerated.

SUB-LEVEL CAVING
Sub-level caving can be used to mine massive or large pockets of ore and thick deposits which dip steeply. It may be employed to mine ore bodies below an open pit where factors are suitable for block caving. The rock material in the deposit needs to be moderately competent with jointing and fractures but not fire caving. The method requires the mining of the deposit from the top down with successive caves produced from sub-level intervals. Suited to low grade deposits, it is capable of high recovery with substantial dilution. (See Fig. 9.16).
Figure 9.16 Sub-level caving

Drills in footwall of the ore body connect development and production openings to ramp system, often emerging at the surface.
CHAPTER 9 MINING METHOD – UNDERGROUND METALLIFEROUS

9.18

Figure 9.17 Block caving

BLOCK CAVING
Block caving is normally utilized to mine massive and disseminated low grade deposits of large horizontal dimensions which are structurally weak. Both the ore and rock material of the deposit and overburden should be incompetent and cave freely. Rocks which are in this category are highly jointed, fractured or thin bedded rocks with low bond strength across joints. The rock material should break but not repack - allowing some grinding action during the caving cycle. Block caving is a low-cost high production method, but while high recovery is possible dilution may cause early cut-off and reduce recovery. (See Fig. 9.17).

TOP SLICING
Top slicing is essentially a method for mining massive, thick-bedded or wide veined deposits containing weak ore and walls which will not stand unsupported except over short spans. Ore extraction is in horizontal or near horizontal timbered slices starting at the top of the ore and working downward. A timbered mat is placed in the first cut and the overburden is caved. As subsequent cuts are advanced, caving is induced by blasting out props, but working room is maintained under the mat. Although the method is not selective, wall variations may be handled. High extraction is possible. The method has seen use in friable shallow deposits. (See Fig. 9.18).
Figure 9.18 Schematic top slicing method

(Reproduced by kind permission of The Society of Mining Engineers, Colorado)
## Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>ANGLE OF REPOSE</td>
<td>The maximum slope at which a heap of any loose or fragmented solid material will stand without sliding or coming to rest when poured or dumped in a pile or on a slope.</td>
</tr>
<tr>
<td>ADVANCING/RETREATING</td>
<td>Location of the working face with respect to the development of access opening.</td>
</tr>
<tr>
<td>BACKS</td>
<td>The mineral between the stope (or level) and the next level.</td>
</tr>
<tr>
<td>CHUTE/BOX HOLE/DRAW POINT</td>
<td>Access points at the stope bottom for removal of broken ore.</td>
</tr>
<tr>
<td>COMPETENT ROCK</td>
<td>One which is relatively free of defects and stands unsupported once openings are established.</td>
</tr>
<tr>
<td>CROSS CUT</td>
<td>A drive heading at right angles to the strike of the ore body.</td>
</tr>
<tr>
<td>DEVELOPMENT</td>
<td>The operations to open up a coal seam or ore body by sinking shafts and drives and cross cuts.</td>
</tr>
<tr>
<td>DIP</td>
<td>The angle of inclination of an ore body or surface below the horizontal.</td>
</tr>
<tr>
<td>DILUTION</td>
<td>The contamination of ore with barren wall rocks in stoping; it also applies to mixing which occurs with back fill material in cut and fill stopes.</td>
</tr>
<tr>
<td>FOOT WALL</td>
<td>The junction of the ore body and the country rock on the lower side of the lode, i.e. the wall upon which the ore body may be considered as resting.</td>
</tr>
<tr>
<td>HANGING WALL</td>
<td>The junction of the ore body and the country rock on the upper side of a lode (ore body).</td>
</tr>
<tr>
<td>INCLINE/DECLINE</td>
<td>Access way of low inclination.</td>
</tr>
<tr>
<td>INCOMPETENT ROCK</td>
<td>Rock which is so weak or so fractured (rock defects) that it will collapse if unsupported.</td>
</tr>
<tr>
<td>LEVEL</td>
<td>A main underground roadway or passage driven along the level course to afford access to the stopes and provide ventilation and haulage ways for removal of ore.</td>
</tr>
<tr>
<td>MINERAL DEPOSIT</td>
<td>An accumulation or concentration of minerals of economic interest.</td>
</tr>
<tr>
<td>ORE DEPOSIT/ORE BODY</td>
<td>A mineral deposit capable of economic exploitation.</td>
</tr>
<tr>
<td>OVERHAND/UNDERHAND/BREAST/RILL</td>
<td>The location of the working face of the stope relative to the workman.</td>
</tr>
<tr>
<td>PILLAR</td>
<td>An area of ore left to support the overlying strata or hanging-wall in a mine.</td>
</tr>
<tr>
<td>Term</td>
<td>Definition</td>
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<td>-------------------------</td>
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</tr>
<tr>
<td>PITCH</td>
<td>The angular inclination of an ore shoot, with respect to the surface, measured in the direction of the strike.</td>
</tr>
<tr>
<td>RAISE</td>
<td>Vertical access-ways driven upwards.</td>
</tr>
<tr>
<td>RECOVERY</td>
<td>In mining, this is the percentage of extraction of the ore reserve.</td>
</tr>
<tr>
<td>RIB PILLAR</td>
<td>Side pillar in a stope.</td>
</tr>
<tr>
<td>RIPPING</td>
<td>The breaking of roof generally in a mine heading to increase headroom for haulage and ventilation.</td>
</tr>
<tr>
<td>STOPE/CHAMBER/ROOM/FACE</td>
<td>The actual working or mining face or the excavation opening.</td>
</tr>
<tr>
<td>SILL AND CROWN PILLARS</td>
<td>Horizontal pillars above and below the stoping area.</td>
</tr>
</tbody>
</table>
CHAPTER 10 – Mining Methods – Underground Coal

TYPES OF MINES

There are basically three different types of under-ground mines classified according to the manner of making the opening from the surface to the coal seam.

A drift mine (See Fig. 10.1) is one in which a horizontal, or nearly horizontal, seam of coal outcrops to the surface in the side of a hill or mountain, and the opening into the mine may be made directly into the coal seam. This type of mine is generally the easiest and cheapest to open because no excavation through rock is required. Transportation of coal to the outside may be by truck haulage, belt conveyor, or by battery powered rubber-tyred equipment.

A slope or incline mine, (See Fig. 10.2) is one in which an inclined opening is used to tap the coal seam (or seams). A slope mine may follow the coal bed if the coal bed itself is inclined and outcrops, or the access incline may be driven through rock strata overlying the coal to reach a coal bed which is below drainage. With the advent of tunnelling machines inclined access is being used to greater vertical depth. Transportation of coal from inclined mines can be by conveyor or by truck haulage, using a trolley locomotive where grade is small or by inclined skip haulage for steeper grades. The most common practice is to use a belt conveyor where grades do not exceed 18°.

A Shaft Mine, (See Fig. 10.3) is one in which the coal seam is reached by a vertical opening from the surface to the coal seam or seams. Shafts are used where depth of cover is great.

Combination of these openings are often used.
CHAPTER 10 MINING METHODS – UNDERGROUND COAL

Figure 10.3 Shaft mine

TYPES OF MINING SYSTEMS

Underground mining systems used in modern coal mines are generally classified, according to the equipment used, such as conventional, continuous and longwall: one version of the latter is that of shortwall mining.

CONVENTIONAL
In the conventional mining system, the coal is extracted in a sequence of operations, using specific equipment in each operation. The coal face is undercut, centre cut or top cut with or without a shear cut (vertical) with a cutting machine. The blocks of coal outline are drilled using mobile powered drills or hand held electric or hydraulic auger drills. The holes are then charged with explosive and the coal is gathered by loading machines, or by combined loading and transportation equipment. Normally a shuttle car is used for transport to a belt conveyor or mine or loading point. Roof supports (wooden stulls) steel cross bars on posts, or roof bolts) are used, the latter usually installed by machine. Ventilation is then extended and the coal face is ready for the next cycle. For maximum efficiency in the conventional system, the number of working places is governed by the number of separate operations in the cycle.

CONTINUOUS
In the continuous mining system, a single machine called a 'continuous miner' breaks the coal mechanically and loads it for transportation. Roof support is then installed, ventilation is advanced and the coal face is then ready for the next cycle. Generally, the advance in a single cut which is about 6 metres, is limited by the length of the machine so as to keep the operator under supported roof at all times.

Continuous miners are generally classified into three categories.

BORING TYPE
Boring type continuous miners break the coal through the scraping action of an arm or arms rotating flat against the face. This type of machine produces a rounded or arched entry which is an advantage in roof support. The entry width and height, however are closely restricted by the size of the machine. The limited flexibility of mining height is a serious deficiency in seams where thickness varies or the top and/or bottom are irregular. Likewise, the limited flexibility of mining width may cause serious ventilation problems.

The boring type machine is comparatively high and can be used only in seams 1.6 metres thick or greater. It has large capacity but is relatively inflexible. Opening width is of the
order of 4 metres. Machines have been developed with cut heights of up to 3.5 metres and widths of 6.0 metres.

**RIPPER TYPE**
Ripper-type continuous miners break the coal through the sawing action of cutting chains against the face. This type of machine can produce a square or rectangular entry of variable dimensions, and is particularly useful where selective mining is practised. The primary disadvantage of this type of machine is that it produces large quantities of fire coal and dust; however, this problem has been largely overcome by the use of improved water sprays with jogging (mist) nozzles. The machine can be used to advantage where there is extremely poor roof, because it cuts a narrower face than is possible with other machines.

**MILLING OR DRUM TYPE**
Milling or drum type, continuous miners break coal through the picking action of bit wheels on cutting heads rotated parallel to the coal face. Although this type of machine is the most widely accepted, there is a variant which uses an oscillating rotating cone cutter, generally referred to as road headers.

**LONGWALL**
In the longwall mining system, large blocks of coal, outlined in the development process, are completely extracted in a single, continuous operation. Hydraulic yielding jacks or self-advancing jack units (chocks) support the roof at the immediate face as the coal is removed. As the face advances the strata is allowed to cave behind the supporting units.

Longwall mining machines fall into two general categories, ploughs (planers), or shearsers (shearer-loaders).

The plough is the simpler of the two machines, composed of a bladelike arrangement fitted with fixed bits or a saw-toothed edge that is pulled along the coal face by a heavy chain, powered and controlled from one end of the face. The plough normally cuts 75 to 150 mm of coal from the face on each pass, and most ploughs are made to cut while travelling in either direction. The broken coal is removed onto a chain-type conveyor by the ploughing action of the machine.

The shearer machine comes in either a single or double drum design. The double-drum style is used for mining thicker coal seams or those of variable thickness. Cutting bits are set in rows around the periphery of the drum or drums. Using the sidewalls of a chain type face conveyor as a track, the machine rides on the conveyor, and is pulled across the face by sprockets on the machine that engage the links of a heavy steel chain. As it moves across the face, the shear cuts a slab of coal as much as 0.6 metres deep from the face. A steel plough-shaped attachment to the machine moves the broken coal onto the face conveyor when the shearer is in motion. As with the plough the shearer can be designed to mine coal in one direction only or in both directions. The power for the shearer is supplied by an electric motor mounted within the machine itself, and operating controls are located on the machine.

The longwall method of coal mining incorporates high recovery and high production in conjunction with powered self-advancing roof supports.

Seam heights generally range between 1.0 metre and 1.8 metres and production capabilities of between 500 to 2000 tonnes per shift with recovery averaging approximately 85%.

Face length can range from 100 to 1000 metres though 180 metres is considered optimum.
Longwall mining systems offer the advantages of coal recovery near that attainable with conventional or continuous systems, and efficient mining under extremely deep cover or weak roof. The surface effects of mining are minimized, by allowing uniform subsidence. Unrecoverable roof support costs are minimized, since the primary supports move with the unit, tonnes per shift and tonnes per man are attractive. Ventilation is simplified and no rock dusting is required along the longwall face.

Disadvantages include the high costs of equipment, setup, teardown and moving to another section. Surface subsidence can also be undesirable. Production is dependent on a small number of units, so equipment failure can be critical.

The coal seam must be fairly uniform in thickness, and the top and bottom must be regular. The system depends upon conventional or continuous mining for development. The extent of development required is considerable. Actual longwall production accounts for approximately 60% of a mine output where the method is used.

SHORTWALL

This method is a combination of continuous mining and longwall systems. Continuous mining or conventional equipment is used to develop the section. A continuous miner in conjunction with longwall type roof supports is used to extract the remaining coal pillars.

The method offers good recovery with reduced roof support costs and by using development equipment on retreat, equipment costs are reduced.

Glossary of Terms

<table>
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</tr>
</thead>
<tbody>
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<td>ADVANCING/RETREATING</td>
<td>Location of the working face with respect to the development of the area.</td>
</tr>
<tr>
<td>BORD</td>
<td>A narrow coal drivage in the pillar and stall method of working.</td>
</tr>
<tr>
<td>CHOCK</td>
<td>An arrangement supporting the roof; may be timber, stone, steel or hydraulic jack.</td>
</tr>
<tr>
<td>CLEAT</td>
<td>Term used to identify the major joint system in the coal seam.</td>
</tr>
<tr>
<td>GOB/GOAF/WASTE</td>
<td>The worked out or waste area after mining.</td>
</tr>
<tr>
<td>INBYE</td>
<td>A direction away from the shaft towards the face.</td>
</tr>
<tr>
<td>KERF</td>
<td>The groove or slot made in a coal seam by a coal cutter.</td>
</tr>
<tr>
<td>OUTBYE</td>
<td>A direction towards the shaft.</td>
</tr>
<tr>
<td>PANEL</td>
<td>The working of coal seams in separate panels or districts.</td>
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</table>
CHAPTER 11 – Mine Development

SURFACE MINING

Initial development work associated with surface mining operations is development drilling and bulk sampling. The purpose of this work is to determine the grade, volume, and three-dimensional outline of a mineralized zone previously located by exploration. It is distinguished from exploration or reconnaissance drilling, which aims at discovering new mineralized areas.

A development drilling and bulk sampling programme should supply the following information:

- Geology of mineralized zone;
- Quantitative data on grade and tonnes of material with pertinent cut off limits (grade or lease boundaries);
- Physical size and shape of the deposit;
- Mineralogical and metallurgical characteristics of ore;
- Physical characteristics of the ore;
- Bulk samples for metallurgical testing and grade check;
- Data on other factors that could affect mining operations, such as ground water, ground conditions, etc.

TESTING METHODS: (PITS, SHAFTS, TUNNELS)

In some potential ore zones amenable to open pit mining, important preliminary information on grade, material mass, and structural conditions can be obtained at reasonable cost by test pitting. With shallow overburden, simple hand pitting, use of a back hoe, or bulldozing can open up areas for mapping and sampling. Use of shallow seismic geophysical equipment can usually determine if the overburden is thin enough to permit bulldozer ripping. Where ore zones are near surface or flat lying coal deposits, test pitting may provide much of the basic data in determining grade and tonnes.

The high cost of underground work as a basic method of testing a mineralized zone for grade and tonnes usually precludes its consideration. However, tunnel or shaft programmes may be required in complex ore zones where grade appears to be markedly influenced by geological features which are difficult to interpret on the basis of drill hole data or meagre surface geology; though much of the work is required for bulk samples for metallurgical and grade analysis.

However most of the ore body information comes from test drilling.

BULK SAMPLING

Bulk sampling is done to provide adequate samples for metallurgical analysis, and to determine if drill hole samples are representative for the area assigned to them in the ore-reserve calculation.

Mill design must be based on samples that are representative of the entire ore deposits: if the ore is uniform in metallurgical characteristics the location of bulk samples presents no
Development in open pit consists of driving working trenches (ramps) and the excavation of barren rock (stripping) on production horizons (benches). Initial development stripping and overburden removal can use a wide range of methods similar to the actual method of mining or equipment suited to the conditions encountered.

The initial opening of an open pit is referred to as a Box Cut. (See Fig. 11.1). Successive development of down ramps in open pits is made with dropping cuts. The box cut may be on the exposed outcrop or in overburden/waste where the ore body is buried. The box cut in open cost coal mining consists of developing the initial trench. This is usually done by side casting a dragline. (See Fig. 11.2).
ADVANCED STRIPPING
Where considerable stripping must be done prior to and during the operation of a surface mining operation, there are a number of factors which have bearing on selection of the stripping methods and equipment used.

1. The size of the ore body and distribution of values within that ore body. Is the ore massive or scattered, bedded or disseminated, thick or thin?
2. The nature of the overburden to be removed. Is it hard dense rock, bedded rock (thick or thin), friable material, earth, sand, clay or marsh?
3. The character and significance of geological structures (fractures, faults, shear zones) associated with ore occurrence. Are there waste bearing formations with resulting water disposal problems?
4. The nature of the overburden, including its alteration products, the physical or chemical conditions which, when combined with anticipated climatic conditions, may render certain equipment inoperable during unfavourable seasons.
5. The life and expected production rate of the operation. Is the production to be continuous or intermittent?
6. The calculated capacity of, and haulage distance to each disposal area.
7. The future use of the equipment. Is it to be used for stripping only? What is the effect of ore blending requirements on equipment size when it is also to be used to mine ore?

The cost of stripping and selection of equipment will also be influenced by the character of the terrain and the proximity of the ore body to waste disposal areas.

PIT PLANNING AND LAYOUT
Open pit mine planning must be correlated with all phases of a mining operation. The pertinent elements that must be included are: assays, geology, tonnage and area extent of ore reserves, topography, mining equipment, economic factors of operating costs, capital expenditure, profits, types of ore, pit limits, cut-off grade, stripping ratio, rate of production, pit slopes, bench heights, road grades, ore metallurgical characteristics, hydrological conditions, property boundaries and marketing considerations.

A significant aspect of pit planning and layout is designing the pit from reference data such as plans and sections as well as developing a mining sequence. Hence the development of the deposit is related to a wide number of variables peculiar to the ore deposit.

UNDERGROUND METALLIFEROUS MINING

SYSTEMATIC DEVELOPMENT
The purposes of mine development are:

- to provide openings for stoping and transporting of mineral;
- to obtain further and more detailed information as to the character and size of the ore body.

Two factors arise: the mode of entry or mine access and lateral or subsidiary development which deals chiefly with workings within the ore body.

MINE ACCESS – MODE OF ENTRY
One of the most important considerations in the planning of an underground mining operation is that of access to the workings.
Mine access is usually required to provide for the following functions:

- transport of men and materials to the workings;
- removal of ore from the mine;
- ventilation of the workings;
- carrying services such as compressed air, water and electric power into the mine;
- gaining access into an ore body or potential ore body for purposes of exploration of the ore.

**TYPES OF ACCESS**

Access into a mine may be provided by means of Shafts and Adits.

**Shafts**

The shaft access may be either vertical or inclined or a combination of both. The vertical shaft is the most common type of access used when the ore body is steeply dipping. The vertical shaft is also common in shallow to medium depth ore bodies irrespective of the dip of the ore.

When an ore body has a dip less than 90°, the distance of the ore body from a vertical shaft increases with depth. As a result the amount of development necessary to connect the shaft to the ore body at lower levels can reach the stage where it is uneconomic to consider a vertical shaft. (See Fig. 11.3).

Where the distance from the shaft at the lowest level of the ore body is too great, the alternative is to use a combined vertical/inclined shaft (See Fig. 11.4) or an inclined shaft as in Fig. 11.5.

**Adits**

An adit is a horizontal or near horizontal heading which is open to the surface/atmosphere at one end, e.g. it has a single portal. (See Fig. 11.6(b)).

Classification of horizontal or near horizontal mine openings may be made according to the access to the ground surface.

A tunnel is a heading connected to the surface at both ends. (See Fig. 11.6(a)).

A drift or drive is an internal mine horizontal opening accessible from a mine shaft via a shaft station or plat landing. (See Fig. 11.6(c)).

For an adit to be considered for selection as an access the topography and position of the ore must be suitable.

An ore body adjacent to a cliff or in the middle of a mountain is ideally suited. Declines are inclined adits with grades of between 1 in 7 and 1 in 9. Near surface, ore bodies may be developed by means of declines. Where Load Haul Dump (LHD) units are used underground, ramp systems are often internal to the mine while major access is by vertical shaft.
Figure 11.3 Vertical shaft

Figure 11.4 Combined vertical/inclined shaft

Figure 11.5 Inclined or underlay shaft
SIZE OF ACCESS
The cross sectional dimensions of the access will depend on:

1. Hoisting method, i.e. skips, conveyors, trucks.
2. Services to be installed, i.e. size of pump mains, number of electric cables.
3. Number of men and quantity of material to be transported i.e. size of cage to be installed.
4. Type of ground through which the shaft is to be sunk.
5. Cost of sinking, equipping and maintaining the access.

POSITION OF ACCESS IN RELATION TO ORE BODY
The cost of sinking a shaft or driving an adit and then equipping it with shaft guides, rail tracks or conveyors may cost millions of dollars. Consequently it is economically justifiable to spend a considerable amount of time and money in the selection of the site for the access.
The thorough investigation of a site requires extensive investigation by drilling and often underground development to check the physical properties of the rock, the presence of faults and water, which will affect the progress and cost of sinking and the later maintenance of the access.

The shaft can be positioned on the hanging wall or footwall side of the ore body or through the ore body. (See Fig. 11.7).

The shaft in the hanging wall will be located in a position based on the knowledge of the type of rock through which the shaft will be sunk. It may be found that the rock in the hanging wall is much stronger than in the foot-wall thus making it more economical to develop the shaft in that position.

Another important point to consider is the mining method or methods selected for the extraction of ore. The H/W shaft is satisfactory if cut and fill stoping, or a method in which fill is introduced into the stope immediately stoping is completed, is selected. The H/W shaft is not desirable if a caving method is selected, as the influence of caving on the stability of the rock in the H/W is usually more pronounced than its influence on the F/W.

![Figure 11.7 Relative positions of shaft location](image_url)

**Figure 11.7 Relative positions of shaft location**

**SHAFT IN THE ORE BODY**

A once common practice was to sink the shaft in the ore body and do as much level development as possible within the ore body. One reason for this was that the ore broken whilst developing assisted in financing the development of the mine.
However, because of the cost of developing a shaft it is important to plan for it remaining in operation for as long as possible, preferably the life of the mine. This means, that in order to protect the shaft, a large amount of ore around the shaft must be left unmined as a shaft pillar.

The footwall is considered the most favourable position in which to sink a shaft. The advantages are:

- The position has a far greater chance of being outside the influence of ground movement caused by stoping operations.
- Because the shaft is sunk outside of the ore body, ore is not left in a shaft pillar.

Other significant factors in the location of the mine access are topography, depth and property/lease boundaries.

**SHAFT SINKING**

There are three common shapes of shafts: circular, rectangular and elliptical.

Circular Shafts are most commonly used in coal mining, but find almost universal application in present mining ventures. The circular section is desirable, especially at depth, because it is best able to resist heavy side pressures. For a given cross-sectional area, it represents the least rubbing surface to the ventilation air current. It is best suited to sinking under difficult conditions and to the insertion of water-tight lining and it is comparatively cheap to maintain.

Circular shafts can be sunk quicker than rectangular shafts of comparable cross-sectional area and require fewer holes to be drilled per round. However they cannot be conveniently divided into compartments as rectangular shafts can; their shape does not lend itself to a permanent timber lining where timber is plentiful and cheap and the life of the shaft relatively short.

The minimum size of a circular shaft is approximately 1.5 metres whilst diameters above 7.6 metres are rare. Fig. 11.8 illustrates a typical general arrangement including plat layout for a circular shaft: in this case K. 57 Shaft Mt Isa Mines.

Rectangular Shafts are used extensively throughout the world. They lend themselves to the use of timber lining and can be divided into compartments with great economy in space. These two points are important where the shaft is used as sole access and for ventilation. Given the right circumstances rectangular shafts are the cheapest and most convenient means of gaining access to and exploring deposits of ore.

The smallest rectangular shaft is about 1.4 metres x 1.8 metres, the minimum size which will allow sinkers to drill and load debris without undue restriction of working. Large shafts up to 13 x 4.2 metres and eight compartments have been sunk. Utilizing modern equipment, high rates of sinking and lining may be achieved at reasonable cost hence they are not likely to be entirely replaced by circular shafts.

The rectangular section has the following serious disadvantages:

- there is very little room for essential services such as electrical cables, compressed air pipes and pump-rising mains;
- it is extremely susceptible to ground movement and, because the clearances are minimal, the cost of maintenance may be prohibitive;
ventilation problems may arise in shafts having few compartments, particularly when cages meet at mid-point. The ventilation by-pass of ten provided may represent a serious weakening of the surrounding rock. (See Fig. 11.9).

Elliptical Shafts aim at combining the advantages of both rectangular and circular shafts, i.e. to obtain a strong and durable section whilst making maximum use of available space. The shape may not be elliptical in the geometrical sense of the word but is composed of four circular areas as illustrated in Fig. 11.10.

An elliptical shaft may be designed to have a concrete lining, (See Fig. 11.11).

Figure 11.8 General arrangement of k57 shaft including a typical plat layout

Figure 11.9 Three compartment rectangular timber lined shaft
SHAFT SINKING ELEMENTS
In conventional shaft sinking the process is a cyclic operation involving drilling, blasting, excavation and ground support.

Shaft Collar
The shaft collar is usually a concrete section going down from the surface to and keyed into competent bedrock. Initial sinking may be done in conjunction with a crane and clam shell.

Drilling – Types of Drills
Hand-held sinkers are used in small shafts especially where bottom bench rounds are employed. Shaft jumbos, either hydraulic or pneumatic, are popular in that fewer operators are required and fast accurate drill patterns can be achieved. This contributes to good fragmentation for efficient mucking with a minimum of overbreak. Jumbos have particular application to circular shafts.

Types of Drill Rounds
It is important to make the greatest advance possible in the drilling cycle with a minimum of overbreak. Round depth is matched to conditions, but is controlled by co-ordination of the drilling, mucking, lining and support cycles of the shaft sinking operation.

There are two types of round used: full face and bench. The choice is based on the type of ground, water in the shaft bottom, and type of mucking method. Full face is generally the best for mechanical type mucking and also for the most rapid advance. Bench rounds are
best for hand mucking because the broken muck lies on a steep slope from which it can be scraped into the bucket or kibble. Bench rounds under wet conditions provide a sump for drainage while working on the top bench.

In square or rectangular shafts with full face blasting a Vee cut may be used.

In a circular shaft a pyramid cut or a burn cut may be used.

When a shaft bottom is benched down a bench cut is used.

**Blasting**

Most shaft sinking is wet and the rounds are tight, so water resistant, high strength explosives are required. It is extremely important to take great care in loading, priming and wiring up to prevent misfires, which can be extremely dangerous in the drilling and mucking cycle. Electric firing is invariably used with a harness. (See Fig. 11.12). Holes are generally bottom primed.

**Mucking**

Hand mucking is not frequently used because of the high labour cost.

Mechanical mucking machines such as track mounted rocker shovels, clam shell buckets, Cryderman units and cactus grabs may be used. All four are usually operated by compressed air.

Riddell Muckers consist of clam shell machines however they are difficult to operate. (See Fig. 11.13).

Cryderman muckers have a positive opening and closing clam. The shaft bottom can be mucked with only one operator who is up off the shaft bottom. The mucker can be mounted in a single compartment so that it can be raised and lowered. (See Fig. 11.14).

The Cryderman is also used in inclined shafts. (See Fig. 11.15).

Emico Rocker shovel mucker, model 630 tracked machine is suited to shafts of diameters of 5.5 metres and above. (See Fig. 11.16).

For inclined shafts an adaption of rail-mounted rocker shovels has been achieved using an air winch mounted on, and operating in conjunction with, the bogger’s traction drive.

Cactus Grab, This type of mucking unit is designed to work from the bottom-most deck of a multi-deck stage and operates radially and circumferentially as it slides along a centrally pivoted boom. (See Fig. 11.17).
Join to hanging wires

Plan of shaft harness–bus–wire layout using 148 electric delay embedded detonators connected in parallel.

Figure 11.12 Shaft sinking detonator harness in shaft sinking

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Figure 11.13 Riddel type mucker in a rectangular shaft
Figure 11.14 Cryderman mucker in a vertical rectangular shaft

Figure 11.15 Cryderman mucker in an inclined (underlay) shaft

Figure 11.16 Tracked bogger mucking out a circular shaft – concrete lined
Sinking Stage – Sinking Hoist
The sinking stage is the in-shaft structure, usually multi-decked, equipped and designed to facilitate shaft sinking and shaft lining installation.

The main requirements of stage design involve:
- rate of sinking and lining;
- spacing of divider sets;
- method of mucking used;
- length of concrete pour where lining is done in conjunction with sinking.

Fig. 11.17 illustrates a multi-deck sinking cage.

![Multi-deck sinking hoist and cactus grab shaft mucker in a circular shaft](image)

Figure 11.17 Multi-deck sinking hoist and cactus grab shaft mucker in a circular shaft

The sinking hoist used in conjunction with multi-deck stages are those of multi-rope suspension types such as the Blair-stage hoist. (See Fig. 11.18). The illustration only shows one of the two ropes.
Ground Support – Lining
Timber Sets. In the past rectangular shafts have relied upon timber sets for ground support and compartment division. (See Fig. 11.19).

Size of the timber used and set spacing is dependent on ground conditions encountered. Timber sets provide flexibility in that set spacing can be changed depending upon the ground encountered. Disadvantages or timber sets are the cost, strength, short life and fire hazard involved.

Timber sets are usually hung from the preceding set with steel hanging rods. After the set is in place, the posts are inserted and the hanging rods tightened up. The corners and dividers are blocked into the wall on line and squared up, and then the lagging is placed in around the sides of the shaft. Shaft bearing sets need to be provided at regular intervals to provide support and ensure alignment of the shaft. Timber sets should be lagged solid on the four walls in friable ground, and between the hoisting compartments and the manway compartment.

Steel Sets
Steel sets may be used instead of timber. These may be used in circular shafts where rigid shaft conveyance guides are used and manway compartments and staging are required. Steel sets allow rapid installation and dimensional accuracy.
CHAPTER 11 MINE DEVELOPMENT

Cast iron – Steel Lining
Where high hydrostatic pressures are encountered concrete lining may fracture, therefore cast-iron or steel linings may be used. These also have application where the factor of depth makes concrete lining impractical.

Concrete Lining
Circular shafts are invariably concrete lined. This enables good sinking rates to be achieved. The advantages associated with the circular concrete section are good strength for ground support, good air flow characteristics and low maintenance. It offers flexibility where requirements change, e.g. from hoisting shaft to ventilation. Also water can be controlled or sealed off more easily in this type of shaft.

The procedure involved in concrete lining is to use a segmented formwork which is suspended from shaft fixtures or the concrete forms in the previous lift. Concreting is done in stages or lifts. The lower section of the concrete form work is referred to as the kerb ring at the bottom of which are sealing apertures for scribing boards to provide a leakproof seal for the concrete. The kerb ring also has provisions for forming pockets or bunton boxes. Buntons are the equivalent of shaft dividers. Equipping the shaft with compartments - installation of buntons etc - can only be done when sinking is completed and the sinking cage removed.

The concrete is mixed on the surface and supplied by pipe line, through a remix kettle (plate deflector), and distributed to the back of the shuttering (formwork).

The concrete used must be fairly fluid with a high ultimate strength and must contain an accelerator to ensure rapid setting.

Ventilation
Natural ventilation does not clear shot-firing fumes fast enough. Forced ventilation using exhaust fans or blowing arrangements is used.

RAISING (CORNISH RISING)
Raising may be accomplished by conventional methods or by mechanized methods. Raises generally are hard to ventilate and working upward at a height is difficult. Raises of over 60 metres in height become difficult and slow because of access and getting supplies to the working face. In raises over 40°, broken muck can be disposed of by gravity, but raises below 40° require mechanical means such as slushers. Raises are used for a number of purposes, for example ventilation, ore passes, manways and stope development. They are often the first stage in making a connection around which a shaft can be enlarged and extended to depth by stripping out by benching.

CONVENTIONAL HAND METHODS
Open – Bald Headed Raises
This type of raise uses no compartmental timbering or division. Access to the face is by ladders usually in the form of chain ladders pinned to the footwall side. For drilling out, a temporary stage is erected using timber and steel pins into short holes. This type of raise is dangerous and requires experienced miners, though it is relatively fast and inexpensive.

It is used for short raises such as draw points and chute openings for stope load outs. (See Fig. 11.20).

Compartmentalized raises consist of crib type and stulled raises.

Cribbed raises are a two-compartment raise in which the ground requires support and is used for permanent manway access to stopes or other mine workings. (See Fig. 11.21).
Stulled raises are used for relatively self-supporting ground, providing a safe manway access to the face during raising. The timber supports can be removed once the raise is developed. (See Fig. 11.22).

Figure 11.20 Bald headed open raise

Reproduced by kind permission of The Society of Mining Engineers, Colorado

Figure 11.21 A three compartment crib raise

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MECHANISED RAISING METHODS
The mechanised methods and equipment available for raising are basically:

- raising by long hole drilling;
- raising with a suspended work cage;
- portable raise climbing drilling platforms;
- raise boring utilizing back reaming machines.

Raises By Long Hole Method
The method consists of drilling holes, in a pattern suitable for the size and shape of the envisaged raise, through the full depth of the ground. Drilling is usually done from the top level using either conventional equipment or special precision drilling rigs.

Large diameter centre holes are used for the burn cut configuration. Once the complete round is drilled for the entire depth, the raise is blasted in sections of approximately 3
metres. The hole bottoms are plugged by down lines and back loading for charging is done. A series of cuts are made starting from the bottom of the intended raise.

The advantages of the method are:

- relatively smooth walls are obtained;
- it eliminates work inside the raise as with conventional raising.

Raises by this method are suitable for developing cut off raises and slots in sub-level stopes.

**Raising With a Suspended Cage**

Raises may be developed from a cage hoisted on, or by a rope suspended through a bore hole. The bore hole is pre-drilled on the line of the raise and has a finished diameter of 75 to 125 mm.

Drilling out and charging of the blast holes is done from the cage. A protective canopy on the cage on re-entry to the raise after firing allows scaling/barring of loose rock fragments in relative safety. Face advances of around 4 metres per blast are possible. Overall length of raise possible by this method is of the order of 100 metres. The Jora Lift which incorporates a hoist in the cage is a version of this method.

**Portable Climbing Platforms**

Climbers such as the Alimak system can develop raises at any angle with large cross sections.

The equipment consists of a steel track guide rail, a working platform equipped with an electric or compressed air motor to enable it to climb the guide rail which incorporates service lines, e.g. compressed air and water. The guide rail is installed on sections pinned by rock bolts to the wall of the raise.

The working platform allows the miner to be raised to the workface to bar down, drill out and charge the round. On descending, the platform is retracted out of the way ready for firing. Forced ventilation to clear blasting fumes and dust is provided by an air/water mixture from the guide track.

This method is suited to long raises in hard rock. (See Fig. 11.23).

**Raise Boring – Reaming**

The raise drilling rig is a rotary drilling machine using short drill section stems. The machine, using a multi-bit is capable of back reaming and cuts a hole of between 1 and 2.4 metres in diameter.

The first step is to establish a connection between the top and bottom of the proposed raise by boring a pilot hole of 175 to 300 mm diameter. This is then back reamed to the required size.

There are however two basic approaches:

1. the pilot hole is drilled down to a mine opening below and subsequently reamed upwards;
2. the pilot hole is drilled upward and the finished raise is reamed downward.

The first approach is preferable because the cuttings from the reaming head, which represent the major portion of the rock being removed, fall to the opening below by gravity with no further interference to the drilling operation. (See Fig. 11.24).
This type of raise drilling system offers speed in advance when compared with conventional drill and blast methods. Costs per cubic metre of rock removed may be higher than conventional raises. However the release of miners for other development projects can be significant.

Raise boring has particular importance to ventilation air-ways. Decline mines can be developed blind and with increasing depth exhaust ventilated by raise-bored ventilation shafts. The major advantage of the smooth walled finish is in reducing air resistance.

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Figure 11.23 Alimak raise climber stage in working position
WINZING

Winzes are development openings below an existing level or drive. The work involved is similar to that of sinking a small shaft. They may be vertical but are often inclined following an ore contact down dip.

Because of the small cross section hand bogging of broken material is necessary. Progress is both slow and difficult, therefore winzing is not frequently chosen as a means of development access.
SHAFT SINKING BY ROTARY DRILLING

SINKING WITH RAISE BORING EQUIPMENT
Shaft deepening or secondary/auxiliary shafts may be done utilizing the pilot raise method. Where a lower level is developed and the reaming bit can be connected to the pilot hole, at the hole through, and mucking out of drill cuttings is possible, then the method can be used. The bored raise is slashed out to its final dimensions by conventional drill and blast techniques.

Large diameter raise bore holes can also be utilized as manway shafts as well as auxiliary hoisting shafts without enlargement.

CONVENTIONAL ROTARY DRILLING
The drilling rigs are usually the same as those used for drilling deep oil and gas wells; however the drilling assembly, drill pipe, and drilling tools, are specially designed for large hole drilling.

Of prime importance in conventional rotary drilling is the ability to drill caving formations or aquifers that would make shaft sinking by conventional techniques economically impractical. Generally rotary drilling has one advantage over shaft sinking by conventional means where hole diameter is small and the ground formations are soft. To date, holes drilled range between 3.6 metres diameter and 1670 metres depth, to 7.6 metres diameter and 520 metres depth.

DOWNHOLE DRILLING MECHANICS – VERTICAL MOLE
This is basically a downhole drilling machine which reams out an initial pilot hole to an existing underground opening. The opening may be reamed out to a diameter up to 4.3 metres. The advantage of this approach is the relatively light weight surface drilling equipment required in comparison with conventional rotary drilling for a shaft of this size. However the method is not suited to very soft formations.

TUNNELLING BY CONVENTIONAL MEANS
Most tunnels which are drawn for access to mining operations fall within a limited range of classification. These range between 7.5 m² and 23 m² in cross sectional area and on comparatively flat gradients; declines for mechanized equipment access may have gradients between 1 in 7 to 1 in 9.

The entry drift into an ore body is used to establish access from the surface and provide a haulage-way for transporting the ore out of the mine. An access tunnel also may serve as a ventilation duct between the mine workings and the outside atmosphere.

There is a fundamental difference between tunnelling and production mining. The advancement of a development heading is cyclic with only one face. Production mining can be organized so that a number of operations are performed simultaneously and continuously. Development headings therefore are usually less efficient and more costly on a volume basis than other mining operations.

Because of space limitations and job duration, tunnel driving usually calls for smaller materials handling equipment whose unit operating cost is higher than that obtained when using larger machines necessary for economical mine production.

Development work such as shaft sinking and initial decline - access may be done by contractors, especially at the beginning of a new project. Subsequent routine development in conjunction with the operating mine is usually performed by the mine personnel.
Tunnels may be divided into three sizes:

- **Small**, i.e. less than 10 sq metres in cross-section area. The smallest drift economical to drive is approximately 1.8 metres wide by 2.4 metres high. Mechanized loaders cannot be used in smaller drives.

- **Medium-size** with cross-sections between 10 sq metres and 25 sq metres. Most conveying and haulage drifts fall into this category.

- **Large-size** with cross-sections between 25 sq metres and 40 sq metres. Access of this size is necessary where standard-gauge railway and large conventional off-highway trucks are used for haulage.

Most mine main levels and drives fall within division 1 and 2. The size of main haulage levels is largely dependent upon the haulage equipment used.

Mine drives and cross cuts other than main levels differ from tunnels:

- usually of small cross section;
- less emphasis on precise alignment and shape of section;
- in breaking of ground, advantage may be taken of relative softness of ore or adjacent rock;
- main level/tunnels are more permanent hence more elaborate support for long term access;
- tunnels/main levels usually bear entire cost of development as capital expenditure.

**TUNNELLING OPERATIONS**

The cyclic operation for medium to hard rock driving consists of drilling, blasting, mucking and support.

**Drilling**

The selection of drilling equipment is governed by:

- Cross-sectional size of the heading.
- Length of drive.
- Cost of labour per man-hour.
- Value of time to be saved (by faster completion).

Small headings usually use pneumatic air-leg machines. Medium to large headings utilize a drill carriage or jumbo either rail-mounted or rubber-tyred, with Drifter drills, mounted on hydraulically positioned booms, which drill holes of between 37 and 50 mm diameter. Large diameter reamed holes of 125 mm can be used for burn cuts.

**Blasting Explosives**

Factors requiring consideration in the selection of explosives for drives are:

- Type of rock.
- Degree of fragmentation required.
- Condition of holes - wet or dry.
- Ventilation.

AN/FO explosives are used where possible. High strength water resistant cartridge explosives are used where conditions are adverse.
The firing sequence of blast holes is critical in the development of a free face to which to break. Burn cuts permit deep rounds to be made but electric firing with short delay detonators is the norm.

Mucking
Hand mucking with the bonjo (shovel) is almost entirely replaced by mechanical muckers or boggers; except for hand clean-up. The type of loader must be matched to the tramming/transport equipment used. Basic considerations for excavator selection include the size of the drift, tramming equipment and the power available.

The most common mucking machine is the air-operated overcast loader, either track/rail or trackless, which may discharge into a transport hopper or muck truck.

Ground Support
This was traditionally provided by timber, however steel has largely replaced timber as a structural support material. Rock bolting and pneumatically applied concrete are used extensively.

ROCK TUNNELLING WITH MOLES
Initially boring machines were used in relatively soft rock where geological conditions were very favourable. However progress is being made in mechanical boring of medium and hard formations. The advantages and disadvantages of boring headings are:

ADVANTAGES
• Speed. Under suitable conditions boring machines will out-perform conventional methods.
• Reduced overbreak and increased safety. Boring machines provide a clean tunnel perimeter because the overbreak often inherent in blasting is eliminated.
• Reduced support requirements. In formations requiring roof and side support, a smooth bore will require less support than a roughly blasted opening.
• Less surface disturbance. Tunnel borers can operate in areas where blasting is either impossible or impractical because of complications or the nature of the formation.
• Reduced labour requirement. Labour requirements are generally less for machine bored drifts.

DISADVANTAGES
• High capital cost. The high capital cost of a boring machine requires that the contractor or mining company be certain that the machine is suitable for use over a sufficient length of drift to write off the machine cost economically.
• Inability to bore hard rock. While some machines have proved they can bore very hard rocks, the cost of cutters has been prohibitive. Cutter design improvements are changing this situation.
• Lack of experience under difficult tunneling conditions. Most drift projects have been under favourable conditions. Planning of projects for difficult conditions is a considerable handicap.
• Assembly and disassembly time. This is a disadvantage for short headings.

Rock classification and compressive strength are important because machines achieve penetration by grinding away the face. The actual type of cutter used will depend upon the type and strength of the rock.
TYPES OF BORING MACHINES
There are fundamentally three categories: jumbo, shield and gripper. Each exhibit some features in common with one another. All have a type of cutterhead in which is incorporated mechanical elements for breaking the rock followed by devices designed to pick up the broken material. A non-rotating structural member supports the cutterhead and provides support for the opening and enables directional control.

Jumbo Machines
These are employed in tunnels of 7.6 metres to 12.0 metres in diameter. The cutterhead is attached to the forward end of a structural-steel framework resembling a conventional drill jumbo. This type of machine is usually employed in soft rock where steel supports are required. Placement of ring support is done immediately behind the cutting head.

Shield Machines
These types of borer consist of a shield with thrust rams and erector system. The cutterhead and cutterhead support are contained within the shield instead of the work platform. Shield machines are usually applied in soil formations or in situations with variable or mixed faces. These machines have application in sewerage tunnel projects.

Gripper Machines
In rock which is competent and generally self-supporting, borers of the gripper type are used. The rock walls must be capable of accepting the grippers or wall thrust loads imposed on them to enable the face thrust to be generated. These machines handle rocks up to Moh's hardness scale of 7 - 8.

Jumbo, shield and gripper machines are generally 'moles' meaning a full face mechanical boring machine. The machines are often custom built for specific projects, though a number of companies produce standard models.

Other types of tunnelling machine fall in the category of continuous miners common to coal mining.

DEVELOPMENT IN COAL
OPENING LOCATION AND MODE OF ENTRY
Where the coal seam or bed lies below drainage the shaft or incline/main portal is located approximately central to the area to keep haulage and other distances short. However surface topography may restrict this choice.

The inclination of the bed is an important consideration. Other things being equal it is more important to keep loaded haulage downhill, where gravity drainage of water is also possible. Where dip is above 3° and track haulage is contemplated, angling entries (headings) across the pitch may allow suitable grades to be maintained.

Where beds outcrop uphill haulage - working down dip - may be the logical approach.

ADVANCE/RETREAT
Full retreat means developing or driving entries to the boundary or limit/lease boundary and working back; although this approach is advantageous overall, it requires a greater investment and more time before production begins. Development production can be increased by utilizing a number of headings and modern continuous miners.
With full retreat, the shaft or the access incline, other facilities and permanent openings are in undistributed territory. Ground problems of pillar recovery and caving are left behind and sealing problems with ventilation are reduced.

A compromise between retreat and advance is to advance on one side of the mine or working territory and retreat on the other to complete extraction. This requires proper protection/support of the main openings during the advance phase. Often during development an area may be used to provide production to assist in generating a cash flow.

NUMBER OF OPENINGS
The coal property is divided and sub-divided into areas by driving sets of entries consistent with the proposed mining system. The number of entries in a set is determined by the requirements of ventilation, haulage, escapeways and mine services such as power, water and drainage. At least two openings in an entry are required, one for intake and the other for return. Actual numbers may vary and may be as many as 8 or 12. The number depends upon desired production as well as ground support factors.

Heading design must consider the ability of an opening to resist strata load, and its expected life as time is a factor in continued stability. Other factors include nature and strength of the floor and coal. Matching headings with haulage equipment is important in providing adequate ventilation which is also critical.

HEADING ADVANCEMENT
A wide variety of equipment may be used in driving the headings making up an entry. Continuous miners, loading machines, shuttle cars and extendable conveyors enable rapid advancement. Additional advantages from boring and cutting machines are better condition of roofs and ribs as a result of the elimination of blasting; the smoother more uniform shape reduces resistance to air flow.
CHAPTER 12 – Rock Stability and Ground Support

INTRODUCTION

Ground control is one of the major problems especially in deep mining because the stresses (force per unit area) are great enough to cause sloughing of developmental openings as well as walls and faces in stopes even in hard rock mines. Deep conditions can exist at modest depths where tectonic stresses are equal to those due to gravity or where formations are weak.

For typical mine openings the passage of stress around the opening produces concentrations in the face and relocation in the walls. Rock bursting (sudden yielding of the rock) as well as less violent forms of instability, usually occur either in the face because of high stresses, or in the wall rock due to fracturing in consequence of the release of stress. Rocks exhibit variability in structural strength and failure cannot be predicted with certainty, therefore the concept of probability of failure must be adopted.

Planning must therefore aim at avoiding unnecessary excessive stress concentrations: it is important to produce stress conditions which do not endanger safety or incur high costs.

GROUND CONTROL OBJECTIVES

The essential elements of mining are extracting ore from the earth’s crust and maintaining adequate stability in the surrounding ground.

The ratio of the maximum stress in the formation to the strength of the rock can be used to predict critical stability conditions. 'Deep' is often interpreted as meaning 1500 metres or more, where stresses are high enough to cause sloughing.

The major problem in deep mining is achieving effective ground control and minimizing sudden release of stress/rock bursts.

With full retreat, the shaft or the access incline, other facilities and permanent openings are in undistributed territory. Ground problems of pillar recovery and caving are left behind and sealing problems with ventilation are reduced.

A compromise between retreat and advance is to advance on one side of the mine or working territory and retreat on the other to complete extraction. This requires proper protection/support of the main openings during the advance phase. Often during development an area may be used to provide production to assist in generating a cash flow.

ROCK FAILURE

Rock failure occurs when the force acting upon the rock is greater than the actual rock strength. This force is known as the rock stress.

Stress is generally due to the gravitational loading of the overlying rock subjecting the rock at a greater depth to higher stresses.
Where an excavation is made in the rock, the load which was originally supported by the excavated rock, has been transferred to the surrounding rock. This causes an increase in the rock stress near the surface of the opening.

The magnitude of this increase depends upon the shape of the opening, the nature of the original stresses, i.e. direction and magnitude, and the geological structure near the opening, such as joints, faults and folds. If the rock is weak or the depth is great, the redistribution of the load may raise the rock stress above the strength of the rock, thus causing the rock to fail.

This can be a sudden failure causing rock bursts or more commonly a gradual failure causing progressive loosening of slabs of rock.

Even when the rock has failed, an immediate rock fall may not occur because of arching effects caused by the interlocking of the broken pieces.

SCALING

The operation of barring down or scaling involves the prying loose of rock from the backs/roof, walls or face of an excavation to make the opening safe for personnel in case of falls of rock fragments.

Traditionally this has been achieved by using a bar, hence the term 'barring down'.

When tapped loose rock makes a hollow sound while solid material has a high pitch - (solid sound).

Mines Regulations 12.8 requires that:

"Every underground excavation in which persons work or travel shall be scaled and maintained in a safe condition".

With mechanization and mobile equipment usage in stoping operations special purpose scaling machines have been introduced. The use of lift platforms enables scaling, and rock bolting to be done in otherwise inaccessible backs.

Mechanized roof scalers which incorporate a boom mounted hammer especially suited to bedded type deposits have also been introduced for scaling backs. With the use of rock bolts, mechanized scalers have brought about significant improvements in accident statistics involving roof falls.

TYPES OF GROUND SUPPORT

ROCK BOLTING

Rock/roof bolting is a widely accepted technique for reinforcing the peripheries of mine and tunnel openings in all types of rock. Bolting is not necessarily cheaper than conventional timber support, but significant indirect advantages accompany bolting: reduced storage and handling requirements for ground support materials, improved ventilation due to removal of obstructions, decreased size of opening that must be excavated to achieve a given clearance, greater freedom of trackless vehicles without the risk of dislodging supports, and negligible maintenance of ground supports.

There are several theories explaining the action of rock bolts under different circumstances. Those of importance are summarized below:
(a) *Laminated beam*

In thin to medium bedded rock, two or more strata are bolted together, forming a laminated beam strong enough to support itself as well as the overlying rock. The principle is similar to that used by the carpenter who bolts together several layers of boards to make a stiff beam. Rock in this condition offers the most favourable opportunity for support by rock bolting and when strata are not broken by cross fracturing, rock bolts 1.2 to 1.5 metres apart will furnish sufficient support for roof or walls. (See Fig. 12.1)

(b) *Suspension outside the caving arch*

It is well known that when the back of a drive fails, it will do so only for a short distance above the drive and the back will assume the shape of an arch. The height of the arch is usually proportional to the width of the drive at the area of failure.

This arch will form when the rock is strong and has deteriorated because of very high stresses, or if the rock is weak and fails under its own weight. The profile will vary but the arch will be there.

To prevent the back of the drive falling in, it is possible to anchor rock bolts into sound rock beyond the line of the natural arch, i.e. ground which is known not to fail. Rock bolts hold ground, which is likely to fail, with bearing plates or mesh hung from the rock bolts. (See Fig. 12.2)

The shape and size of the arch depends on the rock type and probable failure planes in the rock.

(c) *Consolidation of fragments*

Rock may be broken into numerous slabs, blocks or irregular inter-locking fragments but without voids or open fractures. In such cases each rock bolt has the effect of clamping or squeezing together the rock fragments in a circular zone around it so that they tend to form a solid mass.

![Figure 12.1 Rock bolting - Laminated Beam Effect](image)

*e.g. hanging wall strata in a cut and fill stope.*
A number of these competent (rigid) blocks or key stones around the perimeter of a tunnel will act as a masonry arch and will support themselves.

The size of the zone compressed into a competent mass by the rock bolt depends upon many factors. Some influencing factors are the characteristics of the fragment size, shape, strength, rigidity and roughness of surface and the length of the rock bolt. The effect of these factors in any particular situation may be determined by experimental installations.

Rock bolts which depend on this principle for supporting the rock around a tunnel should, for moderately jointed rock be about as long as half the tunnel width. For supporting rock fractured into small pieces, it may be necessary to use bolts as long as 2/3 of the tunnel width. Shorter bolts may be used in the walls. (See Fig. 12.3)

(d) **Surface effects**

In the above instances, the primary effect of the bolts is to strengthen the rock structure. They also perform the important function of compressing the rock surface and limiting the effect of rock expansion by preventing the opening of portings, cracks and joints which would be subject to attack by air, water or temperature change.

The fracture of conventional supports to hold the surface of roof rock tightly in place is often responsible for the development of excessive vertical pressures. If exposed rock is allowed to settle or fall, then the rock above it loosens and settles also. The process continues until broken rock has expanded enough to fill the available space above the support, which must then support the active column of failed rock. The use of mesh and sprayed concrete can assist in holding the surface material in place in conjunction with the rock bolt.
TYPES OF ROCK BOLTING

Rock bolting methods: four major types
(a) partially anchored tensioned rock bolts
(b) fully anchored tensioned rock bolts
(c) fully anchored untensioned rock bolts
(d) tensioned fully grouted rock bolts.

Partially anchored means anchored at the bottom of the hole only. Fully anchored means anchored along the whole length of the hole.

(a) Partially anchored tensioned rock bolts

Typical rock bolts of this type used in mining are:
(i) High tensile rock bolt with cone and shell (expansion shell) anchor;
(ii) Solid bar (plain or deformed) with slot and wedge (split bolt and forged wedge);
(iii) Resin grouted bolts which may also be fully anchored.

These rock bolts exert a positive force against the rock before any rock movement begins. Tensioning is achieved by tightening the bolt for (i) while for (ii) once the wedge is in place the tensioning nut is tightened.

This type of support should be installed as soon as possible after excavation. The bolts are not corrosion resistant or protected in the hole. (See Fig. 12.4)

Tension type rock bolts are often installed with a bearing plate which indicates the installation tightening to the correct tension. The Brown Bearing Plate is one such plate. (See Fig. 12.5 (a) and (b))

(b) Fully anchored tensioned rock bolt

Typical bolts of this type are corrugated and deformed (steel reinforcing bar).

The bolt is fully grouted acting as an anchor for tensioning the short amount of bolt between the grout and bearing plate and it also acts as a reinforcement bar.
Tensioning improves the bolt supporting action where mesh or plates are used. (See Fig. 12.6)

Figure 12.4 Partially tensioned rock bolt

(1) Brown bearing plate

(2) Incorrect installation – No tension in bolt.

Figure 12.5a Brown bearing plates for rock bolts
(3) Correct installation bolt tension plate deformation at 3600 – 5000 tension.

(4) Incorrect installation – overload bolt

Figure 12.5b Brown bearing plates for rock bolts

Figure 12.6 Fully anchored and tensioned rock bolt

Fully anchored tensioned rock bolts may also include resin sealed bolts which may be of steel, fibreglass, wire rope or wood dowels tensioned or untensioned.

Resin type bolting systems utilize a cartridge holding a resin and a catalyst in separate chambers which is pushed into the borehole; then the bolt is inserted, penetrating the capsule. The bolt is rotated mixing the two elements and the resin gels and sets within 12 to 120 seconds, attaining 90% of its final strength in 2 to 5 minutes.

A wide variety of rock bolts suitable for resin grouting is available.

A recent version equivalent to the fully-anchored tensioned rock bolt is that of the split-set friction rock stabilizer which uses a hollow, compressible, high-strength steel tube with a slit extending its full length, a taper at the upper end and a retaining ring at the bottom. The split-set is installed into a smaller diameter bore hole. (See Fig. 12.7)
(c) *Fully-anchored untensioned rock bolts.*

A typical bolt of this type is a solid deformed bar dowel which is fully grouted. These rock bolts will act as reinforcement bars holding cylinders of the rock surrounding the bolt in one solid piece. This action is equivalent to the support given by a conventional timber or steel support. (See Fig. 12.8)

(d) *Tensioned fully grouted rock bolts.*

This type of rock bolt includes solid deformed bar with slot and wedge and fully grouted. Although fully grouted they are regarded as partially anchored tensioned rock bolts mainly for corrosion protection to achieve their full effect; it is essential that they are tensioned before the grout sets. (See Fig. 12.9)
ROCK BOLT SUPPORT DESIGN

(a) Choosing the correct rock bolt

Factors which must be considered when choosing the type of rock bolt in a situation are:

- **Size of excavation to be supported.**
  
The length of rock bolt necessary to support an opening is generally proportional to the span of the excavation and depends on the rock strength. For instance wide excavations will require long rock bolts. In places of soft ground where the width of the drives or turnouts exceeds 3.6 metres, impractically long bolts would be necessary and set support becomes necessary. However in strong rock, most excavations can be suitably supported with rock bolts.

- **Period for which support is required.**
  
  Unprotected high tensile or other rock bolts will last up to 5 years in dry conditions but in wet acid conditions may last for only 3 months. Corrosion resistance may be achieved by fully grouting the bolt.

- **Ground conditions.**
  
The strength of the rock is a major factor in selecting the type of rock bolt support. In weak leached ground only fully anchored corrugated rock bolts are suitable.

  In broken blocky country, any of the tensioned rock bolts will provide adequate support, while in solid unbroken rock, untensioned fully grouted dowels are sufficient.

  The nature of the country will also determine whether conventional plates are suitable or if they have to be supplemented by wire mesh.

- **Pressure expected.**
In areas where the expected pressures are high the use of rock bolting depends on the period for which support is required. If the period is short, then the area may be rock bolted but if long, then some form of set should be installed.

- Depth of Anchorage necessary.
  
  It is essential that rock bolts are well anchored in a competent zone of rock, otherwise the primary aim of compressing the rock is defeated. This factor partly depends on the size of the opening and it determines the length of the rock bolts used.

(b) Rock bolt length and spacing

The following factors determine the length and spacing of rock bolts:

- the estimated depth of loosened distressed zone;
- the geological structure of the area;
- the orientation of the opening to the bedding/strata;
- the width of the excavation.

The length: spacing ratio should not be less than 2 wherever tensioned bolts are used; i.e. for a spacing of 1 metre bolts not shorter than 2 metres should be used. This is necessary to ensure that the zone of compression caused by each bolt overlaps the adjacent one to form a complete ring of rock in compression.

ROCK BOLT QUALITY CONTROL

Rock bolts frequently do not do the job required of them, either through faulty installation or rock bolt failure. Because only the plates, nut and possibly some thread is visible it is difficult to detect rock bolt failure.

A high degree of effectiveness will be achieved by selecting the correct bolt and anchorage for particular circumstances, ensuring quality of installation and using bearing plates on tensioned bolts.

Rock bottom patterns are designed assuming that a given load will be borne by each bolt. When one or more bolts fail to carry the load assigned to them, either through faulty installation or through rock bolt failure, the extra load must be borne by adjacent bolts. The additional load may prove greater than the adjacent bolts can withstand and they may fail. A chain reaction may be the result. Apparent safe roof conditions are actually potentially dangerous.

CAUSES OF FAILURE

(i) Anchorage failure from corrosion or failure of the bearing rock.

(ii) Anchor or bearing plate failure caused by over-tightening of the rock bolt or not using wedge washers where the plate is not square to the rock bolt.

(iii) Bolt failure due to corrosion.

(iv) Ground movement caused by adjacent stoping or development.

(v) Excessive load on the rock bolts.

(vi) Poor installation of rock bolts.

Areas where rock bolt failure is most likely to occur are:

- Ground where bolts are exposed to water.
- Ground where leaching is advised.
- Ground exposed to heavy firing.
• Ground subjected to heavy pressures.
• Where installation has been partly carried out.

ADVANTAGES AND DISADVANTAGES OF ROCK BOLTS AS MEANS OF GROUND SUPPORT

Advantages
• Rock bolting applies a restraining force to the rock soon after installation, thus providing suitable ground support.
• A large amount of flexibility is available i.e. different rock bolt types and patterns are available to provide support in a great variety of situations.
• Rock bolting is normally cheaper than installing sets.
• In many cases rock bolting can proceed with other mining operations, causing only minor disruption to the cycle.
• Rockbolting takes up little of the drive space, thus eliminating distractions to trackless equipment, drilling or airflow which are prevalent with other ground support methods.
• Rock bolts can be used in any shape drive without necessitating stripping.
• Rock bolts are not a fire risk.

Disadvantages
• Rock bolts suitable for supporting wide spans have to be impractically long and sets or cribbing are often more practical.
• Rock bolts give little indication of impending ground failure.
• It is difficult to ensure the quality of rock bolt installations.

Limitations of ground failure quality can be overcome to a degree with the use of bearing plates and rock bolt torque tensioning devices for use with tensioned bolts.

ROCK BOLT SUPPORT IN MINES

METALLIFEROUS AND INDUSTRIAL MINERAL MINES
Rock bolts are most extensively used:
• as individual bolts with bearing plates;
• as straps or mats incorporating two or three bolts;
• as bolts with wire mesh usually with steel straps. These may subsequently be spraycreted.

COAL MINES
Rock bolts are used in conjunction with header plates to increase bearing load; there may be timber or steel plates/strap/mats/roof bars.

Roof trusses, an important support method, are adjustable steel tension members which are anchored into the roof rock to support the span roof load. (See Fig. 12.10)
SET SUPPORT

There are a large number of different types of (structural member) ground support. A typical selection of the various types being examined are:

- Post-column type.
- Rigid sets/frame type supports.
- Sets with yielding hydraulic props as legs.
- Yielding arch sets.

**POST-COLUMN TYPE**

**POST**

Basically this is an upright member which supports the back or mine roof. Referred to as stalls as in stalled stopes. (See Fig. 12.11)

**CRIBBING/PIG STYE**

Timbers laid at right angles on top of each other to support heavy roof loads. (See Fig. 12.12)

**RIGID SETS/FRAME TYPE SUPPORTS**

Rigid sets are either steel or timber supports placed after excavation to prevent ground failure. They are usually designed by assuming that a certain amount of rock has to be held in place by the sets: this is the way they act. With a knowledge of the geology of the area and rock mechanics, sets can be designed for most situations.
TIMBER SETS
These consist of cross beams and posts. A timber frame is used mainly to support the roof, sometimes the sides and the floor of the mine drive. The simplest timber set consists of a cross bar cap, or header supported by two upright posts. (See Fig. 12.13)

STEEL SETS
These utilize R.S.J. (rolled steel joist) (See Fig. 12.14)

This type of set is similar to shaft sets using steel.
12.14

(1) Simple timber set
(2) Three piece set with inclined posts
(3) Four piece timber set
(4) Cap reinforced with stulls eg timbering of a leading stope

RIGID STEEL ARCHES
These are curved segments of steel normally H sections, used for the support of permanent mine openings. The arch shape is a more efficient use of the steel sections than flat cross bars. Without arch, the steel member is in compression instead of bending. Steel arches are used in portals/adits and shaft station areas.

Timber lagging is used along with back filling of voids. (See Fig. 12.15)
ADVANTAGES AND DISADVANTAGES OF RIGID SETS

Advantages
Rigid sets have the advantage over rock bolting in that they can be used to support wide spans effectively.

- Rigid sets do give a warning of impending ground failure.
- Rigid sets have a good effect on morale. They look as though they are effective.

Disadvantages
- Cost. Rigid sets are more expensive than rock bolts and generally offer minimal opportunity for recovery as do temporary supports.
- Timber sets do not fit well into conventional drive stopes: considerable lagging and packing is often required.
- Sets constitute obstructions to airflow, production drilling and trackless equipment.
- Sets can be knocked out easily by trackless equipment, with possible ground collapse.
- Timber is a fire hazard.
- Where rigid sets fail, they provide no further support and have to be replaced. Yielding sets may be utilized to advantage for this reason.
- Rock bolts usually provide a better quality support which will resist ground movement more than sets.

SETS WITH YIELDING HYDRAULIC PROPS AS LEGS

Yieldable jacks and props are mechanical support units which yield at a specified load and are set across the seam, bed or opening. Yielding is accomplished by hydraulic support cylinders equipped with pressure relief valves. Yielding is required where roof convergence occurs. It has particular application to coal mining.

Advantages of hydraulic props.
- They are quick and simple to install.
- They do not require specialized crew (as do timber sets) for installation.
They yield steadily under load. They are much more reliable in this respect than friction props which tend to yield in jerks.

Disadvantages.
- They are liable to damage by machine equipment.
- Prop maintenance is a problem in bad areas.
- They have a high initial cost.

Because hydraulic props are expensive compared with sets, it is essential to recover them for further use at all times.

YIELDABLE ARCH SETS

This type of support incorporates a sliding friction assembly to accommodate heavy pressure and thus delay damage and distortion of the support. Yieldable arches are used under severe conditions where extensive movement occurs because of the presence of faulted ground or in areas of subsidence in the vicinity of active longwall operations. Yieldable arches reduce repetitive maintenance work. Where the ground load exceeds the design load of the arches, yielding occurs at overlapping joints. As this occurs, the joint overlap increases gradually and the radius of arch segments is reduced. The strength of the arch is increased as it yields and the ground settles into a natural contour bringing forces into equilibrium.

Yieldable arches are used most in ring and semi circular shape, and consist of various section sizes in 3 to 5 sections depending upon the size of the opening to be supported.

To provide rigidity in a lateral direction, sheets made of steel channels or rods are used. Lagging and back filling are necessary. (See Fig. 12.16)

![Figure 12.16 Yieldable steel arch assembly](image-url)
Advantages of Yielding Arch Sets

- The arch gets stronger as it yields because the length of the set exposed to the ground pressure becomes smaller.
- The yielding action allows built up stresses to relieve themselves.
- The sliding arches adjust themselves until all of them carry nearly equal loads. Rigid supports usually carry unequal loads and fail in the order in which they are excessively loaded.
- Because of the inter-locking spreaders between arches, the yielding of any arch results in a uniform yield of its neighbour because it exerts equal forces on these arches through the spreaders. Uniform yielding is also promoted by using suitable lagging.

Disadvantages of Yielding Arch Sets

- They are expensive.
- They are heavy to handle and require skilled installation crews.
- They require maintenance.

SEGMENTAL STEEL PLATE

Segmental steel plates of various types consist of curved members of a size and mass which can be handled readily by one or two men in a relatively confined space. This type of support (also called liner plate) is used with safety where loose ground of running nature does not permit the opening up of much unsupported ground. In coal mines, liner plates may be used when driving entries through previously mined out workings. Portal construction can also be facilitated. Cement grout can be injected into the ground behind the plates to stabilize the material.

There are three types of plates:
- Corregated plates without flanges.
- Corregated plates with two flanges.
- Plates with four flanges.

The last two allow assembly bolting from inside the tunnel while the first requires outside access.

CONCRETE SUPPORT

POURED MONOLITHIC CONCRETE

Monolithic concrete is cast on, or poured in place, and has construction joints. It is used mostly for shaft lining and shaft stations; however it may be used for crusher stations, pump sumps, slusher drifts and ore passes. Reinforcement material is steel bar and mesh.

GUNITE

Pneumatically applied portland cement mortar is sprayed on the rock of mine roadways and shaft stations to prevent weathering. Sometimes the rock is reinforced by roof bolts and weld mesh. Cement-sand mortar is referred to as gunite and is applied as a 50 to 75 mm layer.

However gunite, while it has been used for some 50 years, has been found to be incapable of providing a dependable support in underground excavations. It has the tendency to loosen as rock surfaces relax and spall with the greater relaxation of more incompetent rock
area and members. This could be due to the thinness of individual applications aggravated by shrinkage induced by high cement content.

SHOTCRETE
Shotcrete is often defined to include indiscriminately pneumatically applied mortar and concrete. However, it is spray concrete that has been developed to become an effective ground support method in tunnels and other underground excavations. Shotcrete is a concrete with a maximum aggregate of 20 mm which is pneumatically projected at high velocity onto the rock surface. The material is compacted essentially without sagging or sloughing.

Shotcrete bonds effectively with any form of rock surface. Where applied with skill it can support a cohesionless oil. Its adhesion to cohesive surfaces is attributed to the peening effect of the coarse aggregate particles driving their predecessors into the subject surface, and to the high early strengths reached with the aid of a suitable accelerator. Shotcrete can be applied in layers of 100 to 150 mm thick in one pass, thus achieving its supporting function in addition to a seal. Shrinkage is less than with mortar mixes (gunite).

Using accelerating admixes enables the shotcrete to reach a quick enough set to adhere to wet and running water surfaces. Compressive strength afforded by shotcrete lining is high, giving also a fairly good flexural-tensile strength. From 75 to 80% of the 28 day strength is reached in 48 hours.

Shotcrete, applied immediately after blasting will supply both a seal and support, stabilizing a new rock surface. The intimacy of the rock-shotcrete bond is such that a new tough skin is formed upon the opening which restrains loosening, decomposition, and the bending that accompanies normal relaxation. Tensile stresses due to bending are diminished and compressive stresses are absorbed into the surrounding ground. Thus, a rock of minor strength is transformed into a stable one. This explains why an excavation in weak to plastic rocks remains stable against 100 to 200 mm of shotcrete support.

Shotcrete is a cohesive material, tougher than conventional concrete of similar mix proportions. It is waterproof and is characterized by high early strengths, due to the degree of compaction received from impact velocities of 75 m/s to 150 m/s, to its low water-cement ratio (about 0.35), and the use of accelerators developed for the function. Rebound occurs, usually of the order of 15%.

As a support system, it can be utilized either as a structural or non-structural support, particularly in weak to plastic rocks and soils and cohesionless soils requiring application of a rigid, competent structure to restrain the ground from loosening or flowing into the opening. This can be achieved by design thicknesses of shotcrete of 100 mm or more.

In more competent rocks, it may be applied only to joints or fractures to prevent the lesser rock movements that trigger rock pressure and failure. Shotcrete is often used in conjunction with other supports such as rock bolts, rock bolts with weld mesh and steel arches.

BACKFILL
Backfill includes waste sand, rock and classified mill tailings used to fill voids in mines after the removal of ore from stopes or other underground openings.

OVERBURDEN
Overburden material from surface excavations and barrow pits has been used though currently relatively few use the method. Fill is passed down large diameter boreholes or fill
raises. Distribution to stopes is done by mobile equipment. Hang-ups in fill passes can occur, also large particle sizes should be avoided to eliminate large void spaces.

DEVELOPMENT ROCK
Mullock - waste rock from barren mine developments can be handled in a similar manner to overburden. Voids in this type of fill are sometimes filled with fine material, such as tailings sand, to provide more effective resistance against ground movement.

HYDRAULIC BACKFILL
Hydraulic backfilling with classified mill tailings (fine fractions removed) is the most effective method of supporting the walls of mined-out underground openings. Once the fill handling system is installed it requires a minimum of time and labour for backfilling during the mining cycle.

Typical Arrangement of Backfill Operations:
A surface plant consists of classified mill tailing storage or a sand fill plant where dry sand is slurried to be piped underground. Cement may be used in the backfill to provide added strength and fill rill control so that pillars adjacent to filled stopes may be removed without undue dilution by backfill.

Backfill in slurried form is piped underground; actual vertical pipelines can be steel pipe or bore holes.

Level distribution lines are used to transport the fill from the main vertical lines or bore hole to the stopes or other excavation to be filled. Gravity feed is adequate unless excessive horizontal distances are involved.

The hydraulic fill requires reasonable water permeability or percolation rate - allowing drain off of water, instead of the material staying in a fluid state. Up to 10% cement may be added to a 100 to 300 mm fill capping to reduce fill dilution in mucking out cut and fill stopes.

The significant aspect regarding fill placement is adequate stope preparation, sealing off but with adequate provision for water removed and reliable telephone communication between the fill plant and fill operators in the stope.
CHAPTER 13 – Loading and Haulage
(Excavation and Transport)

EXCAVATION EQUIPMENT – UNDERGROUND (HARD-ROCK) EQUIPMENT

After the ore has been broken by drilling and blasting or other methods, it must be excavated from the working face, removed, placed into haulage equipment and delivered from the mine. Excavation equipment, or loaders, are captive or non-captive depending upon how they obtain their power. As captive, they may obtain power through a compressed air hose or electrical power cable. Some loaders are rail or track mounted and have restricted direction and movement; trackless loaders are free to move about the working place on crawlers or wheels.

OVERSHOT LOADERS

These are also known traditionally as mechanical boggers, being devices which pick up blasted ore in front of the machine and discharge it to the rear without turning. They may be compressed-air electric or diesel-powered and may be tracked or trackless. (See Fig. 13.1)

Overshot loaders which discharge from the bucket into mine cars, trucks or other conveyances require large head room clearances. Where headroom is restricted, over-shot loaders with conveyor discharges are often used. Conveyor discharges are able to load longer haulage units, and the conveyors also serve as a storage area for loaded ore while switching empty haulage units into position. Machine operators ride a foot plate on one side of the machine except when remote control is used.

A version of the overshot loader is that of loader and hauler, the hopper loader, which is equipped with an over-shot bucket mechanism. The bucket discharges into a hopper also mounted on the machine and six to ten passes of the bucket are required to fill the hopper. The machine is compressed-air powered and is mounted on rubber wheels, either two or four-wheel drive.

A further development of the compressed air loader is the compressed-air-powered load-haul-dump machine. These machines pick up a load in a single pass and travel to the dump point or truck.

SLUSHER HOISTS AND SCRAPERS

Slusher hoists and scrapers are used to move ore over distances from 15 to 120 metres. Scrapers are well suited for transfer of ore out of multiple-drawpoint headings such as block caving stopes. scrapers are also used in areas where headroom is limited. These are steep stopes or where mobile loading is not applicable.

The scraper is a steel blade equipped with a yoke, which in turn is attached to a pull cable. The scraper's digging and hauling characteristics are a function of its own weight, the curvature of the blade, and the proper fastening of the pull rope. There are four general types of scraper, all bottomless:
1. box
2. hoe
3. crescent and
4. the folding type.

The box scraper is suited to fine, easy flowing materials and because of its long side plates, it is able to contain a load well over long distances.

Figure 13.1 Overshot loader compressed air driven rail mounted

The hoe has good digging characteristics in coarse muck but does not retain material well in transit. Most scraper blades are then a combination box-hoe. The crescent type is suited to fine material but is expensive and unsuited to large material. The folding scraper is basically a hoe with the blade pivoting in the yoke. When loading, the folding scraper blade locks into the proper digging and hauling position. On the return to the muck pile, the scraper blade collapses, providing less return resistance, better roof clearance and faster return.

Scraper blades are often of cast construction without bolted fasteners, thus eliminating working loose in service. (See Fig. 13.2)

The motive power for the scraper is a stationary 2 or 3 drum hoist. The pull cable from the slusher hoist transmits the power to move the scraper and its load to the dumping point. A tail rope runs through a tail sheave anchored at the face to return the scraper to the face for the next load.
Triple drum slusher hoists are used to overcome the problem of having to manually relocate the tail sheave to reach all broken muck in a relatively wide working area. The triple drum hoist, in effect, has two tail ropes, both fastened to the back of the scraper. The scraper can be positioned over a triangular shaped area by reeling in one tail rope and unreeling the other.

Slusher hoists are available in compressed-air-powered form between 4 to 11 kW, while electric power hoists range from 9 to 112 kW.

The power required for the rope pull to move the scraper and load will depend on:

- the mass of the scraper
- the mass of the load
- the coefficient of friction between the loaded scraper and the floor. This is taken to be 1.0 for rough bottoms and 0.5 for concrete railed bottoms;
- the grade of the floor.

The return speed is usually \( \frac{4}{3} \) faster than the pull speed and consumes about \( \frac{4}{3} \) of the pull load power. Rope speeds are of the order of 50 to 80 metres per minute.

**FRONT END LOADERS**

The front-end loader, or tractor-shovel consists of a bucket attached to a set of arms which extend from the front of a tractor frame. The arms, with the attached bucket, are raised and lowered by hydraulic rams. The bucket is also rotated on the ends of the arms by another set of hydraulic rams, permitting the bucket to assume digging, dumping or carrying positions. All front-end loaders depend upon the tractive effort developed between the machine and the ground for penetration into the muck pile. Once the bucket lip has penetrated, the hydraulic cylinders provide the rotational force which breaks the pile loose and allows further penetration of the bucket lip.
Crawler-track-type front-end loaders produce high tractive forces because of the traction provided by the crawler treads. Crawler loaders find best application when loading up or downhill on slopes over 20% or in rough or wet conditions. Side tipping (1 or 2 way) buckets are more advantageous on less manoeuvrable crawler loaders and are used primarily for loading in confined working widths. Crawler loaders have low travel speeds and are best suited where travel between load and dump is short.

Rubber tyred front end loaders are generally four-wheel drive to provide maximum tractive force for crowding the bucket into the muck pile. Most machines have torque converters and power-shift transmissions which prevent engine stall while crowding.

Rubber tyre loaders are more agile, swift and manoeuvrable than crawler units hence their wide application. Articulated-frame, rubber tyre units, while more expensive than rigid frame units, offer sharper turns and greater manoeuvrability.

For rubber-tyred loaders the single most expensive maintenance item is the tyres; therefore attention is necessary to keep road and work areas well maintained. Wheel spin is the major tyre wear problem. Steel chains may be used to advantage to increase tyre life.

The operation of diesel-powered loader units under-ground produces toxic exhaust gases which must be treated by scrubbers and dilated with fresh air before release to the atmosphere.

Most front end loaders used underground were developed from surface units and in general require high headroom heights especially for direct truck loading. Where low headrooms are necessary load-haul-dump units may be used.

LOAD-HAUL-DUMP MACHINES

This type of machine evolved from a machine with a loading dipper ahead of the front wheels which discharged into a hopper behind the front wheels. Ground level discharge of ore is accomplished by opening the bottom of the hopper. The load-haul-dump machine today, features a low-profile, narrow-width, four-wheel drive self-loading hauler with a front-tipping bucket carried forward of the front wheels. (See Fig. 13.4 and Fig. 13.5)

These machines are long in comparison with width and use articulated steering capable of 45° swing. The operator faces sideways on the machine and visibility is the same when travelling in either direction. Machines have the same haul speeds in either direction thus eliminating turning around.
While load-haul-dump machines were originally designed to be self-loading haulers, they have been employed extensively for developing trackless mining systems, including development work of sinking declines. Load-haul-dump machines do not perform well when mucking uphill grades of greater than 12 to 15%; hauling production is most efficient below adverse grades of 15%.

In comparison with load-haul-dump machines, rubber-tyred front-end loaders are less economical, because of the greater mass per rated bucket carrying capacity and much higher power per rated capacity. Front-end loaders are specifically meant for digging and the need for high machine mass and higher power is evident. Where a front-end loader is used primarily to carry material increased operating costs evolve. Also larger haulage ways are necessary for front-end loaders.

To take advantage of the load-out capability of front-end loaders, use of stockpile areas, or truck loading bays in the case of declined development, has been made. Where stope bogging is used and height clearance is not a problem and direct loading of trucks is possible front-end loaders are often selected.

Load-haul-dump equipment has permitted the development of ramp mining methods, permitting the mining equipment to move between levels under its own power without being disassembled for moving. Ramps are usually developed with load-haul-dump machines, and are also used for production haulage in some cases.

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Figure 13.4 Cavo D170 Load-Haul-Dump Unit – bucket capacity 1 m$^3$; Body volume 5 m$^3$
CHAPTER 13 LOADING AND HAULAGE (EXCAVATION AND TRANSPORT)

GATHERING-ARM AND LOADING MACHINES

Machines of this type have sloping aprons on the front with chain conveyors passing up through the centres of the apron and over the chassis to discharge into a track or shuttle car. The conveyor is loaded by two pick-like arms mounted on either side of the conveyor on the apron. These arms are driven by rotating discs and are linked so that the arms travel parallel to the side of the apron as they enter the broken muck, then sweep across the apron and in towards the conveyor. The arms are timed so they alternatively pass towards the conveyor. The machine travels on crawler tracks and is operated from side mounted controls.

Initially these machines were designed for coal, though some use has been made of them in hard rock.

COAL EXCAVATION EQUIPMENT

There are three types of face-loading equipment in use; these are mobile loading machines, continuous miners and longwall miners.

The loading machine loads the broken coal from the face into shuttle cars. The gathering arm loader is typical of this type.
Continuous miners consist of four basic types: ripper, milling, boring and low-vein auger (this type is used in conjunction with short wall mining methods). Each, except the low-vein auger type, incorporates a gathering arm-conveyor discharge for loading shuttle cars.

Longwall mining machines are either ploughs or shears. These excavate or cut the coal from the face where it is removed by a chain-type conveyor.

SURFACE EXCAVATING EQUIPMENT

Excavating equipment used in open pits and quarries may be broadly classified into cyclical methods and continuous excavators.

CYCLICAL METHODS

SHOVELS
The quarry-mine shovel has particular application to hard rock loading of shot rock. Machines range from small size to those of 23 to 30 cubic metre capacity. Increases in capital costs have meant the increase in size with other types of excavators replacing small capacity shovels.

The basic shovel has three sets of machinery. First, hoist machinery is mounted on the deck or revolving frame. This controls the up and down movement of the dipper bucket for digging and dumping. Where rope crowd is used a hoist controls the in and out movement of the dipper stick or handle for dipper positioning.

The second set of machinery is the rotating or swing mechanism, consisting of a circular track with reversible gear control. This is a major factor in the efficiency and adaptability of the shovel enabling it to face any direction for digging and dumping.

The third set is the lower works or power train. Propel mechanisms have become independent of drive shafting from the upper deck and incorporate separate propel motors and gearing for powering the tracks or crawlers.

Types of machines include:

- Quarry mining shovels which are mounted on twin crawlers and are suitable for most pit conditions. Tracks ensure stability and resistance to shock loads, when digging.
- Dobler machines which are a combination of basic crawler machine and digging action of a front end loader i.e. hydraulic bucket instead of a dipper.
- Hydraulic excavators which are adaptations of back hoes and have front loading or shovel features.

DRAGLINES
A drag-line is basically an excavator which casts a rope-hung bucket a considerable distance, collects the dug material by pulling the bucket towards itself on the ground with a pull rope, elevates the bucket, and dumps the material on a spoil bank, in a hopper or on a pile. Draglines are differentiated according to the mounting method which includes track mounted, crawler mounted and walking.

Large drag lines with walking propulsion mechanisms and capacities of up to 180 cubic metres are referred to as stripping machines. These are used to side cast overburden material such as open cast coal mining.

Draglines are suited to digging beneath their own level and require reasonable digging conditions or well fragmented material.
TRACTOR SHOVELS
These are basically two types of front end loader used in mining: track type and wheel loader. The track type is essentially an excavating tool offering stability and floatation characteristics. Track loaders are chosen against wheel loaders for conditions such as hard digging requirements, in soft footing or rough terrain where stability is required.

The wheel loader is a high speed loading tool which is basically designed for loading, and offers versatility and mobility. Development has been towards more durable machines and increase in size. Initially wheel loaders were intended only to handle materials either loose or for stockpile storage. These machines are finding increased use in primary loading of blasted rock and unconsolidated material in the face.

TRACTOR DOZER
The tractor dozer with an hydraulically operated dozing blade is available in a wide range of sizes, both crawler-tractor and four-wheel drive rubber tyred.

The bulldozer finds application in excavation, moving material, clean up, loading buried loaders, ripping of rock and many miscellaneous duties.

TRACTOR SCRAPERS
Scrapers used in surface mining are of two basic types: wheel tractor scraper and crawler tractor-towed units. The most common in use is the wheel-type machine, because of its greater range, speed and productivity. The towed type is used for small operations and short haul distances, in soft underfoot conditions and where use is short term or periodic. Wheel scrapers have their greatest application in overburden stripping. Less widespread is their use in loading and hauling mineral product to a processing plant. Scrapers are particularly important for construction of earth wall dams and roads.

Configuration of machines available are three axle types using a towing tractor, two axle types, tandem power, either two or three axle unit, and elevating scrapers.
CONTINUOUS EXCAVATORS
The continuous excavator simultaneously digs and discharges material. Machine types are:

- trenches and ditches
- conveyor loaders
- augers
- bucket line and hydraulic dredges
- bucket wheel and bucket chain excavators.

For dry surface mining, bucket wheels and chain excavators have application but only bucket wheels are considered here.

A bucket wheel excavator consists of a digging mechanism mounted on a slewing boom, which can be raised or lowered, on which is mounted a rotating vertical bucket wheel. As the rotating wheel is pressed into the material to be dug, the buckets cut, gather and discharge the material onto a conveyor belt where it is moved to the mined-materials transport system.
TRANSPORT HAULAGE (UNDERGROUND)

An important consideration in the efficient operation of an underground mine is the haulage system.

A determining factor between profit and loss is the quick removal of ore and waste from the working place to secondary and main-line haulage and so to the outside. Important, too, is the moving of supplies from the surface to the working faces so that the excavating process continues without undue interruption. Men need to be transported in a rapid but safe manner.

Transport within a mine may be divided into three phases:

- face transport
- subsidiary haulage
- main haulage.
In some instances two or the whole three phases may be combined e.g.

1. A face shovel loading a dump truck which delivers the product directly to the processing plant.

2. A load-haul-dump unit which loads from a working face and dumps onto a trunk conveyor or into an ore pass which feeds main haulage at a lower level.

Types of haulage within the three groups can be identified as:

**FACE TRANSPORT**
This includes:
- shuttle cars
- load-haul-dump unit
- chain conveyors at longwall faces in coal mines
- bridge conveyors or continuous miners.

**SUBSIDIARY HAULAGE**
These units bridge the gap between face haulage and main haulage systems. However it is usually more efficient to use as few modes of transport as possible because much of the time used in haulage systems involves loading and unloading points.

**SUBSIDIARY TRANSPORT SYSTEMS**
These include:
- load-haul-dump unit
- chain and belt conveyors
- battery or small diesel loco haul trains
- diesel tracks.

**MAIN HAULAGE**
These systems which involve major levels and comparatively long hauls, include:
- trunk conveyors
- haulage – diesel
  - diesel electric
  - electric trolley wire
- road haulage (trackless)

**FACE/HAULAGE UNITS**

**SHUTTLE CARS**
Shuttle cars are virtually mobile hoppers equipped with a flight chain conveyer. They are rubber-tyre mounted and may be powered by either diesel or electric motor. Initially electric shuttle cars were battery operated, but lacked capacity; cable reel cars with cable lengths of 200 metres carried on a self winding spool overcame this problem. Four-wheel steering is provided for small turning circles. The machine is capable of discharging to an elevated hopper. This elevating discharge allows the car to discharge into a mine car, onto a belt, or into the receiving hopper of a belt feeder.
Two shuttle cars are normally used to transport coal from a continuous miner or a coal loader to a belt conveyer system. Truck speed is of the order of 6 km per hour and load capacity is up to 10 tonnes.

LOAD HAUL DUMP UNITS
These units range from 1 to 10 cubic metres. The smaller units are commonly used for development work and tunneling where mullock-waste disposal is close to the development mining location.

CHAIN CONVEYERS
This type of conveyer is robust, simple and reliable; it is used in conjunction with longwall mining systems. It has the advantage that it may be used with inclination either in favour of or against the load. Its main disadvantage is the weight of the chain and fights which make power consumption heavy.

Armoured conveyers are developments from chain conveyers. They are similar but the structure is strong enough to support the cutting machine as it cuts coal from the longwall face. The conveyer structure is so constructed that it can flex in the horizontal plane, be advanced and be held continuously against the coal face as the machine removes successive layers from the face. Forward movement of the conveyer against the face is usually effected by hydraulic jacks attached to self-advancing chocks on the goaf/waste side of the conveyer.

SUBSIDIARY HAULAGE
Subsidiary haulage is alternatively known as gathering haulage, and for this purpose load-haul-dump units or chain conveyers could be used. A load-haul-dump unit could be used in a metal mine to gather ore from step draw points and deliver to an ore pass which in turn delivers to a mains level haulage system at a lower level. A chain conveyer could be used to deliver coal from a face conveyor to a main trunk conveyer.

BELT CONVEYERS
Belt conveyers, whether gathering or trunk hauling units, generally possess characteristics. The load carrying capacity of the belt is determined by:

- the amount by which the belt is troughed
- the belt speed
- the belt width.

Basic conveyer elements include:

- head section and drive unit;
- belt take-up section (manual, gravity, pneumatic or hydraulic)
- intermediate section; consisting of framing and belt idles used to guide and support the belts between head and tail end;
- tail section, consisting of tail pulley and supporting structure.

While belt conveyers are used extensively in coal mines where the material mined is broken into small lumps at the face and is not very abrasive, their use is limited in metalliferous mines where the material is usually large and abrasive.

In handling oversize coal and to facilitate rapid discharge of shuttle cars, feed breakers which receive the total load of the car and reduce oversize by horizontal mounted breakers are used. The breaker provides a controlled rate of discharge, to the belt conveyer.
BATTERY AND DIESEL LOCOMOTIVES

Where rail haulage is used, and where trucks cannot be loaded quickly, gathering locomotives may be used.

BATTERY LOCOMOTIVES

These locomotives use self-contained batteries and are available in the 1.5 to 12 tonne range with power capacities of 4 kW to 30 kW. The battery locomotive offers independent mobility, simplicity and inherent safety. These locomotives because of their limited capacity and range are used for small to medium haulage conditions. Haulage track grade should be downhill in favour of loaded trains and is usually 1 in 100. Batteries have a usable battery charge of about 8 hours' working capacity. Battery-charging stations are required where one battery is charged over 8 hours while another battery is used; once charged the initial battery is available for re-use.

DIESEL LOCOMOTIVES

These are available in a wide range of weights and power outputs. For underground haulage these usually range between 2 to 20 tonnes with power outputs of 20 kW to 150 kW. Internal combustion engines of the compression ignition type (diesel) may be used underground as long as the exhaust gas emissions are within permissible limits and adequate ventilation is provided.
TROLLEY LOCOMOTIVES
Trolley locomotives use an overhead power supply line and a pantograph arrangement. This type of locomotive can be multiple-coupled for heavy or large train mass. Trolley locomotives may incorporate batteries for use on sections without overhead supply such as loading points. They have wider application as main line haulage. A disadvantage is the sparking of collector on the overhead power line especially in coal mines.

DIESEL TRUCKS
Low profile diesel dump trucks are available in a wide range of units. These range from 5 tonne to 20 tonne units. Large units are used for out of mine haulage declines and adits. Trucks offer flexibility in haulage and have found wide acceptance with spiral ramp access.

MAIN HAULAGE

TRUNK CONVEYORS
Trunk conveyors may be developed to move tonnages of up to and in excess of 3000 tonnes per hour. Belts may be in a single flight or in series. Operating belts within mines include single flight coal handling installations some 8.8 km long. Where the material is of a suitable size and nature to be transported by belt conveyor and the supply to be moved is continuous, or near continuous, the trunk conveyor is an excellent means of main haulage.

Belts, however, are unsuitable for hauling big chunks which clog hoppers, damage the belt and are likely to fall off in transit.

RAIL HAULAGE SYSTEMS

MAIN LEVEL
Rail haulage systems are suited to relatively horizontal grades and medium to long hauls.

Diesel locomotives for underground train haulage utilize hydraulic-torque converter drive or hydrostatic pump and motor drive. Diesel locomotives of explosion proof design are available.

Trolley locomotives are available in 4 to 35 tonne units with power of 7 kW to 220 kW. They can be coupled to haul long trains both in and out of a mine.

For main level haulage, gathering haulage requirements should where possible, be eliminated so that the main train loads from chutes are delivered to the main tipping point near the shaft.

To this end truck types which allow loading under a chute with the train continuously moving, as well as moving while dumping at the tipple, are important. Continuous loading is possible because each truck is designed with a lip which overlaps the adjacent truck. Outputs of 3000 tonnes per shift are possible operating on one-way haul distances of 1000 metres.

ROAD HAULAGE (TRACKLESS HAULAGE UNDERGROUND)
The demand for higher productivity in underground mines, in part to offset diminishing ore grades, has resulted in the use of diesel equipment as well as larger physical dimensions of mine openings. Haul lengths considered economical for load-haul-dump units are of the order of 500 metres; trucks are often a viable alternative. Truck-ramp haulage competes with shaft hoisting to depths of up to 300 metres currently where:
(i) rock conditions permit the excavation of a large cross-section decline to accommodate the modified surface dump trucks;

(ii) production rates are below one million tones of ore per year. At greater depths and with sufficient ore, the operating cost benefits of shaft hoisting make it the preferred system.

Most units are articulated four, or six-wheel drive with end dump and low profile. Modified surface dump units are used successfully in some operations; these generally require higher heading and clearances at loading points especially where loaded by front-end loaders.

Capacity varies from 10 to 50 plus tonnes.

**LAYOUT OF A HAULAGE SYSTEM**

**BASIC REQUIREMENTS**

A well designed haulage system should have the following characteristics:

- safety
- traffic control
- gradients with the load
- uncongested loading areas
- tipping on the move
- mass determination on the move where weighting is necessary
- high quality track work and conditions
- good drainage
- adequate ventilation.

*Figure 13.13 Low profile EIMCO 980-T10 truck 5 m³ capacity, 10 tonnes rating*
Safety
Where possible the train should be driven with the locomotive at the front to reduce the reaction time between a hazard presenting itself and the driver reacting to the hazard.

Where the haulage way is used for a travelling way for men and materials, overhead trolley wires require protection to prevent accidental contact.

The grades of the haulage way, although usually designed to be with the load, should be matched to the braking power of the locomotive so that the train may be stopped within a reasonable distance in an emergency.

Where traffic control is not used, block lights need to be used so that two trains do not occupy the same section of track, especially when travelling in opposite directions.

Traffic Control
For efficiency of operation where several trains are operating in the same system a centralized control system is necessary. This may involve automated control as well as computer monitoring.

Gradient
A well designed haulage system has the grades arranged so that the train is always travelling down grade when fully loaded. The actual grade will depend on the type of train used.

The ideal gradient is such that the locomotive requires the same effort to pull the full train down the grade as it does to pull the empty train up grade. This is usually of the order of 2 to 3%.

Where the haulage system is a loop haulage there must be departures from this ideal gradient to obtain closures when laying out the haulage system.

Loading Areas
While trains may be loaded with overshot loaders, front-end loaders or by scrapers operating to mill holes above the train, efficient high production haulage systems are usually loaded from a surge bin which is often the ore pass or raise from the haulage level up to the stoping area.

Where trains are made up of trucks which cannot be loaded continuously, spotting of each truck is necessary. This may be accomplished by a brakeman who operates the chute and spots the train. Other methods include the driver using remote control for train spotting.

Mass Determination
This is usually done automatically by weightometers on surface conveyors. Underground systems may be used with automated systems to enable trains on the move to be weighed and recorded.

Track Work
Efficient rail transport requires a high standard of track work both to reduce rolling resistance and to prevent mishaps such as derailments. High train masses at speeds of 15 km per hour require large minimum radius of curvature of tracks and heavy rail section.

Drainage
Where a level layout is designed for downhill gradients, drainage is usually well catered for. Rapid deterioration of sleepers, ballast and track will occur where drainage is poor.
Ventilation
The ventilation of a haulage system must be adequate to provide for:

- an acceptable environment for the operating train crew;
- dilution of the noxious fumes from a diesel exhaust to an acceptable level where diesel equipment is used;
- reduction of dust concentrations to an acceptable level or preferably removal of dust to a return airway rather than allowing its re-entry to a work area.

The second ventilation requirement favours trolley locomotives as against main level diesel locomotives.

SURFACE HAULAGE
Surface mining transport/haulage may utilize a wide range of methods which include: railway, trucks, belt conveyors, slurry pipelines and other methods such as aerial ropeways.

RAILWAYS
Rail transportation is suitable for low gradient long haul of ore and waste. However where large elevation differences occur in or out of the pit track space requirements in layout are considerable.

Careful main-line track planning is essential so that relocation during the life of the mine is kept to a minimum. Because relocation can adversely affect production, only large mines offer opportunities for rail haulage.

Loading of in-pit trains is usually accomplished by shovel.

TRUCK HAULAGE
Open pit haulage by off-highway truck offers flexibility as well as the ability to handle adverse out-of-pit grades.

The most common haulage unit is the load on the back truck unit which has good gradability/traction characteristics.

Coal haulage units often utilize tractor-trailers of bottom dump design which offer manoeuvrability due to their articulation.

Off highway truck capacities range from 35 to 350 tonne units.

BELT CONVEYORS
Belt conveyors can effectively handle reasonable uniform-size material up inclines of up to 18 degrees. For use as in-pit transport the maximum lump size requires control. Belts have particular application to continuous excavating equipment such as bucket wheel excavators. Equipment especially designed, such as conveyor bridges, is used to handle waste stripped by bucket wheel excavators.
CHAPTER 13 LOADING AND HAULAGE (EXCAVATION AND TRANSPORT)

SLURRY PIPELINES
Pumping of solids in suspension has become a common and acceptable method of transporting and elevating solids. For in-pit transportation this usually consists of slurrying sand material (in the case of beach sands) and pumping the material to a concentrator plant. The hydraulic dredge utilizes a pump to elevate and transport the material either to placement discharge or concentrator plant.

Transportation of material ex concentration plant as well as coal by slurry pipe line has been a significant factor in establishing the economic accessibility of minerals. Savage River Mines (Tasmania) pump iron concentrate some 85 km at a rate of 2.25 million tones a year, over terrain where the costs of road or rail transport would be prohibitive.

OTHER SYSTEMS
Aerial tramways transport loads in carriers suspended from wire ropes forming the tracks, between fixed points, usually considerable distances apart. They are capable of spanning rivers or rough terrain. Aerial cableways have limited capacity and require considerable maintenance and periodic cable replacement.

FIXED LOAD OUTS (CHUTES AND DRAW POINTS UNDERGROUND)
The facilities provide the interface between sloping and haulage operations. Usually some storage is provided in the stope or in the ore pass leading to the chute or draw point.
GUILLOTINE CHUTES
These consist of a lever activated shutter or guillotine, however they are restricted to small mines. The action is readily fouled and is unsuitable for openings above 500 x 500 mm.

ARC GATES
These initially consisted of hand-operated lever arc-gates operated from an elevated position/leading stope above the haulage level. This provided some protection to the operator as well as allowing barring down of hung up material as overflows could occur when material ran.

AIR OPERATED ARC GATES
To provide better control of ore flow and positive cut off action to reduce overflows, pneumatic-air operated arc-gates were developed. The arc-gate at the lip provided the control of ore flow into the truck while the higher control gate throttled the overflow.

ARC GATE CHUTES WITH CHAIN CONTROLS
The upper control gate is replaced by a number of heavy anchor chains that rest against the ore at the bottom of the chute. These anchor chains can be lifted off the ore with an air cylinder allowing ore to flow.

ORE PASSES
Ore passes serve two functions:

- to connect the various tipples to the underground crushers and loading stations;
- to provide a surge capacity for material.

The first function is usually essential in metalliferous mines where extraction takes place on a number of levels, but where only one crusher station and loading station is practical.

This applies in particular where a single friction mine hoist is used.

Ore passes must have angles generally greater than 65 degrees to prevent hang ups which block the pass.

Passes with a circular section and with a diameter at least 3 times the size of the largest boulder expected are ideal. However passes are often rectangular with smaller cross sections.
Ore pass draw off control is achieved by chain curtain. This allows ore passes from succeeding levels to be interconnected, allowing a degree of broken ore storage other than in the stopes. This device consists of a chain curtain which effectively blocks off the pass but which can be raised to control the flow of ore, feeding the ore pass system, the underground crusher and hence the hoisting system.

TRANSPORTATION OF MEN UNDERGROUND

An important phase of underground haulage is the safe and efficient deployment of miners to their working places. It is important to reduce travel time especially where idle equipment represents extensive cost; it is also important to reduce non-productive effort of personnel.

Where vehicles are provided for the transport of men, their design depends on the physical characteristics of the openings. Such vehicles are required for:

- transportation of crews to and from the work face at the beginning and end of the shift;
- transportation of supervisors, inspectors and maintenance personnel during shift;
- evacuation of personnel or crew whenever required.

Units used can be either self-propelled or towed depending on whether they are trackless or rail mounted. For ramp-mines personnel/material carriers range from small diesel four-wheel drive units to low profile units.

Time lost due to travelling between mine entry and working place can be a significant portion of a miner's work day. The problem is greater in mines of horizontal beds than in a localized ore body mine. The profitability of a mine can be considerably affected by excessive time spent in travel.
CHAPTER 14 – Mine Hoisting

INTRODUCTION

The mine hoist provides transport via shaft conveyances for men and material into a mine and the hoisting out of broken ore and mullock (waste).

Mine shafts may be either vertical or inclined. A mine winding system comprises the following elements:

- power supply; winder - controls, motor, drum;
- ropes and rope attachments;
- conveyances;
- head frame and sky shaft;
- conveyance guides;
- input - output devices such as loading and tipping station;
- shaft.

The mine winding system can be a bottleneck between the underground mine and the surface mill. Correct selection of the right type of hoist is imperative. Accurate assessment of actual hoisting demands will enable shaft and winder installations to be designed to meet scheduled production demands. (See Fig. 14.1)

MINE HOIST TYPES

There are two basic types of mine hoists currently available: the drum hoist on which the hoist rope is actually stored during the hoisting cycle and the friction (Koepe) hoist, which merely passes the rope over the driving wheel during the hoisting process. A number of variations occur in both categories.
DRUM HOISTS

Drum hoists consist of a rope storage drum onto which the rope is wound in one or two layers. The rope required is that equal to the maximum hoisting distance plus sufficient rope for cutting and inspection at regular intervals and for holding to the drum.

SINGLE DRUM HOISTS

Single drum hoists can be equipped with a single rope. This imposes the greatest starting load on the motor of any of the winding systems. This condition is referred to as unbalanced, because the motor has to provide torque for the inertia of motor and gearing, rope and sheaves as well as the suspended load. (See Fig. 14.2)
Where a single drum incorporates two ropes with two conveyances or a conveyance and counter weight, the arrangement is referred to as conveyances in balance or counter-weighted. (See Figs. 14.3 and 14.4 (A))

**DOUBLE DRUM HOISTS AND DIVIDED DRUM HOISTS**

Double drum hoists consist of two rope storage drums and incorporate one or both drums clutched to enable multi-level winding. This winding system can be operated with two conveyances or a conveyance and counter weight. (See Figs. 14.4 (9) and 14.5)

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**Figure 14.3** Single drum hoist (equipped with 2 ropes – balanced)

**Figure 14.4a** Single drum configuration using end lift

**Figure 14.4b** Double drum configuration using side lift

Where multi-layer winding is necessary, the single drum hoist when used in balance must have a divider to allow separate storage of each rope. Where a counter balance is used the counter weight drum section can be of smaller diameter. (See Fig. 14.5)
CHAPTER 14 MINE HOISTING

14.4

CONICAL DRUM HOISTS
The torque required for a winder motor is related to the rope tension times the winding drum radius, therefore the smaller the drum radius the lower the motor torque required. The conical or cylinder conical drum hoist consists of two drums; the loaded conveyance at the bottom of the shaft is fed onto the smallest diameter of the drum. (See Fig. 14.6)

BLAIR HOISTS
Blair hoists are used where drum winders are preferred but single ropes are too large. Usually two headropes both off the same drum, which is divided to accommodate the two ropes are used per conveyance. (See Fig. 14.7)

FRICTION HOISTS
The basic principle of the (Koepe) friction hoist is that a hoist rope passing over a friction-type prime-mover-driven wheel has a tension differential between the point where the rope enters the wheel and the point where it leaves the wheel.

Friction hoists are designed to use a tail rope, which is attached to the undersides of the conveyances.

Friction hoists can be either single or multirope units.

The single rope friction hoist however is limited because of the single rope safety factor. The hoist can be used either balanced or unbalanced. (See Fig. 14.8)
Multirope friction hoists allow the use of large ore carrying skips with lower rope speeds with a number of small-diameter ropes. This allows smaller diameter friction drives to be used without the need for deflection sheaves. (See Figs. 14.9 and 14.10)

Koepe (friction) hoists may be either tower mounted or ground mounted. Fig. 14.11 shows a tower mounted friction hoist. Ground mounted hoists have a similar configuration to a drum winder (see Fig. 14.11) but the fleet angle to the head sheave must be zero and the sheaves in line.
Figure 14.9 Schematic arrangement of two rope friction hoist

Twin-rope ground mounted friction hoist

Figure 14.10 Schematic arrangement of multi-rope friction hoist

Four rope tower mounted friction hoist with deflectors
HOIST CHARACTERISTICS AND SELECTION BY USAGE

The choice of winding systems will depend upon:

- money available;
- the depth from which loads have to be hoisted;
- the loads to be hoisted;
- space available in the shaft;
- the availability of winders.

In general the winding system chosen will be the simplest that satisfies the particular winding duty. The more demanding the winding duty the more complex the winder may become. For instance, for hoisting from shallow depths where the load to be hoisted is small and trips infrequent, a single drum operating directly to the load is quite adequate, but with heavy loads hoisted from deep shafts, means have to be found to reduce the initial torque required for the hoist motor. The means available are generally:

a. the use of counter weight;
b. the use of tail ropes.

HOIST USAGE

A single drum hoist and friction hoist when operating in balance only have the ability to service one level. When one conveyance is at the surface the other is at the loading level.
To allow a friction hoist to service a deeper level the addition of new, longer ropes would be required. To allow the single-drum hoist to service a deeper level a new hoist would be required because of limited single layer rope storage.

To be able to service multiple levels in a shaft either the counterweighted friction hoist, counterweighted or unbalanced single drum hoists or double drum hoist with one or both drums being clutched, is required.

With counterweighted friction hoists and counter weighted single drum hoists the position of the counter weight in the shaft is irrelevant. With double drum single or double clutch hoists the relative suspended rope lengths wound on the drum can be changed by fixing the loose drum into locked position, disengaging the clutch and rotating the fixed drum to desired position.

The use of counterbalanced hoists - either single drum, double drum or friction winders - is best suited to servicing of mines with several levels.

For production hoisting the in-balance hoist has half the cycle time of a counter-balanced hoist.

**HOIST ROPES**

The hoist rope is the single most important component of the mechanical winding system. There are three basic types of hoisting ropes: round-strand, flattened-strand and locked coil, the difference being in their internal construction, mass per unit length and breaking strength.

Actual description of ropes is given under the following criteria:

- length
- nominal diameter
- minimum breaking load (aggregate or actual)
- Lang's lay or ordinary lay, right hand or left hand, preformed or non-preformed
- nominal tensile grade of wire
- wire: plain or galvanised
- material of core: fibre or wire.

**ROPE CONSTRUCTION**

**ROUND/STRAND ROPES**

A steel wire rope may be defined as a group of strands, usually in one layer twisted symmetrically and helically with a uniform pitch and direction around a central core of fibre or wire.

Each strand consists of a group of steel wires similarly twisted in one or more layers around a central fibre core or wire. (See Fig. 14.12)

Construction refers to the number of strands and the arrangement of wires per strand.

Lay is the manner of twisting the wires and the strands together. The two main types are:

(a) ordinary lay – where the wires appear to run parallel to the longitudinal axis of the rope. (Wires in the strand run in an opposite direction to the strands). This construction resists unlaying or untwisting of the rope.
(b) Lang’s lay – where the wires appear to run diagonally across the longitudinal axis of the rope. Both wires and strands have the same direction of twist. Lang’s lay distributes the wear on the wires and is more flexible but tends to untwist.

**FLATTENED STRAND – TRIANGULAR STRAND ROPES**
With this type of construction, wear is more evenly distributed over a greater external surface. They also possess improved abrasion resistance and greater resistance to heavy crushing loads. (See Fig. 14.13)

**LOCKED COIL ROPES**
Locked coil ropes are constructed with one or more outer layers of interlocking wires: as such, they possess no strand. The advantages of locked coil are:
- size for size they are of greater strength than stranded ropes in the same tensile grade;
- the smooth external surface gives greater resistance to wear by abrasion;
- they are virtually non-rotating;
- the elastic and permanent stretch is less than that of stranded ropes;
- internal cross cutting is reduced. (See Fig. 14.14)

**NON ROTATING ROPES**
Non-rotating or non-spinning ropes are produced by cross layering consecutive layers of strands within the rope to take advantage of counter winding one layer of strands against the other. These ropes may be round or flattened strand construction. (See Fig. 14.15) Balance ropes for friction winders often utilize a non-rotating rope with the external layers of coreless strands around a conventional stranded rope.

<table>
<thead>
<tr>
<th>Rope construction</th>
<th>Strand construction</th>
<th>Core</th>
<th>Dia. (mm)</th>
<th>Minimum breaking load kN</th>
</tr>
</thead>
<tbody>
<tr>
<td>Example 1</td>
<td>6 x 7</td>
<td>6 x 1</td>
<td>Fibre</td>
<td>26</td>
</tr>
<tr>
<td>2</td>
<td>6 x 19</td>
<td>9/9/1</td>
<td>Fibre</td>
<td>26</td>
</tr>
<tr>
<td>3</td>
<td>6 x 19</td>
<td>12/6 + 6F/1</td>
<td>Fibre</td>
<td>26</td>
</tr>
</tbody>
</table>

Ropes available in a range of diameters and tensile (wire) grades.

*Figure 14.12 Typical round strand ropes*
Rope construction | Strand construction | Core | Dia. (mm) | Minimum breaking load (kN)
--- | --- | --- | --- | ---
Example 4 | 6 x 8 | 7/8 | fibre | 26 | 383
5 | 6 x 9 | 8/9 | fibre | 26 | 383
6 | 6 x 22 | 9/12/8 | fibre | 26 | 383
7 | 6 x 25 | 12/12/8 | fibre | 26 | 414

Ropes available in a range of diameters and tensile (wire) grades

Figure 14.13 Flattened/triangular strand ropes

Locked Coil Hoist Ropes

Locked coil | Dia. (mm) | Minimum breaking load (kN)
--- | --- | ---
Available in a range of sizes and construction

Figure 14.14 Typical locked coil hoist ropes
GUIDE AND RUBBING ROPES

Steel ropes are used extensively for guiding conveyances in vertical shafts, being cheaper to install and maintain than fixed guides. They eliminate side impulses and vibrations which occur with fixed guides and cause fatigue of conveyance fittings and attachments. Negligible resistance to the flow of air can reduce ventilation costs.

Locked coil guide ropes with large section outer wires are almost universally used. They may be galvanized depending upon shaft conditions. (See Fig. 14.16)
MATCHING HOIST TYPE AND ROPE CONSTRUCTION

DRUM HOISTS
Various types may be used, but the final choice is largely one of economics and involves mainly shaft depth and size.

6-strand ropes, round and flattened strand, are satisfactory for all depths of shaft with fixed guides. With depths greater than 600 metres and rope guides they are unsuitable because they tend to twist or turn the conveyance. For multiple layering and where fleet angles are high, flattened strand ropes are superior. All ropes should be Lang’s lay.

A general construction criteria is for an outer wire diameter ratio of between 1000 - 1500 to 1 compared with the drum or head sheave diameter.

Multi-strand, non-rotating ropes may be used for sinking purposes but are unsuitable for multi-layer coiling because of their relatively low resistance to compressive forces.

Locked coil ropes have advantages in that they can be used on any depth of shaft with either fixed or rope guides and are virtually non-rotating. The cross section is compact and can withstand compressional forces.

Ratio of drum or head sheave diameter to rope diameter should be of the order of 100 : 1.

FRICITION HOISTS
Here it is important to avoid fleet angles otherwise rope rotation can occur on the driving drum.

6-strand ropes are suitable for depths of 750 metres, however torsional fatigue may be a problem. Flattened/triangular strand ropes are usual. With rope guides, alternate left and right hand lay ropes are employed to prevent conveyance twist.

Multi-strand non-rotating ropes are usable to most depths.

Locked coil ropes are suited to rope guide shafts and are widely used.

SAFETY FACTORS AND HOIST ROPES
Safety factors for Western Australian mines are specified by the Mines Regulations 15.56, for drum winders.

Ropes have to possess minimum safety factors of:

- 7.5 - 0.00 L (L in metres - depth of wind) when transporting men.
- 5.5 - 0.0003 L when transporting ore or materials.
- 4.5 exclusively rock hoisting.

For friction winding operations Regulation 15.67 specifies these minimum safety factors:

<table>
<thead>
<tr>
<th>Duty</th>
<th>Single rope</th>
<th>Two or three ropes</th>
<th>Four or more ropes</th>
<th>Duty</th>
</tr>
</thead>
<tbody>
<tr>
<td>Transporting of men</td>
<td>7.5</td>
<td>6.9</td>
<td>6.3</td>
<td>Transporting of men</td>
</tr>
<tr>
<td>Transporting ore materials</td>
<td>6.8</td>
<td>6.2</td>
<td>5.6</td>
<td>Transporting ore materials</td>
</tr>
<tr>
<td>Exclusively rock hoisting</td>
<td>6.3</td>
<td>5.7</td>
<td>5.1</td>
<td>Exclusively rock hoisting</td>
</tr>
<tr>
<td>Balance ropes</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>Balance ropes</td>
</tr>
</tbody>
</table>
ROPE MAINTENANCE
Ropes on drum winders are lubricated externally, however friction hoist ropes are not lubricated because of the likelihood of drive wheel slipping.

DETERIORATION OF HOIST ROPES
The major factors involved are wear, corrosion and fatigue.
Rope wear can occur from:
- badly fitting head sheave grooving;
- head sheaves of excessive mass hence inertia causing it to revolve at end of wind;
- too large a fleet angle;
- the rope rubbing on a fixture.
While this external wear can be either corrected or offset by using a rope with high abrasion resistance wear surface, internal wear can be significant.
Internal wear occurs from stress rubbing of inner wires: small diameter drums induce this.
Corrosion can occur when lubricant inside the rope is displaced and not replenished, allowing water to accumulate. If the shaft environment is particularly corrosive, galvanized wire in the rope may be necessary.
Fatigue cracks appear when the repeated stress is above the endurance stress of the wire.
Mines Regulations 15.61 require that the diameter of a drum or head sheave should be not less than 100 times the diameter of the rope for locked coil ropes, and 80 times for any other rope.
For friction hoists the driving sheave diameter has to be 100 times the diameter of the rope for locked coil ropes and 90 times for flattened strand ropes.
Regular inspections, both visual and electromagnetic are required to detect broken wires.
Corrosion can result in fatigue being generated from corrosion pitting, therefore adequate lubrication is necessary to avoid corrosion.

EXAMINATION AND TESTING OF ROPES
Mines Regulations 15.44 and 15.58 require periodic inspection of winding ropes. Ropes on drum winders are required to be recapped every six months; two metres from the conveyance end are cut off and sent to an approved testing station for destructive tensile testing.
Where the tensile test is 90% of the original then the rope must be removed.
For friction winders the service life specified by Mines Regulations is two years for the hoist rope and three years for balance ropes.
Non-destructive testing using electromagnetic methods is important in enabling ropes to be examined rapidly over their entire length, especially lock coil ropes and friction hoist ropes which can not be cropped and re-used. These methods can detect changes in steel area from losses due to wear and corrosion, while changes in steel permeability indicate possible broken wires.
In general, the time for removal of a rope is indicated by:

- a marked reduction in rope diameter (measured across the crowns of the strands);
- evidence of excessive abrasion on outside wires;
- broken outside wires;
- indications of severe corrosion.

Mines Regulations require the replacement of a friction hoist rope where more than six wires in any one section equal to the length of one external lay occur.

**SHAFT GUIDES**

Vertical shafts are equipped with either fixed or rope guides. Fixed guides are most commonly used especially for small shafts or shafts with close operating clearances. Rope guides are generally used for multi-rope hoisting where rope torque-twist is minimal.

Clearances between shaft sets and conveyances with fixed guides is about 150 mm depending upon size and shape of conveyance and the guides. With rope guides, clearances need to be of the order of 300 mm between conveyance and 200 mm between shaft and conveyance. Rope guides with a single rope hoisting system require non-rotating rope and attention to clearances.

Guide ropes are more economical than fixed guides and require less maintenance. However fixed guides at terminal or loading stations are required to take horizontal reactions.

Fixed guides consist of wood, steel rail, steel tube or fabricated top hat sections. Wood guides are used where conveyances require safety devices because the retarding or decelerating characteristics of safety dogs on wood are more reliable than on steel. Wood guides however are a fire hazard, wear readily and require maintenance of fittings and often lack smooth joints.

Steel guides using rail or tubular steel sections are common. Rail offers a smaller, harder wearing surface, but steel tube offers a rigid guide suited to high speed hoisting.

**SKIPS AND CAGES**

**CAGES**

The prime function of cages is the transport of men and materials to and from the working levels. In high production mines handling of large pieces of equipment is important. Cages used for lowering or raising persons must be covered and equipped with gates.

The size and number of decks on the cage depends on the size of operation and equipment being handled. Cages vary from single deck to double deck: men-carrying capacity varies from around 4 to 130 per deck.

Cages are normally required to be equipped with safety devices for protection in the event of rope failure. These safety devices are designed to operate on the timber guides.

**SKIPS**

Skips are shaft conveyances for hoisting ore or waste from a mine. The most significant feature is the method by which ore is discharged from the skip.

A skip design which was commonly used (with a capacity of up to 10 - 12 tonnes and an overturning dumping arrangement) is the Kimberley skip. This type of skip requires considerable sky-shaft travel and imposes severe loading in the head frame.
Skips now in use are almost all of the bottom-dump design which requires shorter travel distance to dump.

Bottom dump skips are of two basic types. In one the skip body does not move laterally with respect to the guides and only the door and door mechanism are articulated. The skip bottom is sloped at an angle sufficient to permit ore to flow freely after the door is opened. Skips of this type have been the Anaconda, Jeto and Rollamatic. A version of this type of skip using rope guides and or self-contained mechanical operating force is the DSL skip by SALA. Dumping is done when the skip comes to a halt.

In the second type of bottom dump skip, the door forms the bottom of the skip body. As the skip goes into the dump, the bottom of the skip body is pulled out over the receiving bin with the door pivoting downward to form a chute for discharging the ore directly into the bin. The Saunders skip is of this type.

**SKIP LOADERS**

Skip loading is often automated in conjunction with automated dumping and automated ore hoisting systems. Loading systems which involve measurement by load or by volume can be used.

Volumetric systems are used primarily for use of relatively uniform density and where occasional waste rock is of lesser density than the ore. Any condition where ore density variation can cause overload on the hoist necessitates the use of measurement by load system. Volume measurement is by cartridge or pocket with switching or gate control.

Measurement by load utilizes a load cell device supporting the pocket plus the ore. This allows a constant load irrespective of density with limit devices for over-fill.

**ROPE ATTACHMENTS**

Rope attachments (often referred to as cappels) enable the rope to be connected via a detaching hook and chain or other devices to the shaft conveyance. For drum winders Mines Regulations require the ropes to be recapped at intervals of six months with a minimum of 2 metres being sent to an approved destructive tensile testing station.

There are fundamentally four types of capping:

(a) bent back wire capping;
(b) white metal capping;
(c) interlocking wedge capping;
(d) gland type capping.

The first is not recommended for winding ropes while the other three types, if done with care, achieve a capping strength equivalent to the rope.

a. Bent back wire capping utilizes 'u' bolt type fasteners such as Bull dog Grips. These are not used unless specifically authorized for hoisting ore.

b. White metal capping uses a zinc-fitted rope socket. This involves spraying out the wires, cleaning, then pouring the white metal into the socket.

c. Interlocking wedge type capping utilizes a tapered frame incorporating inter-locking wedges held together by a series of bands. A safety block is attached to the rope end by means of white metal which would react against the wedges in the event of slipping. This type is common with lock coiled ropes. (See Fig. 14.17)
d. Gland type capping is a clamping device rather than a capping. The end of the rope is turned back upon itself over a grooved block of suitable radius and the short end of the rope clamped on to the gland casing. This type is known as a Continental Wedge Cappel.

CAGE AND SKIP ATTACHMENTS

Mines Regulations require that all attachments, including the cappel to the skip or cage are of approved design and construction and have been tested. The items have to be stamped with the safe working load and date and the parts have a prescribed maximum life period. Where coupling chains are used there must be a minimum of two with a combined factor of safety or not less than 20.

CAGE ARRESTORS AND DETACHING HOOKS

Mines Regulations require the provision of safety catches/arrestor devices on conveyances when transporting men to prevent a sudden fall in the event of a rope or hoist failure. This only applies to drum winder and is subject to exemption by approval where non-destructive rope testing and rope guides are in use.

Safety or arresting devices consist of gripper dogs, which, in the event of release of suspended load, lock onto the timber conveyance guides. Where installed these devices are required to be tested every two weeks.

Inclined shaft conveyances are generally not fitted with safety gripper devices because they do not incorporate guide timbers.

Figure 14.17 Interlocking wedge cappel
For friction hoists neither safety catches nor detaching hooks can be used but this is offset by using higher rope safety factors. Wedging type arrestors in the case of overwind are incorporated in the sky shaft.

Detaching hooks are required for drum hoisting installations. This device in conjunction with a suspension thimble allows a winding rope to be disconnected from a conveyance in the event of an overwind in the head frame, suspending the conveyance in the sky shaft. The detaching hook can be of a number of designs consisting of plates, shackle bolts and shearable copper pins. (See Fig. 14.18)

**OTHER SAFETY DEVICES**

Mine hoists need to be protected from travelling too far or too fast. Protection is required for personnel safety and to protect the hoist, head frame and shaft.

Overtravel at the shaft extremity either top or bottom incorporates the use of limit switches as well as safety devices such as detaching devices in the sky shaft.

Hoists are designed with fail safety braking so that in the case of power failure, braking is applied automatically.

Hoists which handle men as well as ore require man- safety overrides which prevent the cage or skip going into the dump position or into the loading pocket.

Protection from overspeed during the duty cycle is provided by a governing device known as a Lilly controller. This incorporates braking functions as well as regulating brake application in emergencies.

In order to prevent accidents in hoisting, all possible potential causes must be examined. Mines Regulations impose minimum standards and have detailed requirements on hoist braking functions.

**SIGNALLING**

Every person employed underground is required by Mines Regulations to be acquainted with the Code of Signals for the communication/control of shaft conveyances. Students should be familiar with the requirements and codes prescribed.
INCLINED SKIP HOISTING IN SURFACE MINING

Inclined skip hoisting has been used for open pit operations where pits are small, deep, and steep-sided, where rail or truck inclines are prohibitively long, steep, difficult to maintain or detrimental to the flexibility of the mining plan. Being a permanent or semi-permanent installation, inclined skip hoisting involves special attention to planning slope stability requirements. In general off-highway trucks provide flexibility and have displaced the use of skip-system installations.
CHAPTER 15 – Mine Services – Compressed Air

INTRODUCTION

Compressed air is used in mining operations as an energy source to drive such items as rock drills and air tools as well as having auxiliary uses.

Air is a mechanical mixture of a number of gases, each of which has different physical and chemical properties. The main ingredients of air are oxygen and nitrogen.

All matter consists of molecules which are in constant motion but held together by molecular forces. In a gas the molecules are relatively far apart and move freely about; a gas can therefore expand through space and disperse. The total volume of the molecules in a gas is very small in relation to the volume of the gas. A gas can therefore be compressed into a very small part of its original volume.

Air usually has a certain amount of water vapour associated with it which can account for anything up to four per cent of the mass of the mixture. The actual mass of air is dependent on its temperature and the pressure exerted on it. The pressure exerted on a gas consists of atmospheric pressure and of additional pressures caused by compressors or fans.

Pressure is defined as a force exerted on an area. The unit of force, defined as the product of mass and acceleration, is the newton (1 N = 1 kg m/sec²). The basic unit of pressure, the newton per square metre (N/m²), is called a pascal (Pa).

Atmospheric pressure at any particular location is equivalent to the force exerted by the atmosphere vertically above it and is consequently lower at high-lying places than at low-lying places. Atmospheric pressure at sea level is usually taken as 101.3 kPa, where 1 kPa = 1000 Pa.

GAS LAWS

Two basic laws are used to express the relationships of the variables in compressing a gas.

Boyle's law states that the volume occupied by the sample of gas varies inversely as the absolute pressure if the temperature remains constant i.e. $PV = C$

where $P$, $V$, $C$ Represents absolute pressure
      Represents volume
      Represents constant.

Hence the original condition can be related to the changed condition i.e. $P_0V_0 = P_1V_1 = C$

This is the basis of isothermic compression (no temperature change).

Charles' law state that under constant pressure the volume of a given mass of gas is proportional to its absolute temperature, i.e.

$$V_1 = \frac{T_1}{T_2}$$

where $T$ represents absolute temperature.
When a gas is compressed or expanded it is not usual for either the temperature or the pressure to be constant but for all three parameters, pressure, volume and temperature to vary. Hence, combining the two:

Boyle’s Law $PV = \text{constant}$

Charles’s Law $\frac{V}{T} = \text{constant}$

the combined equation being

$$\frac{PV}{T} = \text{constant R (gas constant)}$$

hence the relationship from one condition to another can be related.

$$\frac{P_1 V_1}{T_1} = \frac{P_2 V_2}{T_2}$$

This is known as the general gas law or the Characteristic Gas Equation. Using this formula the density of air can be determined at any temperature or pressure.

**ABSOLUTE PRESSURE AND ABSOLUTE TEMPERATURE**

Absolute pressure refers to the total pressure registered by a gauge plus the pressure of the atmosphere.

The pressure registered by most gauges is the difference between the pressure they are recording and the pressure of the atmosphere; this is referred to as the gauge pressure.

e.g. a gauge-pressure of 550 kPa would be an absolute pressure of $550 + 101$ (atmospheric pressure at sea level) = 651 kPa (absolute).

Absolute temperature refers to the Kelvin temperature scale which has its zero as the absolute zero. The Celsius temperature scale has reference points of $0^\circ \text{C}$ being the melting point of ice and $100^\circ \text{C}$ being the boiling point of water. These correspond with Kelvin temperatures of 273 K and 373 K, absolute zero being $-273^\circ \text{C}$ or zero Kelvin. It is necessary to use absolute temperature when using gas equations.

**COMPRESSION AND EXPANSION OF AIR**

**ISOTHERMATIC COMPRESSION AND EXPANSION**

The term isothermal means that there is no change in temperature during the compression or expansion of the molecules which constitute the air. If a gas is expanded or compressed under the conditions of Boyle’s law it is said to be isothermally expanded (or compressed).

**ADIABATIC COMPRESSION AND EXPANSION**

Where a gas is expanded (or compressed) with no loss or gain of heat it is said to be adiabatically expanded (or compressed). Although temperature may vary in the process of expansion, if the heat content of the gas remains the same before and after expansion the process is adiabatic.

Compression or expansion follows the formula:

$$P_1 V_1^k = P_2 V_2^k$$

where $k$ is the ratio of specific heats (adiabatic index).
While neither isothermal compression nor adiabatic compression is achieved in practice they serve as bases for comparison. Adiabatic compression is close to that attained in practice especially for positive displacement compressors.

The compression process tends to fall between isothermal and adiabatic conditions. The process cycle which is used is the polytropic one:

\[ PV^n = \text{constant} \]

where \( n \) is an experimental value which may be higher or lower than the adiabatic exponent \( k \). This process is often applied to dynamic compressor units.

INTERNAL ENERGY
The internal energy of a gas is the total heat energy stored per unit mass of gas because of the molecular motion and position in confinement. Motion of the molecules depends upon the temperature as measured by a thermometer. Internal energy is then fixed if there is no change in temperature while pressure and volume is varied. Energy derived from compressed air is due to the temperature change it undergoes when released i.e. expanding with temperature drop.

AIR COMPRESSORS

COMPRESSOR TECHNIQUE
Compressor processes are polytropic, which means that the temperature increases with the pressure ratio. With increasing temperature the work of compression increases. In order to limit the temperature rise and to improve the compression efficiency the compression is normally carried out in stages and the gas cooled between those stages.

STAGING
Multi-stage compression involves utilizing two or more steps in the compression process, operating in series. Multi-stage compression increases the volumetric efficiency as the pressure ratio over the first stage is decreased. With an increasing number of stages with perfect inter cooling the compression approaches the isotherm. While compression efficiency increases with multiple staging the compressor turns out to be more complicated and expensive. Therefore for each pressure level there exists an optimum number of stages for the intended use of the compressor.

INTERCOOLING
In order to dissipate compression heat, use is made of cooling systems such as cylinder cooling while cooling the air between compression stages is known as inter-cooling. A cooler is a heat exchange unit using air, water or oil as a heat absorber to remove excess heat.

COMPRESSOR DISPLACEMENT AND EFFICIENCIES
When examined with specific reference to reciprocating piston compressors the actual displacement of a compressor is less than the theoretical swept volume because of valve leakage, heating of the intake air, and clearance volume expansion.

The efficiencies of a compressor are expressed as:

- Compression efficiency, the ratio of theoretical power required to compress air to the actual power in the dry cylinder.
- Volumetric efficiency - the ratio of the actual free air output to the calculated volume.
• Mechanical efficiency - the ratio of air indicated power to the indicated power of the driving unit.

• Overall efficiency - the compression efficiency times the mechanical efficiency.

Actual compressor displacement is usually described as the volume displaced by the compressor per unit time (usually the first stage).

Capacity of a compressor is the actual volume rate of flow of gas compressed and delivered at the discharge point, referred to conditions of total temperature, total pressure and composition (humidity) prevailing at the inlet point. The units used are cubic metres per second or litres per second.

Rating of a compressor is then the conditions at which a unit can maintain capacity - volume of air at a stipulated pressure and designed speed.

Output on displacement is usually based on atmospheric pressure at sea level. Where a compressor is used at high altitude the diminished atmospheric pressure and lower air density alters the output performance. Increase in altitude means a greater compression is required to achieve a given pressure.

**COMPRESSOR TYPES**

There are two basic groups of compressors. The first is the displacement type where the pressure rise is obtained by enclosing a volume of gas in a confined space and the volume is reduced by mechanical action e.g. reciprocating piston compressor and rotary screw compressor. These exhibit intermittent flow characteristics. Second, dynamic compressors where the pressure rise is obtained by importing kinetic energy to a continuously flowing gas and converting this energy to pressure energy via a diffuser, e.g. ejector and axial compressors. These exhibit continuous flow characteristics.

Displacement types are the most common for compressed air supplies in mining operations.

**RECIPROCATING PISTON COMPRESSORS**

These are the oldest and most common type. These may consist of single or multi cylinder: those of multi-cylinder design can either be single-stage or multi-stage. Pistons can be either single acting i.e. compressing at the crown of the piston on the upstroke or double acting, compressing on the down stroke of the piston in the cylinder.

A wide range of configuration of piston layout and design is available in the reciprocating piston compressor.

Piston compressors are available both in lubricated and non-lubricated versions; this refers to the piston and piston ring and hence whether the air delivery is likely to carry oil or be oil-free.

**ROTARY COMPRESSORS**

Rotary compressors include types such as screw, vane and turbo-blower. The first is the most common.

Rotary screw compressors operate on the principle that air is drawn in by way of an intake port into the space between lobes of helical rotors (screw). The revolving rotors force the air into successively smaller inter-lobal spaces.
Compression is a single stage without pulsation. (See Figure 15.1)

This type of compressor though less efficient than reciprocating compressors offers the advantage of in-field exchange of the screw-compressor module effectively reducing downtime.

The vane type rotary compressor consists of an eccentric rotor in relation to the housing or cylinder. The vanes of the rotor slide in and out of the rotor on rotation. The space formed between two vanes diminishes during rotation towards the pressure chamber or outlet.

**COMPRESSOR CONFIGURATIONS**

Broadly this is taken to be the type of mounting and includes portable compressors and stationary compressors.

Portable compressors are usually almost of the rotary type utilizing the rotary screw principle. Most units are diesel powered. Units range from outputs of:

- 40 litres/second (80 cfm)
- to 600 litres/second (1200 cfm)

These units have application to in pit operations for powering crawler mounted drills and down-the-hole hammer machines.

Stationary compressors may be either piston or screw depending upon the project requirements. Usually this incorporates compressor packages which include all necessary components such as after cooler and moisture separator, starter, sound attenuation (silencing), inlet filtration and control equipment.

Compressor packs are available in piston design from 1 to 1040 litres/second output and screw design from 38 to 560 litres/second. Much larger units are available but require foundations.

**COMPRESSED AIR CIRCUITS**

In examining the utilization aspects of compressed air it is appropriate to consider the distribution network within an underground mine which normally includes the following.

![Figure 15.1 Working principle of a screw compressor (simplified)](image)

**COMPRESSOR**

Air is drawn through a filter to remove dust and particulate matter and is compressed, often involving two stages with intercooling. The final discharge pressure is usually of the order of 700 to 900 kPa. From the compressor it passes to a receiver often via an aftercooler. This unit increases capacity because cooler air will occupy a smaller volume.
RECEIVER
The function of the receiver is to store a reserve of air to meet fluctuations in demand for air. In the receiver also the majority of the moisture that will condense, condenses out and provision for a bleed tap at the bottom of the receiver is necessary. An over-pressure safety valve is required by the Machinery Act and Regulations.

AIR MAIN
From the receiver the air passes to the air main. This is a large diameter pipe which conveys the air from the surface to underground. The diameter of the pipe will vary according to the volume required, however it is usually not less than 200 mm in diameter. A large diameter main is necessary to avoid friction losses. Also the mains serve as a secondary receiver.

WATER IN COMPRESSED AIR LINES
There is always a certain amount of water vapour or humidity in the air. When air is compressed, cooled, stored transmitted and used it will experience changes in volume, pressure, temperature and velocity. These changes in turn will vary its ability to hold moisture. As a result, at certain points in the compressed air circuit, condensation will occur which may be blown along by the movement of the air. This water, if allowed to accumulate, may cause damage to the pipeline as well as to the machines utilizing the compressed air.

WATER TRAPS
These consist of receptacles usually consisting of a section of pipe below a Tee piece with a draw-off tap at the bottom. Water traps allow moisture to accumulate and be blown off periodically.

LEVEL AIR MAINS
Air mains from the shaft main are tapped off at the various levels by level mains. These are usually of 100 to 150 mm diameter and incorporate valves and connections allowing draw off. Compressed air to stopes is usually via 50 mm pipe with final connections to machines via reinforced hose which offers flexibility of movement.

QUANTITY OF AIR REQUIRED
The quantity of air required varies from hour to hour. Therefore it is necessary to provide a number of compressors often of different sizes which will meet the mine demand (machines) as well as leakage. Installed rating must have excess capacity i.e. maximum demand as well as standby units.

TRANSMISSION AND FRICTION LOSSES
In theory the pressure at any point in a compressed air network should equal the pressure at any other point. However this is not so in practice because friction losses occur. Friction losses are proportional to the square of the velocity of the air in the pipe. For this reason the velocity of the air in the pipe should be kept low and this is achieved by having a large diameter pipe where large quantities of air are transmitted. However an air line will actually carry more air at a high pressure than it can carry at a lower pressure. This is explained by the different densities of air at low and high pressure.
Permissible friction - pressure loss in transmission - air lines should be of the order of:

- long permanent mains 1 kPa per 100 metres
- level branch mains 2 kPa per 100 metres
- temporary/hoses 40 kPa per 100 metres.

Using this type of pressure drop criteria the pressure drop between a compressor station and underground drill using compressed air with 4000 m of mains (loss 40 kPa), 1000 m of branch pipes (loss 20 kPa) and 50 m of hose (loss 20 kPa) totals 80 kPa.

Where high losses occur and with increased flows, the resultant pressure loss may seriously affect machine effectiveness even to the extent of making drilling impossible.

TRANSMISSION LINES
Most compressed air lines are galvanized metal steel and should exhibit smooth internal walls to avoid friction losses.

The forms of pipe joints used include sockets, unions, flange joints and victualic (semi circular clamp and shouldered pipe with ring seal) joints. Compressed air hoses use snap-on type couplings. The most common joints for large pipes are victualic joints which incorporate pipes with a small lip over which a rubber ring is placed. The two pipe lips and rubber are held in place by a clamp which prevents the pipes from moving apart.

LEAKS
One of the major losses in compressed air circuits can be in leaks. A small leak in a pipeline may seem insignificant but on a medium sized mine the losses incurred by small leaks can cost thousands of dollars a year in loss of compressed air and loss of pressure.

The leakage through a hole will consume air continuously whereas an air tool on average operates 40 to 50 per cent of the time. A leak therefore consumes about twice the power of a tool with the same momentary air consumption.

Table 15.1 shows hole size, leakage and power loss.

COMPRESSED AIR SAFETY
All forms of energy sources are potentially dangerous. Compressed air is intrinsically less dangerous than most other forms of energy transmission but does have peculiar hazards of its own. Most accidents happen with air hoses. If an air hose becomes disconnected from a machine it will discharge air violently and whip about endangering workmen in the vicinity. Large air hoses should be equipped with chained couplings. Safety couplings with automatic shut off may be used where hose rupture may occur and endanger personnel.

Compressed air mains should be installed out of the way and be adequately secured, especially where trackless equipment is used. Should an air pipe be ruptured, both pipe ends can react violently, endangering personnel. Adequate low resistance isolation valves need to be provided.
CHAPTER 15 MINE SERVICES – COMPRESSED AIR

Table 15.1 Air leakage losses

<table>
<thead>
<tr>
<th>Hole diameter</th>
<th>Air leakage at 600 kPa</th>
<th>Power required for compression</th>
</tr>
</thead>
<tbody>
<tr>
<td>True size</td>
<td>mm</td>
<td>l/s</td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>1</td>
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<tr>
<td></td>
<td>3</td>
<td>10</td>
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<tr>
<td></td>
<td>5</td>
<td>27</td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>105</td>
</tr>
</tbody>
</table>

USES OF COMPRESSED AIR AND CONSUMPTION
Compressed air is used in mining operations as a source of power to drive such items of equipment as rock drills, air tools, air hoists, pneumatic cylinders, water lifts and auxiliary uses such as atomizing water jets (water blast) for allaying dust. (See Fig. 15.2)

CONSUMPTION
Percussion drills utilize compressed air at pressures of 700 kPa and have consumption rates of 35 litres per second for small hand held machines to 250 litres per second for large mounted machines.

Mechanical boggers have air consumptions which range from 100 to 250 litres per second at an operating pressure of 600 kPa.

ADVANTAGES AND DISADVANTAGES OF COMPRESSED AIR

ADVANTAGES
• The air after use is a valuable addition to the mine ventilation system. This can account for up to 5% of the air entering a mine and is generally released at the mine work face.
• The compressed air system is robust and allows ready installation of pipe lines without skilled trades.

• Compressed air is a primary power source for the pneumatic rock drill, although hydraulic rock drills have made significant contributions especially for jumbo mounted drill rigs. For soft rocks electric rotary drills work satisfactorily.

• Compressed air is extremely adaptable. It can be used for both rotary and reciprocating motions and has been used for every type of small motor underground, including fans, pumps, scrapers, hoists as well as compressed air/generator lights.

• Compressed air equipment is generally of robust construction and has low maintenance requirements, provided adequate lubrication is provided. Air line lubricators (oil bottles) are used to provide controlled lubrication to machines.

DISADVANTAGES

• Compressed air is an expensive form of energy source costing far more than electric or diesel power.

• Large machines using compressed air require heavy construction because of forces encountered in air chambers and connections.

• Compressed air machines are noisy, some excessively so. Noise is a recognized industrial hazard and should be minimized.

• Compressed air lines are prone to leaks and are expensive in energy losses.

• Use of compressed air and air line leaks tend to keep dust suspended in the mine atmosphere.

• Excessive lubrication and water in the air can cause fogging at the work face when pneumatic rock drills are used.
CHAPTER 16 – Mine Services – Water, Lighting and Power

INTRODUCTION

Water is used in mines for drinking as well as for dust suppression - mainly by means of a hole flushing medium for drilling. Reticulated water in mines is usually potable (water fit for human consumption) however distinction is made by reference to non-drinkable water as individual water where used in an operating process.

The factors which determine the suitability of industrial water for use include its acidity (or alkalinity), the concentration of solids in solution and in suspension, its pressure and its temperature.

Acidity of water is usually measured and indicated by the pH value (active acidity) and is in terms of the concentration of hydrogen ions. Water with a pH factor of 7.0 is neutral, while a factor of less than 7 indicates acidity and more than 7 indicates alkalinity. The scale ranges from 0 to 14 and is logarithmic.

Water with a pH of less than 5 is considered unsuitable for use or for pumping and lime is usually added to bring the pH to a value of between 5 and 6. If the pH is too high, furring up of pipes occurs. Concentration of solids in solution indicates whether build up in pipes will occur and is expressed in parts per thousand or milligrams per litre. Salts in solution by analysis are referred to generally in terms of total dissolved solids.

The concentration of solids in suspension in dirty water can be determined by filtering and weighing the solids obtained. If the concentration is too high the water is likely to be abrasive and will cause pump and pipe wear.

The pressure of water determines whether sufficient water will be available through the water tube in a rock drill for hole flushing or through an atomizer or nozzle for laying dust.

Water classified for drinking purposes is that which passes certain standards with regard to concentrations of bacteria. Water sterilization is usually achieved by means of chlorination. This process consists of adding doses of chlorine to kill any bacteria present.

RETICULATED WATER IN MINES

Water piped underground is usually delivered in stages because of the effective accumulation of pressure head with depth. One metre of water (H20) develops a pressure of 9.8 kPa. So as to limit the water pressure on the levels as well as at the work face for rock drilling, working pressures of around 600 kPa are aimed at. This then is equivalent to 60 metres of water head and is attained by the use of breakdown tanks generally arranged from level to level i.e. an upper level breakdown tank supplying a lower level.

UNDERGROUND MINE DRAINAGE AND DEWATERING

The effect of water on deep excavation is of importance because in some instances it constitutes a direct hazard as well as incurring considerable cost in handling and can result in reduction of operation efficiency. Working wet ground may incur less productive methods, less efficient equipment and more expensive explosives. Wet conditions can cause higher maintenance on equipment and often less efficient labour.
OBJECTIVES OF WATER CONTROL

Water in mines must be controlled to permit efficient safe working with the least side effects. The most common control is by pumping. However significant savings can be achieved by methods preventing or controlling inflows. These methods include grouting, surface diversion and other drainage controls.

PUMPING FROM WELLS AND MINES COMPARED

Pumping from water wells for water supply is done on a design criterion of producing a desired quantity of water with the least drawdown. Where additional water is required another well is placed at such a distance that the new well does not appreciably affect the drawdown of existing wells.

Dewatering wells are close spaced so that cones of exhaustion of the dewatered ground overlay amply. This is usually done rapidly without interruptions. Drawdown is usually maintained until a shaft can be sunk and lined or until permanent pumping facilities within the mine are installed.

A conventionally sunk mine shaft or mine (both open pit and underground) is like a large well. Openings below the water table allow water to flow in at a rate determined by the permeability of the rock and the size of the opening.

Pumping from an accessible place in a mine has similar characteristics to dewatering through wells, however sumps and water clarification can be provided and pumps of larger capacity and higher heads can be used with readily available access for servicing.

MINE WATER SOURCES

The occurrence of water in mines can be attributed to a number of sources including inrushes from flooding or bodies of water, flaw from rock permeability and introduced water such as mine hydraulic fill and drilling water.

Inrushes of water can be from sources such as from the sea, a lake, river, swamp or wet cover as well as cavernous carbonate rocks, isolated water pockets, fault conduits and old flooded mine workings. Many of these may be directly related to high rates of rainfall and surface flooding.

Flows from rock permeability can be from the inherent porosity of the rock as well as secondary fractures. The water encountered from this source is generally underground or ground-water. Openings below the water table encounter such water inflows which can be relatively fresh or highly mineralized. This water comes mainly from percolating rainfall. In some instances inflows can be from magmatic water or from water originally deposited on formation of the rock strata.

Introduced water can come wholly or partly from man-made sources. These include flooding from adjacent mines, drainage from hydraulic fill and drilling water. Open boreholes in a shaft or mine opening can also cause sudden but high inflows of water.

EFFECTS OF MINE WATER

The effects of water on a mine can not be overlooked; some are related to cost while others are potential hazards.

Direct effects of mine water are both dangerous and costly. Major inrushes or failure to handle inflows can interrupt production and cause damage to installations and equipment with potential danger to personnel. Recovery of a flooded mine is usually slow and expensive and may be uneconomic. There may be some benefit when the water removed is used in the surface treatment process or for residue disposal.
The indirect effects of water in a mine include corrosion and deterioration of shaft installations, especially hoist ropes. Worker efficiency is reduced under wet conditions by the wearing of wet gear. Equipment and handling systems perform less efficiently, e.g. wheeled vehicles develop wheel spin and road surfaces deteriorate. Maintenance of equipment increases due to loss of lubrication and wear can increase with the ingress of water and abrasive slush. Electrical hazards can increase because of wet conditions. Weak ground susceptible to deterioration may fail due to constant wetting. Presence of excessive water increases humidity, decreasing the possible effectiveness of the ventilation system. Water resistant explosives may be necessary, e.g. A.N./F.O. blasting agent is susceptible to deterioration under wet conditions. Excess water in handling causes fines to be washed out, increasing cleanup of sumps, accessways. Likewise mud as a contaminant can affect the marketability of a mine product.

Indirect effects outside the mine include additional or high moisture content in the mine product incurring transport, treatment and handling costs especially for those materials which need processing in a dry state, also those which may freeze when exposed to low temperatures. Discharge of mine water at the surface may cause pollution of surface waterways e.g. acid mine water from the decomposition of sulphides particularly pyrite. Drawdown of ground-water can adversely affect water supply wells both in output and quality of water. Drawdown can contribute to surface subsidences due to the collapse of subterranean openings and solution cavities.

Mine water may only constitute a problem during shaft sinking when passing through permeable water-bearing beds. Where mines encounter water bearing strata the water inflow generally persists over the life of the mine and will eventually flood the mine on cessation of pumping.

**MEANS OF CONTROL**

Means of control are related to circumstances and objectives and involve investigation and testing prior to mining and need to be pursued as long as costs and hazards can be justified. While some controls are applicable in an emergency for example, to protect a mine flood from an inrush of water or to recover a mine, controls are most effective when used in conjunction with one another rather than individually.

Control starts with avoidance of potential problems. Rivers are significant in location of shaft in relation to topographical features and their effect on likely inflow from flash floods. Where fissures are likely to open up or ground subsidence with potential inrushes of water is likely to occur, pillars should be left; this applies particularly to workings beneath bodies of water.

Surface works include the diversion and drainage of surface and near-surface water and the reduction of direct inflow or likely recharge of inflow and ground-water flows. Intakes from permeable beds may be plugged with impermeable material. Shallow surface water can be intercepted by dewatering wells. Water which enters shafts can often be collected by water rings.

Where shafts are likely to intercept water bearing ground impervious linings are introduced. These include pre-grouting from the surface - which involves the introduction of concrete through test drill holes around the shaft location. Another approach is to freeze the water bearing ground in advance of the shaft, sink the shaft, set the lining and then thaw. Large diameter bored holes for shafts with drilling mud support may be applicable and casing is then set.
Permeability of a rock mass can be reduced by grouting with cement slurry where ground is heavily water bearing. Solution channels may be plugged or grouted by intersection with drill holes.

In some operations, actual dewatering of a mine lay operation can be done by workings or by a drainage adit, either of which may be applicable to open pits.

Exploration holes should be cased off or cemented or plugged rather than left open with possible inflows from surface flooding.

Deep ground-water may be handled by deep well pumps from the surface. In some instances these may be set from underground openings or shaft bottoms. Advance drain holes drilled in advance of mine openings as well as into faults and shear or wet areas of the ore body are also used.

Pumping systems for handling water include those which have pumps of abrasion-resistant design. Generally pumps need to be corrosion-resistant, with protection from galvanitic corrosion at pipe joints. Removal of solids may be accomplished by use of sumps and dams for settling and some treatment of water to reduce sealing and pipe corrosion.

Monitoring of water conditions includes the use of test holes, ground-water surveys and other practices including advance probing.

Emergency provisions for control of inrushes include extra water-handling capacity, doors or bulkheads to protect pump stations or the use of submersible or deep well units. Procedures adopted in an emergency include removal of personnel and the isolation of sections of the mine by waterproof bulkheads.

**PUMPING**

Pumping is the most common means of water control. In some instances the mass of water pumped from a mine is greater than the mass of rock hoisted.

Pumping offers advantages where:

- dewatering is generally the most practical way to sink shafts in saturated rock masses;
- dewatering/lowering of the water table adds to the stability of the rock mass;
- the recovered water is needed in the treatment process;
- inflow is small and pumping may be preferable to seeking an alternative.

**TYPES OF PUMPS**

There are three main types of pumps used for handling water:

- reciprocating
- centrifugal
- positive action rotary pumps.

**RECIPROCATING PUMPS**

Reciprocating pumps are of the positive displacement type usually incorporating a piston in tight fit within a cylinder. This type can be single or double acting. A reciprocating design not requiring a tight fit between piston and cylinder but using a packing seal is the plunger or ram pump. Reciprocating pumps produce a pulsating flow due to the opening and closing of inlet and outlet valves and motion of the piston.
Inherent problems with positive displacement pumps of this type arise from a blocked line. Full pressure can burst the delivery line or cause failure of pump or drive components. Water hammer can occur from the rapid opening and closing of valves.

While capable of high pressures, discharge rates are kept low to reduce wear as dirty or abrasive material in the water can be a problem. Settling and clarification dams prior to pumping are generally required.

A piston pump especially designed to handle solids in suspension (abrasive - dirty mine water) is the Mitsubishi-Mars Reciprocating pump. This uses an oil/water interface chamber, such that the pump causes displacement of oil without the water/solids entering the cylinder.

**CENTRIFUGAL PUMPS**

Centrifugal pumps predominate in mine pumping. These units provide a continuous non-pulsating flow under normal conditions. A rotating impeller imparts pressure to the pumped fluid. Pump designs are of several types. To achieve high head (pressure) multi-stage pumps using impellers pumping in series are used.

Centrifugal pumps exhibit the following relationships of capacity, discharge pressure and power to impeller speed:

- Capacity or rate of discharge varies with the peripheral speed of the impeller.
- Maximum discharge pressure varies with the square of the peripheral speed.
- Power input varies with the cube of the peripheral speed.

Actual heads (pressure) handled can be of the order of up to 2000 metres.

Positive action rotary pumps are of special purpose design such as the Mone unit which has a rubber stator in the form of a double internal helix and a single helical rotor which rolls in the stator with slightly eccentric motion. Positive action rotary pumps have relatively restricted usage for mine dewatering.

**COMMONLY USED PUMPS**

While the major duty of mine dewatering is generally handled by a pump station installation equipped with electrically driven centrifugal pumps, pumping duties in or about a mine employ a range of other units.

These include compressed air operated diaphragm units suitable for handling muddy and abrasive slurries such as decline and winze drilling water.

Shaft dewatering may be handled by submersible units which can pump by flexible discharge line to settling/storage dam. The water from the dam is handled by a more permanent larger capacity/high head pump possibly running only intermittently.

Sump dewatering can be done effectively by means of an air lift which exhibits low efficiency but is of simple construction and is only used infrequently where the water handled is limited.

**PUMPING INSTALLATIONS**

The need to enable pump installations to handle likely inflows with complete reliability means the provision of a standby unit or units. Where one unit is capable of handling all the water, a second unit is often provided as a spare. For multiple pump installations a smaller proportion of units can be provided as spares. Where only brief interruptions in pumping can be permitted, duplicates of all essential items must be provided.
Pump station provision objectives include:

- Sufficient units inclusive of reserve capacity;
- Provision for protection from flooding of both pumps and electrics;
- Sufficient dam storage and settling sumps;
- Automated pumping controls.

**ENERGY REQUIRED FOR PUMPING WATER**

The work done in changing the potential energy of a mass due to a change in its elevation, is equal to the mass times the change in elevation (vertical height) times the pull of gravity ‘g’, as measured by the acceleration of a freely falling body.

\[
\text{Mass} \times \text{distance} \times \text{acceleration} = kg \times m \times ms^{-2}
\]

\[
= (kg \times m \times s^{-2}) \ m
\]

\[
= N \times m
\]

\[
= J
\]

\[
\text{Work} = \text{kg/s} \times \text{height} \times
\]

\[
= L/s \times m \times 9.8 \ ms^{-2} (1 \ L \ H_2O = 1 \ kg)
\]

\[
\text{Power} = \text{work/second (Joules/second)}
\]

Example: Water is pumped a vertical height of 1000 metres at a rate of 1 litre per second.

\[
\text{Power required} = 1 \times 1000 \times 9.8
\]

\[
= 9800 \ W \ or \ 9.8 \ kW
\]

However when water is pumped in through a pipe, a certain amount of pressure is also required to overcome the frictional resistance of the pipe. The pressure loss depends upon the size, straightness and smoothness of the pipe and the volume of velocity of the water. This can be calculated but is generally taken from pipe friction tables. Frictional pressure loss is equated to additional head (height). Static head is the difference in elevation between pump impellor intake and pipe discharge, friction head is the additional head in metres due to friction loss. The total head is then used to calculate the theoretical power required. Actual motor power is obtained by incorporating the pump efficiency which is usually of the order of 60 to 80 per cent.

Large pipe sizes offer lower frictional losses, however a balance is necessary between friction loss/operational cost as against pipe cost.

Rising mains pipe work in shafts requires adequate mounting as well as being designed to withstand operational pressures and should provide for relative ease of replacement.

**SPECIAL PROVISIONS FOR WATER CONTROL**

**EXPLORATION HOLES**

Exploration holes need to be plugged or cemented off in some areas to prevent the migration of water.
Location of surface drilled exploration hole collar, if suspected of conveying water to rock strata or even when exposed underground, at best can be difficult unless cased and marked.

Relatively dry mines have encountered high inflows from diamond drill holes in development headings and stopes.

PILOT HOLES
Wherever there may be a sudden inflow of water from water bearing strata or old workings, pilot holes should be drilled in advance of the development heading. Mine Regulations require that any position of a working likely to approach within three metres of a heading and known to be flooded must be dewatered.

GROUTING
Grouting is conducted mainly to facilitate the sinking of shafts or tunnels within weak - wet ground so as to avoid inrushes as well as reduce delays and to assist in the placement of concrete linings. It is rarely used to enable stopping: where extensive wet ground occurs and is likely to be encountered during stopping advanced dewatering by means of bore hold or drainage headings is usually adopted.

Grouting is also used to plug conduits which are responsible for inflows into mines as well as sealing around water bulkheads and underground plugs to prevent mine flooding. The grout may consist of a wide variety of substances ranging from cement mixtures to chemical grouts. Clays and mixtures of bridging material such as sand, saw dust and shredded plastic are also used.

The most significant application of grout is in the practice of pre-grouting of deep shafts, in particular in South Africa. This involves drilling of diamond drill holes, monitoring the ground formation and grouting accordingly. This enables shaft sinking to proceed relatively undelayed.

Where excessively wet - weak (saturated) formations occur, shafts must be sunk though freezing of the ground may need to be resorted to.

Water flows through rock pores and fissures may be monitored by the introduction of dyes, e.g. fluorescein. This may be necessary where water within a mine may seep out to contaminate other water sources. A significant factor is often to attempt to identify possible sources of inflow. This may be done through bore holes, but large bodies of water pose particular problems.

Where mining operations are conducted under large bodies of water whether lakes, seas or reservoirs, legislation in general will specify a minimum solid coverage, the type of mine opening and the stopping method. This particularly applies to coal mining which covers an extensive lateral area with possible subsidence where high extraction is achieved.

UNDERGROUND BULKHEADS AND PLUGS
Deep mines subject to possible flooding utilize a series of bulkheads for the purpose of compartmentalizing mines in the case of flooding. This is a particular practice in South Africa. A major inrush of water at West Driefontein mine occurred in 1968. The complete flooding of the mine was prevented by emergency installation of plugs. Water pressures withstood by these being of the order of 6 mPa.

The installation of plugs and bulkheads required investigation into site location especially with regard to competent ground. Plugs need to be of sufficient length to have moderate pressure gradients over their length. All fittings within the structure such as flanged joints etc. need to be able to withstand the pressure encountered. Failure or major leakage can
lead to subsequent failure of the structure. Grouting is usually conducted around a structure to reduce leakage and strengthen the rock mass.

Plugs with control valves enable controlled rates of water to be handled by the installed pumping equipment without the danger of flooding.

**WATER IN SURFACE MINES**

Water in this situation is not likely to be as spectacular a problem as in underground operations, however it is still an important factor in costs and output of a pit. Maintenance of in-pit equipment generally increases in wet muddy conditions. Advanced dewatering increases the stability of pit walls below the normal static water table level.

In open pits pumping equipment may require protection from blasting yet be capable of handling wide fluctuations of inflow. Rainfall run off increases with increasing depth - expanding area of an open pit. Water can affect the stability of pit slopes appreciably.

**CONTROL MEASURES**

Pits which have stepped multi-level benches above the water table may require drainage ditches and sump pumps at the bottom of the pit. Water occurrence is mainly rainfall and is usually seasonal.

Pits of moderate inflow in general require provision for surface water diversion as well as inpit pumping installations. A relatively dry pit reduces maintenance, aids efficiency and adds to wall stability. Pit bottom pumps are used generally in conjunction with bore holes.

Pits which encounter excessive wet conditions require installation of extensive advanced dewatering wells/bores.

Lake bed deposits and river placers can often be best handled by wet mining methods such as suction cutter dredges or the like where ground hardness is suitable - e.g. loose or unconsolidated material.

**PUMPS AND DRAINAGE IN PIT**

Where possible, in-pit drainage should be away from the working faces. This may mean the use of drainage ditches in conjunction with sumps. Pumping is then done from a small sump at the face (advancing drop cut). The types of pump used include small submersible ones discharging from the lowest advancing face to the nearest sump. Sumps are equipped with skid mounted units or pontoon/floating units which are controlled automatically.

**ADVANCED DEWATERING**

Advanced dewatering is often the only practical way of handling large volumes of water likely to be encountered by an open pit, e.g. surface lignite mines. Submersible bore hole pumps are used for this purpose.

**ENVIRONMENT AND LEGAL ASPECTS OF GROUND-WATER CONTROL**

There are a number of aspects in mine dewatering which need to be borne in mind with reference to environmental impact and usage of water. These include:

- The effect of pumping on the use of ground water by others. Advanced dewatering can effectively dry out a wide region of ground-water.

- Effect of any discharged water either from surface operations or underground - in particular with relation to quality and quantity. Such factors as salt or dissolved
mineral content, pH (acidity), and turbidity are potential problems where discharge is into natural water courses.

- The possible encroachment of undesirable water by uncontrolled dewatering.
- Possible effect of subsidence due to dewatering - especially the collapse of solution passages.

An overall detailed analysis of the possible implications and consequences of a dewatering and water disposal problem should be closely investigated by mine management personnel.

**LIGHTING**

The ability of a person to see what he needs to see in order to do his job safely and efficiently, is determined by three general classes of variables:

- individual differences,
- the nature of the visual task,
- illumination.

Individuals differ with respect to visual acuity (sharpness of vision), colour blindness and various other aspects. In underground operations one factor is dark adaptation, the process of change which occurs in the eyes when the environment changes from bright to dark. During this change workers are prone to accidents.

Visual discrimination can be improved by providing colour contrast or brightness contrast and by avoiding glare.

Illumination is a measure of the brightness of the light falling on an object. It depends on the intensity of the light source, the distance from the source and the angle of impact of the light on the object.

The terms and definitions which are used for light sources, intensity and lighting effects are:

- candela is the unit used to define the luminous intensity of a light source; it supersedes the old unit defined as a candle;
- luminous flux (unit lumen) is the light energy emitted per second within a unit solid angle by a uniform point source of given luminous intensity;
- illumination is defined as the luminous flux reaching a surface perpendicularly per unit area and is expressed as lumens/metre$^2$, termed lux.

Lighting specifications are then made in terms of illumination or lumens/m$^2$. Adequate lighting is within the range of 050 to 0200 lumens/m$^2$ or lux.

The four most common types of electric light are:

- Filament lamps, also called incandescent lamps because their filaments (usually of Tungsten) are heated to incandescence, i.e. until they are glowing with heat. Only about 10% of the electricity used by the lamp is converted into visible light.
- Fluorescent lamps consist of gas fitted tubes with heated electrodes in each end. The tubes contain a little mercury and argon at low pressure, which emits invisible ultra-violet light when an electric current is flowing. The inner surface of the tube is coated with a fluorescent powder which produces light of a desired colour when exposed to
ultra-violet radiation. Fluorescent lamps are much more efficient than filament lamps and last longer, however switching on and off reduces their burning life.

- Mercury vapour lamps produce light from the passage of an electric current through mercury vapour. They are used for tower flood lighting especially in open pits for night operation.
- Sodium vapour lamps have very high efficiency but the light is almost monochromatic yellow, so that all objects are seen as yellow or various shades of grey to black and it is impossible to distinguish any colour. Crosswalk lighting is often of this type.

The purpose of lighting is obviously to promote safety and efficiency.

UNDERGROUND LIGHTING
Each miner carries his own cap lamp because working places involving blasting and rock handling make permanent installations impractical.

Cap lamps are rated in terms of Mean Spherical Luminous Intensity and must have a rating greater than 1.8 after approximately 8 hours, i.e. the equivalent of 23 lumens. Initial light output is usually of the order of 36 lumens.

Note that the unit lux is the unit of illumination and is equal to a flux of 1 lumen on an area of 1 m$^2$. Illumination on a surface decreases as the square of the distance from the light source. Therefore a cap lamp providing a useful source of light may provide a relatively low standard of illumination of around 5 lux compared to daylight in excess of 100 000 lux.

CAP LAMP – ELECTRIC SAFETY LAMP
This consists of a belt-carried lead acid battery of 4 volts in a polycarbonate case, a cord connected head-piece which carries a main light bulb and an auxiliary bulb in case of main bulb failure. Overall weight of the unit is 2.3 kg. The unit is capable of providing light over an 8 hour shift. At the end of a shift the lamp is returned to a charging frame where it is automatically recharged over a 12 to 13 hour period and is then available for use the next day.

FIXED LIGHTING UNDERGROUND
Fixed lighting is generally provided by incandescent lights or fluorescent lighting at shaft stations (plats) and permanent installation points such as ore pass train discharge points and shaft loading stations.

High levels of illumination of isolated locations should be avoided because it can create dangerous conditions. A gradual grading of light from the surface to the work face is recommended. Fluorescent lighting can provide a relatively glare-free lighting. To assist in improving light on plats and main travelling ways, walls are often spray painted. This improves reflection but also allows a degree of monitoring of any ground movement or slabbing of rock from backs or walls.

MOBILE EQUIPMENT
Trackless vehicles are required to have effective lights which provide lighting for both forward and reverse travel and which adequately indicate the width of the vehicle.

Underground locomotives require headlights as well as a tail light where more than one track is used.
LIGHTING – SURFACE MINES

Mines Regulations state that all working places in a quarry worked at night require illumination. Likewise all mobile equipment is required to be provided with adequate lighting.

In pit lighting is generally provided by tower mounted flood lighting. Where operations are remote or isolated the light tower may be powered by a small diesel generator.

Large excavators are often equipped with clusters of flood lights such that a single light failure does not adversely affect efficiency of vision of the operator or create hazardous situations.

Where reticulated power is available catenary overhead wires may be run from which road way or road junction lighting may be provided.

Particular care is required in placement of lighting in pits in order to avoid glare as well as aiming at an acceptable gradation from well illuminated areas to those of low illumination so that operators, especially truck drivers’ eyes can adapt.

POWER

Electricity provides perhaps the most important power source in mines both underground and surface.

The State Energy Commission in Western Australia has control of electrical installations under the legislated powers of the Mines Electrical Inspector for electrical equipment operated in mines and quarries.

Each mining operation is required to have a qualified Electrical Supervisor who is responsible for the electrical installations of that operation.

Mines Regulations require that a plan of operating mines be kept which shows the position of all permanent electrical equipment and fixed cables.

Power is provided to the mine site in general by overhead high voltage transmission lines.

Power is taken underground by armoured cable either by means of the shaft service way, decline opening or in some circumstances via a bore hole. Approved transformers and switchgear are provided so that power is available at the working voltage for the operating equipment.

Equipment in underground mines using electric power includes: ventilation fans, dewatering pumps, electric locomotives or battery locomotive chargers, welders, underground crushers if necessary, and compressors and refrigeration units.

Special provisions are required for electrical equipment in coal mines so that it is flame proof. Equipment in underground coal mines operating by electricity includes continuous miners, trailer cable-operated shuttle cars and conveyors. Auger drills used for drilling out coal faces are normally electrically operated.

Power for open pits and quarries is often provided by overhead lines direct into the pit. Transformer sub stations and switch gear are provided with major in pit equipment being provided by trailing cable.

These cables provide power to pumps, power-shovels and lights. Adequate protection from tracked and wheeled vehicles is necessary to avoid cable damage and hazards from electrical shock.
Non-captive equipment, i.e. not dependent upon trailing cables, offers greater flexibility but involves higher operation costs. Increases in diesel fuel prices have aggravated this situation recently.

Where there is reticulated electric power and electric shot firing is used, special precautions need to be taken to avoid possible premature initiation of electric detonators/explosives by stray electrical currents.
CHAPTER 17 – Mine Ventilation – Mine Air and Contaminants

INTRODUCTION

Air is required in a mine for four main reasons: to supply oxygen for breathing, to remove heat, and to dilute and remove dust and gases.

Ventilation is the control of air movement, its amount and its direction. Along with simultaneous control of quality and temperature - humidity of the air ventilation often involves air conditioning to maintain prescribed standards for the mine environment.

In mine ventilation air is coursed through the mine workings and openings and for auxiliary purposes, through vent tubing and ducts. In general, ventilation is provided by large fans although air flow may result from natural conditions.

As mines increase in depth, size and complexity, demands on the ventilation system to maintain standards of environment quality also increase.

Ventilation is auxiliary to the primary objective of mineral extraction but is perhaps the most essential of the functions attendant on mining. As such it is necessary for the preservation of human life and for the conduct of underground mining operations.

CONSTITUENTS OF AIR

Air is a mechanical mixture of several gases and water existing in vapour form (0.1 to 3% by volume).

Natural pure dry air composition:

<table>
<thead>
<tr>
<th></th>
<th>By Volume %</th>
<th>By Mass %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nitrogen</td>
<td>78.09</td>
<td>75.53</td>
</tr>
<tr>
<td>Oxygen</td>
<td>20.95</td>
<td>23.14</td>
</tr>
<tr>
<td>Carbon dioxide</td>
<td>0.03</td>
<td>0.04</td>
</tr>
<tr>
<td>Argon and other rare gases</td>
<td>0.93</td>
<td>1.28</td>
</tr>
</tbody>
</table>

For ventilation purposes involving quality control it is customary to assume dry air to consist of:

<table>
<thead>
<tr>
<th></th>
<th>Volume Basis</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen</td>
<td>21%</td>
</tr>
<tr>
<td>Nitrogen</td>
<td>79%</td>
</tr>
</tbody>
</table>

Air usually has a certain amount of water vapour associated with it, with a space of one cubic metre containing one cubic metre of air and one cubic metre of water vapour (a mixture).

The actual mass of a specific gas or vapour contained in a space of one cubic metre is dependent on its temperature and on the pressure exerted on it. The pressure exerted on a
gas consists of atmospheric pressure and of additional pressures such as those caused by fans and compressors.

**ATMOSPHERIC PRESSURE**

Pressure is force exerted on an area. The unit of force, defined as the product of mass and acceleration, is the newton (1 N = 1 kg m/s²). The basic unit of pressure is the newton per square metre (N/m²) termed the pascal (Pa).

Atmospheric pressure at any particular point is equivalent to the weight of the atmosphere vertically above that point and it is consequently lower at high altitude than at low altitude. Atmospheric pressure is usually taken as 101.3 kPa which is the equivalent to 760 mm of mercury at sea level.

Barometric or atmospheric pressure is measured by a barometer of either mercury or aneroid type. The first is a sealed tube in excess of 760 mm fitted and inverted with the open bottom in an open bowl of mercury (See Fig. 17.1). The aneroid type consists of a vacuum chamber to which is attached a dial indicator mechanism. Changes in pressure cause fluctuation in mercury level in the case of the mercury barometer (1 mm = 0.133 kPa) or distortion of the vacuum chamber of the aneroid which is then read off on a calibrated dial.

**TEMPERATURE SCALES**

The temperature scale in the S. I. system is called the Celsius scale. It is based on the boiling and freezing points of pure water at normal sea level pressure.

- Freezing point of pure water 0° C
- Boiling point of pure water 100° C

The corresponding absolute temperature scale is that known as the Kelvin scale. This has its zero point corresponding to –273° C which is called the absolute zero temperature. The corresponding scale temperatures are shown in Fig. 17.2.
HUMAN RESPIRATION

The importance of supplying clean air with adequate oxygen content in the ventilation of mines lies in the sustenance of human life. The human respiratory system requires the supply of oxygen, and release of carbon dioxide in the breathing process. The carbon dioxide liberated must be controlled. The rate and volume of respiration and hence consumption of oxygen increases with the physical activity of the individual.

Refer to Table 17.1 which relates rate and volume of respiration and hence consumption of oxygen to physical activity of the individual. Bear in mind that the respiratory capacity of a human (the volume of air inhaled or exhaled) is many times the actual volume of oxygen consumed per inhalation. Exhaled air contains approximately the following:

<table>
<thead>
<tr>
<th></th>
<th>By Volume</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen</td>
<td>16%</td>
</tr>
<tr>
<td>Nitrogen</td>
<td>79%</td>
</tr>
<tr>
<td>Carbon dioxide</td>
<td>5%</td>
</tr>
</tbody>
</table>

In Table 17.1 the respiratory quotient is the ratio of carbon dioxide expelled to the oxygen consumed.

OXYGEN DEPLETION

The depletion of oxygen may be caused by

- the removal of oxygen or
- a combination of processes.

The processes at work underground which cause oxygen depletion involve removal by absorption, adsorption and oxidation. Ground water may absorb oxygen, coal may occlude oxygen by adsorption and sulfide minerals may remove it by oxidation.

Dilution, the worst cause of oxygen depletion, occurs when a foreign gas, usually one of the strata gases, is released into the mine atmosphere. This reduces the effective concentration of oxygen present and may constitute a hazard in its own right.
Combined processes of oxygen depletion which consume oxygen and introduce an impurity in its place are significant.

<table>
<thead>
<tr>
<th>Activity</th>
<th>Respiratory Rate per min.</th>
<th>Air Inhaled per Respiration litres</th>
<th>Air Inhaled litres/min.</th>
<th>Oxygen Consumed litres/minute</th>
<th>Respiratory Quotient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rest</td>
<td>12 – 18</td>
<td>0.4 – 0.7</td>
<td>5 – 12</td>
<td>0.3</td>
<td>0.75</td>
</tr>
<tr>
<td>Moderate</td>
<td>30</td>
<td>1.5 – 2.0</td>
<td>45 – 60</td>
<td>2.0</td>
<td>0.9</td>
</tr>
<tr>
<td>Very vigorous</td>
<td>40</td>
<td>2.5</td>
<td>100</td>
<td>2.8</td>
<td>1.0</td>
</tr>
</tbody>
</table>

*Table 17.1 Human respiration oxygen and air inhalation rates*

These include oxidation processes of internal combustion engines, burning and slow combustion of timber or coal, which remove oxygen and liberate carbon dioxide and other gases. Human respiration also consumes oxygen and releases carbon dioxide.

**Effects of oxygen depletion involve the following observed physiological reactions:**

<table>
<thead>
<tr>
<th>Oxygen Content</th>
<th>Effect (varies slightly with the individual)</th>
</tr>
</thead>
<tbody>
<tr>
<td>17%</td>
<td>Faster, deeper breathing</td>
</tr>
<tr>
<td>15%</td>
<td>Dizziness, buzzing in ears, rapid heart beat</td>
</tr>
<tr>
<td>13%</td>
<td>May lose consciousness if exposure prolonged</td>
</tr>
<tr>
<td>9%</td>
<td>Fainting; unconscious</td>
</tr>
<tr>
<td>7%</td>
<td>Life endangered</td>
</tr>
<tr>
<td>6%</td>
<td>Death</td>
</tr>
</tbody>
</table>

The permissible minimum oxygen content of mine air specified by Western Australian Mines Regulation Act Regulations is 20%.

The quantity of air required can be determined from the information in Table 17.1 and from the minimum oxygen content permissible. This will of course be the minimum quantity or air required for the breathing process.

Two conditions may be used to arrive at a figure:

(a) The oxygen content being diluted to the recommended minimum level and
(b) the carbon dioxide content to rise to the recommended threshold level.

Case (a) Minimum oxygen level. Mines Regulations (Metalliferous) sets a level of 20% Oxygen as a minimum.

Determine the quantity of ventilating air required (Q) in cubic metres.

The required $O_2$ for vigorous activity (Table 17.1) is 2.8 litres/minute.

The balance equation is:

\[
\text{Amount of } O_2 \text{ in intake air} - \text{Amount of } O_2 \text{ for breathing} = \text{Amount of } O_2 \text{ in exhaust air}
\]

\[
0.21Q - 2.8 = 0.20Q
\]
\[ \frac{2.8}{0.21 - 0.20} = 280 \text{ litres/minute} \]
\[ = 0.28 \text{ m}^3/\text{minute} \]

Case (b) Maximum \( \text{CO}_2 \) content of 5000 ppm (0.5\%) as per Mines Regulations (Metalliferous). The respiratory quotient being 1, the quantity of exhaled \( \text{CO}_2 \) Case (b) Maximum \( \text{CO}_2 \) is \( (1)(2.8) = 2.8 \text{ litres/min} \).

The balance equation is:
\[
\text{Amount of } \text{CO}_2 \text{ in intake air} + \text{Amount } \text{CO}_2 \text{ from breathing} = \text{Amount } \text{CO}_2 \text{ in exhaust air}
\]
\[
Q = \frac{2.8}{0.005 - 0.0003} = 596 \text{ litres/minute}
\]
\[
Q = 0.60 \text{ m}^3/\text{minute}
\]

**AIRBORNE CONTAMINANTS**

The maintaining of air within desired limits of purity is termed quality control. This is accomplished in the mine atmosphere primarily through ventilation.

Any undesirable substance not normally present in air or present in excessive amounts is termed a contaminant. Vitiation (depriving air of its quality) may occur where impurities find their way into the air.

The contaminants which may be encountered are:

- gaseous, non-particulate contaminants e.g. gases or vapour;
- liquid particulate contaminants e.g. mists, fogs;
- solid particulate contaminants e.g. dusts, fumes, smoke and organisms.

Gases are the most common non-particulate contaminant of underground air. Mists and water vapour fogs are nuisances. The most common particulate contaminant and the worst environment problem in mines is dust. It is dust (particulate) and gas (non particulate) contaminants that focus the most attention in ventilation.

**MINE GASES**

While a wide variety of gases may occur in mines, only a few are commonly encountered as contaminants (see Table 17.2). All except oxygen can be considered as impurities, while nitrogen and carbon dioxide are so regarded when they are present in quantities exceeding those in normal air.

The gases with no distinguishing physical properties perceptible to the human senses (odour, colour, or taste) are most difficult to detect. This includes oxygen, nitrogen, carbon dioxide, carbon monoxide and methane, all relatively common, as well as hydrogen and radon.

Gaseous contaminants are hazardous mainly because of their physiological effect on humans. A gas may be asphyxiating, toxic or radioactive. Methane and hydrogen have little physiological effect but are dangerous to life and property because of their explosiveness. Carbon monoxide is toxic as well as explosive.
The colloquial terms for gaseous mixtures are:

<table>
<thead>
<tr>
<th>Term</th>
<th>Composition Mixture</th>
<th>Effect</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fire damp</td>
<td>CH₄ (methane) + air</td>
<td>Explosive</td>
</tr>
<tr>
<td>Black damp</td>
<td>CO₂ (Carbon monoxide) + air</td>
<td>Asphyxiating</td>
</tr>
<tr>
<td>After damp</td>
<td>CO (Carbon monoxide) + CH₄ (methane)</td>
<td>Toxic</td>
</tr>
<tr>
<td>White damp</td>
<td>CO (Carbon monoxide + air</td>
<td>Toxic</td>
</tr>
<tr>
<td>Stink damp</td>
<td>H₂S (Hydrogen sulfide + air</td>
<td>Toxic</td>
</tr>
</tbody>
</table>

**SOURCES OF GASEOUS CONTAMINANTS**

The most prolific sources of gases underground are:

**NATURAL**

Strata gas formed or trapped in the mineral deposit and released when mine workings are excavated. Methane in coal seams is the most common. Release rates fluctuate with barometric pressure, exposure area of workings, mining rate and air flow.

**MAN-MADE GASES**

These are generally produced by combustion processes such as blasting, internal combustion engines, fires and chemical reactions such as human breathing and hydrogen produced from charging of storage batteries.

The most common mine gas is methane (a strata gas) which is common with high rank coals. Because of its explosive nature adequate measures need to be taken, in monitoring the level of concentration in the mine air, withdrawal of miners and turning off of machinery when the level reaches 1.5%.

**GAS SAMPLING**

The extent of the gas hazard in mines can be evaluated only by adequate sampling of the atmosphere. For rapid, on the spot, approximate determinations, portable samplers called detectors are used. These take instantaneous grab (snap) samples and are used to detect the presence, identity and approximate amount of contaminating gas present. In general most are suited to a single gas. Continuous gas samplers called monitors are used in gassy mines which will sound an alarm or switch off machinery when the gas threshold (maximum allowable concentration) is exceeded. Complete and exact determinations of the nature and amount of impurities present can be made by laboratory analysis of gas samples collected underground.

**DETECTORS**

Detectors are the most common sampling device relied upon in mines to indicate gas.

An open flame such as a candle has been used in mines known to contain non-explosive contaminants. The extinguishing of the flame indicates a deficiency of oxygen (16 – 17° oxygen concentration or less). However, the identity of the contaminating gas or actual amount of oxygen present cannot be determined.

Safety lamps are the standard method of detecting methane in coal mines as well as being an indicator of oxygen depletion. The flame safety lamp was developed in 1815 by Sir Humphrey Davy, who demonstrated that a flame contained within suitable gauzes or
plates could not ignite any explosive mixture outside the gauzes because of the rapid conduction of heat by the gauzes.

The standard type of safety lamp (See Fig. 17.3) used in testing for gas, is held in the general body of the air. The height of the gas cap on the flame is an approximate measure of the methane content of the air.

<table>
<thead>
<tr>
<th>Name</th>
<th>Physical properties</th>
<th>Hazardous Effects</th>
<th>Source</th>
<th>Harmful Effects</th>
<th>Detection Point</th>
<th>Max. Allowable Concentration</th>
<th>Fatal Point Concent.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen</td>
<td>Odourless colourless tasteless floatable</td>
<td>non toxic</td>
<td>Normal air</td>
<td>Asphyxiating</td>
<td>Breathing, strata</td>
<td>0.005</td>
<td>-</td>
</tr>
<tr>
<td>Nitrogen</td>
<td>Odourless colourless tasteless floatable</td>
<td>Non toxic</td>
<td>Normal air</td>
<td>Asphyxiating</td>
<td>Breathing, strata</td>
<td>0.005</td>
<td>0.03 (12.74 explosive)</td>
</tr>
<tr>
<td>Carbon Dioxide</td>
<td>Odourless colourless tasteless floatable</td>
<td>Toxic</td>
<td>Normal air</td>
<td>Explosive</td>
<td>Blast, I.C. engines, incomplete combustion</td>
<td>0.005</td>
<td>0.1 (0.46 explosive)</td>
</tr>
<tr>
<td>Hydrogen Sulfide</td>
<td>Odourless colourless tasteless floatable</td>
<td>Toxic</td>
<td>Normal air</td>
<td>Explosive</td>
<td>Blast, I.C. engines, incomplete combustion</td>
<td>0.0005</td>
<td>0.1 (0.46 explosive)</td>
</tr>
<tr>
<td>Methane</td>
<td>Odourless colourless tasteless floatable</td>
<td>Irritant</td>
<td>Coal seams</td>
<td>Asphyxiating</td>
<td>Safety lamp detector</td>
<td>0.005</td>
<td>0.005</td>
</tr>
<tr>
<td>Nitrogen Oxide</td>
<td>Odourless colourless tasteless floatable</td>
<td>Irritant</td>
<td>Strata</td>
<td>Toxic</td>
<td>Oxidation I.C. engines</td>
<td>0.005</td>
<td>0.005</td>
</tr>
<tr>
<td>Sulfur Dioxide</td>
<td>Odourless colourless tasteless floatable</td>
<td>Irritant</td>
<td>Strata</td>
<td>Toxic</td>
<td>I.C. engines</td>
<td>0.005</td>
<td>0.005</td>
</tr>
<tr>
<td>Radon</td>
<td>Odourless colourless tasteless floatable</td>
<td>Radioactive</td>
<td>Strata</td>
<td>Toxic</td>
<td>I.C. engines</td>
<td>0.005</td>
<td>-</td>
</tr>
</tbody>
</table>

Table 17.2 Characteristics of mine gases
In the absence of sufficient oxygen the flame of the safety lamp will become dim, flicker and eventually go out, while in the presence of an inflammable gas a secondary flame or cap will form above the normal flame. Usually the cap is faint and hazy and can only be seen if the normal flame is reduced to a pinpoint and all other light is excluded. The height of the cap is proportional to the percentage of inflammable gas present in the air. As a reference the cap is fully formed in the shape of an equilateral triangle when the concentration of methane is 2.5%. Roughly the observations made are as per Table 17.3.

The modified or probe flame safety lamp has an inlet nipple on the lamp to allow it to be used with an aspirator bulb and probe in testing samples taken from the roof or other places difficult of access.

The Garforth Flame Safety Lamp feature is that samples collected in a bulb can be injected into the lamp through an annulus round the wick. This allows the lamp to be held in a
convenient position for observation of the gas cap. High concentrations of gas from the bulb will not extinguish the flame.

Where gas concentrations of between 1 and 2 per cent are detected in the general body of mine air it is mandatory that men be withdrawn and equipment shut down.

Methane detectors apart from the safety lamp are of two types. Both use electricity which is supplied by batteries. The resistance type of methanometer consists of a balanced electrical bridge containing 1000 platinum filaments which are heated by a battery. The one coated by a catalyst has the inflammable gas pass over it causing oxidation and altering the electrical resistance. The resultant imbalance of the bridge is indicated on a galvanometer which gives a measure of the percentage of inflammable gas present in the air. An example is the M.S.A. G.P. (General Purpose) Detector (Methanometer).

The interference type of methanometer or interferometer works on the principle that the passage of light through a gas is affected by the refractive index of the gas. Light beam passing through a gas like methane is thrown out of phase with another beam passing through pure air, giving rise to black interference lines which provide a measure of the percentage of methane in the air.

Continuous gas samples (monitors) are generally stationary units and only sample ore location. However remote monitoring systems have been developed for complete underground operations. These instruments may either sound an alarm or cut off the electrical supply to a machine at a pre-set percentage of methane concentration.

In general the air being sampled reaches the unit sensing elements by diffusion.

An example of an alarm type detector is the English Electric Automatic Firedamp Detector and Alarm. This unit indicates continuously (8 - 9 hours) the concentration of firedamp, and automatically shows a red alarm light when a set percentage of firedamp is reached.

<table>
<thead>
<tr>
<th>Percentage Methane</th>
<th>Observed Effect</th>
</tr>
</thead>
<tbody>
<tr>
<td>2</td>
<td>A pale blue cap or hale, caused by the burning of methane is seen around the top edge of the flame.</td>
</tr>
<tr>
<td>3 – 5</td>
<td>The cap becomes bigger and bigger as the percentage of methane is increased.</td>
</tr>
<tr>
<td>5 – 15</td>
<td>The flame spreads throughout the mixture causing an explosion. Explosive concentration of 9½% is the most violent.*</td>
</tr>
<tr>
<td>20</td>
<td>The gas burns around the flame only.</td>
</tr>
<tr>
<td>25</td>
<td>The flame is extinguished.</td>
</tr>
</tbody>
</table>

* May be within the lamp or external depending upon lamp type and condition.

*Table 17.3 Safety lamp– Observed effect of methane*
POISONOUS (OR TOXIC) GASES
These are not easily dealt with and should not occur in the mine environment. When they are present men must be withdrawn until ventilation has cleared the mine. Maximum allowable concentrations are specified by various statutory regulations.

Mines Regulation Act and Regulations (Western Australia) specify the following standards:
Not more than:

5000 parts per million of carbon dioxide \( \text{C}_2 \)
50 parts per million of carbon monoxide \( \text{CO} \)
10 parts per million of hydrogen sulfide \( \text{H}_2 \)
5 parts per million of nitrogen dioxide \( \text{N}_2 \)
5 parts per million of sulfur dioxide \( \text{S}_2 \)
5 parts per million of aldehydes

The most common method of detecting poisonous gases underground is a chemical method, based on chromatography (colorimetric detector). This type of instrument consists of two parts:

- A hand pump or aspirator which will suck up a measured quantity of air;
- A sealed glass tube containing a chemical which changes colour when in the presence of the particular gas being tested.

The operation of this type of instrument is relatively simple. Suppose tests for the presence and concentration of carbon monoxide (CO) and carbon dioxide (CO2) are required. First a tube is selected for carbon dioxide, the ends are broken (removed) allowing air to be drawn through the tube. The tube is graduated and marked with the number of pump strokes required. The required amount of air is drawn through the tube. The presence of the gas (carbon dioxide) will cause colour change; the level of concentration is indicated by the graduated markings reading from the air admission end. Rubber stoppers are often used to seal the tube for record purposes.

Where carbon dioxide is to be tested the appropriate tube is selected and the procedure is repeated. There are available a wide range of gas detection tubes for various gases and levels of concentration. An example of this type is the Drager Multi Gas Detector (See Fig. 17.4).

CHEMICAL ANALYSIS
When exact analysis of the mine atmosphere is required, samples must be collected underground and analysed in the laboratory. Sampling flasks of glass or metal may be employed which are either evacuated for self-fitting or are filled by aspirator bulb. Most gases may be collected in this way, but nitrogen oxides require a flask with a liquid absorber.

The most common procedure for analyzing air or gas samples is the volumetric method in which the constituent gases are removed successively by absorption. Methods used have been by Haldane, and Orsat apparatus.

Laboratory analysis may be applicable to calibration of direct reading instruments as well as testing mine atmospheres under special circumstances.
Figure 17.4 Drager Multi Gas Detector—Used in mining and for the analysis of technical gases

**GAS CONTROL**

The control of atmospheric gaseous contaminants is based on a number of principles:

1. Prevention (avoidance)
2. Removal (elimination)
3. Suppression (treatment)
4. Containment (enclosure)
5. Dilution (removal).

These are listed in their preferred order of application with prevention the most desirable, although dilution in the ventilating airstream is most commonly and effectively employed. Ventilation alone may be ineffective or prohibitively expensive in controlling contaminants therefore other methods are necessary.
CONTROL MEASURES IN DETAIL

PREVENTION
This includes proper procedures in blasting, adjustment maintenance of internal combustion engines and the avoidance of open fires underground.

REMOVAL
This involves drainage of gas from strata in advance of mining, e.g. methane drainage. Bleeder entries may be used along with local exhaust ventilation to aid removal.

SUPPRESSION
Absorption by chemical reaction such as by the internal combustion engine exhaust scrubber is involved. Solution of gases by air-water sprays after blasting is applicable to rises.

CONTAINMENT
This involves sealing off by isolation, abandoned workings or areas undergoing combustion (fire). Off-shift blasting or restricted blasting may assist in isolating the effect of contaminant gases.

Dilution involves the mixing or lowering of concentrations by either auxiliary or main ventilation air stream.

A summary of quality control measures applicable to particular gaseous conditions is as follows:

a. Control of strata gases: Achieved by bore hole drainage in advance of mining, bleeder entries around production areas, sealing off old workings, directed ventilation via bratticing and auxiliary ventilation.

b. Control of blasting gases: Achieved by good practice and proper selection of explosives, fume removal through exhaust systems, localized or off shift blasting times and main ventilation.

c. Control of I.C. engine exhaust (diesel): Achieved by proper engine maintenance, high air to fuel ratio, correct type of fuel, absorption and dilution of exhaust by scrubbers and adequate main ventilation.

d. Control of fumes and gases from fires: Involves the isolation of fire zones by bulkheads, flooding or suffocation of fire, reducing fume escape and dilution where necessary.

DILUTION REQUIREMENTS
Dilution by general ventilation is the most common method of gas control but should not be relied upon solely.

The quantity of fresh air required to dilute an airborne contaminant below MAC (maximum allowable concentration) can be calculated from:

\[ Q = \frac{Q_g}{L - L_o} - Q_g \]

where

- \( Q \) = quantity of air flow required
- \( Q_g \) = gas in flow in cubic metres/min.
- \( h \) = the (MAC) or maximum allowable concentration
- \( L_o \) = the gas concentration in the intake air
Case (a) A coal-mine sector has a methane emission ratio of 5.6 cubic metres/minute with a 0.2% methane content in the intake air. $h =$ Threshold unit of say 1% methane.

The Ventilation required:

$$Q = \frac{5.6}{0.01 - 0.002} = 5.6694 \text{ m}^3/\text{min}.$$ 

Such an air flow may be necessary in the case of a continuously mined section of a gaseous coal seam. Other control measures may reduce the quantity.

Case (b) Slow oxidation of a sulfide ore in a stope liberates 4.2 litres per minute of gaseous impurity (sulfur dioxide). The maximum allowable concentration is 0.0005%. The air intake is uncontaminated.

The ventilation required: (omitting the second $Q_g$)

$$Q = \frac{0.0042}{0.000005} - 0 = 840 \text{ m}^3/\text{min}.$$ 

**RADIOACTIVE GASES**

In Uranium mining, the radiation emitted by the ever-present radon gas and its daughter products are a particular problem. Air flowing through mine openings becomes contaminated before it reaches the working faces.

Auxiliary ventilation via ducting to the faces may be used to overcome this. Detection of the gas can be determined by the use of radon survey meters.

Actual standards applicable to ventilation for metalliferous mines are in part specified by the Mines Act and Regulations as well as the requirements of the Hygienic Standards for Contaminants of the Air of the Workplace from the Australian National Health and Medical Research Council publication Atmospheric Contaminants 1970 and amendments.

**MINE DUSTS**

Dust as particulate matter is the second category of airborne contaminants of major concern in underground mines. Airborne suspensions of particulate matter are referred to as aerosols. The main concern is dust particles having a diameter less than 10 $\mu$m ($10^{-6}$ metres or 10 micro metres) although hazardous respirable dust particles are those of $\frac{1}{4}$ to 5 $\mu$m in size.

The dynamic behaviour of dust particles of this size are such that they have very low settling velocities and will tend to remain air borne even with the slightest turbulence or movement. Particles over 10 $\mu$m will settle out relatively rapidly. Nuisance dust as seen in dust storms and dust from gravel roads, which is predominantly of particle size greater than 40 $\mu$m, will settle very rapidly.

**CLASSIFICATION OF DUSTS**

Dusts may be classified according to their harmful physiological effects or explosive properties:

- Pulmonary dusts (harmful to respiratory systems). e.g. silica, silicates, metal fumes, ores, coal.
- Toxic dusts (poisonous to human body) e.g. heavy metals - arsenic, lead, vanadium, mercury.
- Radioactive dusts (radioactive hazard) e.g. ores of uranium, radium, thorium.
- Explosive dusts (combustible when air borne) e.g. metallic dusts - magnesium, coal, sulfide ores.
- Inert dusts - few if any.

Dusts if present in excessive amounts for a sufficient length of time can cause pathological damage to humans, but this involves factors of composition, particle size range and susceptibility of the individual are involved.

Even where the amounts of dusts present in a mine are demonstrated to be harmless to life or property, control is still warranted. They may then only be nuisance dusts, lessening visibility and creating uncomfortable environments. Poor visibility in itself is a potential hazard.

**PHYSIOLOGICAL EFFECTS OF DUSTS**

While the human respiratory system has built in protection mechanisms. Lung damage may occur when dust within the respiratory range of \( \frac{1}{4} \) to 5 µm when excessive or chemically active is present or the exposure is continuous and prolonged over a number of years.

Respiratory diseases arising from inhalation of dusts is group-named pneumonoconiosis. This may be fibrous or non-fibrous damage to the lungs which causes shortness of breath and increases the frequency of other respiratory ailments.

Principal associated dust ailments are:

<table>
<thead>
<tr>
<th>Ailment</th>
<th>Dust</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Silicosis (miner’s phthisis)</td>
<td>Silica</td>
</tr>
<tr>
<td>2. Asbestosis</td>
<td>Asbestos</td>
</tr>
<tr>
<td>3. Siderosis</td>
<td>Iron</td>
</tr>
<tr>
<td>4. Anthracosis</td>
<td>Coal</td>
</tr>
</tbody>
</table>

Mine workers under the Mines Regulation Act and Regulations in Western Australia are required to possess a Mine Workers’ Health Certificate which involves x-ray examination for pneumonoconiosis or other lung disorders. These Certificates have varying periods of validity according to the type of mine.

Mine dusts which involve toxic or irritant effects and radioactive dusts are hazardous and require stringent control measures.

The effect of dust exposure may only be apparent many years after actual contact with the material.

There are a number of factors which are associated with harmfulness of dusts to humans. These are often referred to as excessive dust, however they can be identified as:

**Air borne Dust Harmfulness**

1. Composition e.g. chemical, mineralogical
2. Concentration e.g. number basis, mass basis
3. Particle size e.g. mean and range
4. Exposure time
5. Individual susceptibility.

The incidence of pneumonoconiosis has been clearly related to the level of concentration and the exposure time, e.g. high levels of dust and extensive periods of exposure especially in excess of fifteen years.

**EXPLOSIVE DUSTS**

A dust explosion is a sudden pressure rise caused by very rapid combustion of airborne dust. Ignition of suspensions of combustible dusts can occur by:

- initiation by flame or spark;
- propagation by a gas explosion or blasting;
- spontaneous combustion (the last is relatively rare).

Coal dusts have been particularly dangerous however the composition, particle size and concentration have an important bearing on the likelihood of an explosion. The presence of methane gas enhances the ease of ignition of coal dust.

Coal dust explosions can be prevented by maintaining sufficient incombustible content in the settled dust. This is usually achieved by stone dusting with limestone or gypsum constituting approximately 65% of the dust likely to become airborne in the event of gas explosion which will create a suspended dust cloud.

**MAXIMUM ALLOWABLE CONCENTRATIONS OF DUSTS**

The standards of purity of mine air in W.A. Mines is specified by the Mines Regulation Act and Regulations.

The standards specify a number of definitions in relation to free silica, inert or nuisance dust, respirable dust, silicious dust and total dust.

Respirable dust is the fraction of dust sampled from airborne dust in a mine which is retained on the filter of a size selective sampler.

Silicious dust is airborne dust which contains more than 5% quartz - \( (\text{SiO}_2) \) by mass.

Inert or nuisance dust is a dust which contains 5% or less free silica by mass (otherwise it is not classified as a hazardous dust).

Total dust is the mass of dust which is retained on the filter of a non-size selective gravimetric filter.

Prescriber sampling methods and maximum permissible levels.

**SILICIOUS DUSTS**

A size selective gravimetric dust sampler is used for a period of not less than four hours at a working place.

The prescribed permissible mass of respirable dust in milligrams per cubic metre of air is derived by the formula:

\[
\frac{25}{\% \text{ of respirable free silica}} + 5
\]

The maximum allowable concentration in dust sample is presently 2.5 milligrams per cubic metre.
TOTAL DUST – INERT OR NUISANCE DUSTS
A non-size selective gravimetric dust sampler is used for a continuous period of not less than 15 minutes at a working place.

The permissible maximum mass of inert or nuisance dust ( > 5% free silica) is 15 milligrams per cubic metre.

Standards applicable to specific materials are:

<table>
<thead>
<tr>
<th>Material</th>
<th>Maximum Concentration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Asbestos or fibrous talc</td>
<td>4 fibres per cubic cm or air</td>
</tr>
<tr>
<td>Rock containing lead</td>
<td>0.2 milligrams of lead per cubic metre</td>
</tr>
<tr>
<td>Rock containing manganese</td>
<td>5 milligrams per cubic metre</td>
</tr>
<tr>
<td>Talc or mica</td>
<td>1.5 milligrams per cubic metre</td>
</tr>
<tr>
<td>Vanadium</td>
<td>0.5 milligrams per cubic metre</td>
</tr>
</tbody>
</table>

Others are as prescribed by the recommended value for the metal or mineral in the Australian National Health and Medical Research Council publication ‘Atmospheric Contaminants’.

 SOURCES OF DUSTS IN MINES
Nearly all mining activity is responsible for air pollution by dusts. Some operations produce or create dust and may be termed primary sources, others agitate or disperse dust and are referred to as secondary sources.

Major primary sources are blasting, cutting – continuous mining, drilling. These operations involve high rates of energy dissipation in fragmentary rock.

Secondary sources are those associated with ore handling and transfer such as loading – excavation, drawing chutes, dumping of trucks and slushing.

 DUST CONTROL TECHNIQUES
The basic methods applicable to the reduction of harmful dust include the following:

- reducing the production of dust
- preventing the dispersal of dust clouds
- providing dilution ventilation
- utilizing personal protective measures.

In reducing the production of dusts it may be necessary to modify operations or improve practice. This applies in particular to the primary sources of dust such as drilling. Actual operational procedures of hole collaring wet, (i.e. with water) and the use of sharp bits can reduce dust generation. Drill holes underground are required to be water-flushed unless fitted with dust extraction equipment. It is essential that dust is prevented from becoming airborne. Once airborne the problem of control is difficult and costly. Therefore prevention is desirable at all stages.

Dust dispersal can be controlled by the use of water and the control of air currents.

While water and other wetting agents are not significantly effective in allaying airborne dusts, they are useful in suppressing dust at its source. Wetting down of muck piles etc.
assist in this regard. Finely atomized sprays of water via air-water nozzles are used to reduce dust clouds created by blasting and at transfer points for broken ore. However at best they are only partially effective in reducing airborne respirable dust.

Dispersal of dust can be substantially reduced by confining the dust producing operation in an enclosure. The contaminated air is exhausted to an upcast airway or to a filtering unit. Where air must be recirculated the latter is necessary.

The most common method of controlling airborne contaminants is by dilution and removal via ventilation. This is common to producing areas such as stopes and haulage drives. In general the volume of air required is based on air flow velocities of 10 to 15 metres per minute.

For development headings, ventilation is usually provided by auxiliary systems such as ducted systems either blowing, or exhausting air, sometimes using a combined system.

Personnel protection is generally considered as a last resort measure but can be effective. An indirect measure is the scheduling of blasting at the end of shift. Dust respirators generally have limited acceptance. Development towards safety hats which incorporate a filtration-fan system which blows fresh clean air over the face of the wearer may have particular applications to underground operations where dust is a particular problem e.g. continuous coal mining.

While surface mining is carried on in the relatively fresh atmosphere dust contaminants are still a problem. Dust control measures should be provided for operations which produce excessive dust over long periods of time. Wetting is applicable to excavation, haulage, dumping and crushing. Water sprays, water cannons and the like have been used. Productivity of an operation can increase where dust is virtually eliminated; dust clouds are hazardous in impairing vision of haulage unit operators and other transport drivers.

Dust control measures in drilling include detergent additives to aid dust collection (floculation of particles) and use of cyclone collectors. Dirt removal can pay off in health and clear conditions whilst reducing machine wear and maintenance.
INTRODUCTION

Ventilation is fundamentally the control of air movement, its amount and its direction. Air is required in a mine for the supply of oxygen for breathing, to remove heat, and to dilute and remove dust and gases. In examining aspects of air movement it is fundamental to look at the physical properties of the medium.

PROPERTIES OF AIR

Air is colourless, odourless, tasteless and supports combustion and life. Other properties can be classified as physical or psychrometric. The physical properties consist of those of the fluid, both at rest and in motion; quantity control is concerned with dynamic properties. Psychrometric properties relate to the thermodynamic behaviour of air and water-vapour mixtures which are particularly important in temperature-humidity control.

DENSITY OF DRY AIR

The density of dry air can be determined using the general gas law or Characteristic Gas Equation (a combination of Boyle’s and Charles’ laws) i.e.

\[
P V = M R T
\]

where

- \( P \) = pressure
- \( V \) = volume
- \( T \) = temperature (K)
- \( M \) = mass
- \( R \) = gas constant

The gas constant \( R \) for dry air is 0.287 kJ/(kg K).

From this formula the density (mass per cubic metre) or specific volume (volume of one kilogram) of dry air at any temperature or pressure can be calculated.

Example: Calculate the density of air at 90 kPa and at 20° C (273 + 20 = 293 K).

\[
PV = MRT
\]

\[
90 \times 10^3 \times V = 1 \times 287 \times 293
\]

\[
= \frac{287 \times 293}{90 \times 10^3} = 0.934
\]

\[
\text{density} = \frac{1}{V} = \frac{1}{0.934} = 1.07 \text{ kg/m}^3
\]

(Note this is of DRY AIR)

Attention is drawn to the use of dry air as a reference base in defining most of the psychrometric properties of air. This imaginary standard is employed because it is the only
quantity (1 kg mass of dry air) which remains constant when air undergoes thermodynamic changes during air conditioning processes.

**CONSTANTS FOR DRY AIR**

- Atomic weight: 28.96 kg/kmo1
- Gas constant: 0.287 kJ/(kg k)
- Density at 15° and 101.3 kPa: 1.21 kg/m³

**HUMIDITY AND DEW POINT OF AIR**

The amount of water vapour that can be contained in a volume of air is dependent upon the temperature. Where the space contains as much vapour as it can hold at the existing temperature it is said to be saturated.

The water vapour in air exerts a vapour pressure. Barometric pressure is the sum of the partial pressure of dry air and water vapour.

The effect of water vapour in air is to lower the density of the mixture: i.e. one cubic metre of dry air and saturated water vapour has less mass than one cubic metre of dry air at the same temperature and pressure.

Where there is insufficient water present the pressure does not build up to the saturated value. The ratio of the actual vapour pressure and that at saturation for the same conditions is termed the relative humidity.

This is different from the term percentage humidity, the ratio of the actual mass of water vapour to the mass of water vapour which could be contained at saturation.

The gas law which relates to the pressures exerted by gaseous mixtures is defined by Dalton's law. This equates the partial pressures of the individual gases to the total pressure of the mixture.

\[
P_b = P_a + P_v\]

where

- \(P_b\) (pressure - total or barometric)
- \(P_v\) (partial pressure of water vapour)
- \(P_a\) = (partial pressure of dry air)

When air containing water vapour is cooled a temperature will be reached where it will no longer be able to hold all the vapour and some of the vapour will condense. This temperature is called the dew-point.

**AIR MEASUREMENTS**

In ventilation, measurements of the various properties of air are required. These include both static and dynamic both of which are relative to physical and psychrometric (measurement of humidity of air). The following properties are commonly determined for quantity control either by direct measurement or by calculations:

- temperature, dry bulb, wet bulb,
- atmospheric pressure,
- density,
- velocity,
• quantity,
• pressure head (static, velocity, total).

Ventilation measurements are required for use in ventilation surveys and to provide information for:
• extent and adequacy of existing ventilation in meeting specific needs, standards and regulations;
• planning for contingencies;
• planning for improvements of present environmental conditions or system efficiency;
• to make provision for modifications due to mining extensions involving new fans, airway changes or circuits.

TEMPERATURES

The dry bulb and wet bulb temperature of air can be determined simultaneously with a sling psychrometer or whirling hygrometer.

The sling psychrometer consists of two thermometers mounted side by side on a rigid frame. The thermometers cover the range of 0° to 50° C with 0.5° C graduations. One, the wet bulb, has a wick of muslin net encasing the Mercury bulb. The mounting frame is attached by a swivel connection to a handle, and the entire assembly can be whirled by applying a simple, rotation wrist action to the handle. (See Fig. 18.1)

The principle of the psychrometer is that the dry-bulb thermometer reading is a measure of the sensible heat of the atmosphere, while the wet bulb measures the evaporation rate of the air. Evaporation of moisture from the wet bulb proceeds at a rate which is proportional to the vapour pressure of the moisture in the ambient air. Moisture is supplied from a water reservoir to the wet bulb on the psychrometer.

If the air contains little moisture, its vapour pressure is low, and evaporation from the wet bulb takes place rapidly. This removes heat from the bulb and the temperature drops. Whirling removes the layer of saturated air which would soon surround the bulb. Radiation effects from the operator or other objects is reduced by whirling generally at a speed of about 3 m/s.

The psychrometer readings of the dry and wet-bulb temperatures are used in determining air density and temperature-humidity.
ROCK TEMPERATURES

Measurement of the temperature of the rock adjacent to underground workings is necessary in temperature - humidity air conditioning. Readings are usually taken with a maximum-reading thermometer placed in a drill hole to obtain a virgin-rock temperature unaffected by the presence of a mine opening.

However clinical mercury thermometers or electrical resistance thermometers may be used to measure temperatures in drill holes.

AIR DENSITY

The air density (p) or specific mass of normal moist air is not measured directly but is determined by calculation or from a chart of temperature and pressure measurements.

Psychrometric formulae can be derived from the General Gas Equation.

Example

Determine the density of humid air at 90 kPa and 20°C. Note: density of dry air at 90 kPa and 20°C is 1.07 kg/m³.

(a) Assume that the air is saturated. Read from a set of steam tables, that the vapour pressure of water at 20°C is 2.34 kPa and that a cubic metre of saturated vapour at 20°C has a mass of 0.017 kg (i.e. \( \frac{1}{57.8} \))

The partial pressure of the air is 90 – 2.34 = 87.66 kPa

Using this pressure in the General Gas Equation

\[ PV = RT \]
\[ 87.66 \times V = 0.2871 \times 293 \]

however \( p = \frac{1}{V} \) where \( p = \) density

\[ p = \frac{87.66}{0.2871 \times 293} = 1.042 \text{ kg/m}^3 \]

This is the mass of air in one cubic metre of mixture (air and water vapour).

The total mass of air plus water vapour in a cubic metre of the mixture is then:

1.042 + 0.017 = 1.059 kg

Note that the density of moist air is less than the density of dry air.

(b) Relative humidity of the air is 50%.

The partial pressure of the water vapour is now 50% i.e. 2.34 \times 0.5 = 1.17 kPa.

The partial pressure of the air is 90 – 1.17 = 88.83 kPa.

Calculation:

\[ 88.83 \times V = 0.2871 \times 293 \]

\[ p = \frac{1}{V} = \frac{88.83}{0.2871 \times 293} = 1.056 \]
The total mass of air plus water vapour in a cubic metre of the mixture is then
\[ 1.056 + 0.017 \times 0.5 \]
\[ = 1.056 + 0.009 = 1.065 \text{ kg} \]
A variety of charts and tables is available for determining the humidity and air density given the dry and wet-bulb temperatures. These are useful because calculation formulae are in general not convenient for rapid determination of density.

**AIR VELOCITY**

The velocity of air is one of the most frequently measured properties in mine ventilation. While not significant in itself, the velocity must be known in order to calculate the quantity of air flow. Instruments are often classified according to an arbitrary velocity range: low (0.5 m/s), moderate or intermediate (0.5 m/s to 4 m/s) and high (over 4 m/s).

**INSTRUMENTS**

**SMOKE TUBE**
These are used to determine the presence of moving air, direction of flow, and the approximate velocity of flow. The device consists of an aspirator bulb which discharges through a glass tube containing a smoke generating reagent. The velocity is determined by timing the cloud as it travels a specific distance. A correction factor of about 0.8 is applied because of velocity distribution in an airway. (The velocity of air in the middle of an opening is higher than at the side).

Calculation of mean air velocity (frame smoke method).

The formula

\[ V = \frac{fd}{t} \]

where

- \( f \) = velocity distribution factor (0.8)
- \( d \) = distance in metres
- \( t \) = time in seconds.

Velocities are reported to the nearest 0.05 metres per second.

**VANE ANEMOMETER**
This is generally referred to as anemometer. It consists of a small windmill geared through a small clutch to be a cumulative – recording mechanism. The sweep pointer records directly on the dial in metres of linear air flow with the velocity being measured over a minute time interval hence velocity in m/min or m/s can be obtained. The clutch is engaged at will at the start and stop of the run; a zero-setting device is also generally provided. Instruments may be hand-held or mounted on an extension rod. High velocity anemometers are available for air velocities in the range of 10 to 50 m/s. Anemometers are the standard type of instrument for most mine ventilation work.

Instruments are usually supplied with a calibration chart for correction of measurements. (See Fig. 18.2)

**VELOMETER**
This instrument provides a rapid – instantaneous velocity reading. Air enters a port on one side of the instrument deflecting a vane proportional to the velocity head and is registered by a pointer on a dial. With the use of adaptors and multiple scales the instrument is usable...
over a range of air velocities. Pressure readings are possible with appropriate scales and probe for use with ventilation ducts.

**Thermometer**
There are several types which measure velocity by the rate at which heat is removed from a heated object in the air stream.

A common type is the Hot-wire anemometer which has a wire heated by batteries with the amount of heat dissipated being determined by a thermocouple in the probe. The instrument is usually capable of performing in a number of velocity ranges; however it is unsuited to atmospheres containing explosive gases.

**Kata Thermometer**
The Kata thermometer is used mainly for determining the cooling power of the air in mine temperature – humidity control, but it may be used to measure air velocity.

It is similar to a thermo-anemometer because it functions on the rate at which heat is removed from a thermometer. The bulb of the instrument is heated in water and then allowed to cool in the atmosphere being measured. The time taken for it to cool over a specified range of temperature, divided into a given factor, gives the wet kata reading. The bulb of the instrument for wet kata readings has a sleeve surrounding it. Dry kata readings may also be used to determine air velocity. (See Fig. 18.3)

**Pitot Tube**
The pitot tube with manometer has wide acceptance for the measurement of high air velocities. It is accurate and provides a means of determining pressure of head conditions within an airway or duct.

The pitot tube consists of two concentric tubes bent in an L shape. The inner tube is open on the end and receives the total pressure of the air stream. The outer tube is closed on the end and receives the static pressure of the air stream through a number of pin holes set back from the end.
The tip is tapered or made hemispherical to reduce interference with the air stream. The opposite end of the pitot tube is provided with two fittings, one for each tube, to which the legs of the manometer or water gauge are connected.

![Diagram of a pitot tube and manometer](image)

**Figure 18.3 Wet kata thermometer**

The manometer or 'U' tube consists of a U tube either of glass or transparent plastic with a minimum inside diameter of 5 mm. The limbs of the tube are half-filled with water; any pressure difference from the location of connected tubing is measurable as a difference in height of the water level in the two legs. A graduate scale calibrated in pressure units of 20 Pa intervals may be incorporated. A simple manometer is shown in Fig. 18.5.

A more sensitive manometer is the inclined manometer which has a long inclined limb. (See Fig. 18.6)

The units registered by fluid displacement in a manometer can be a measurement of height difference; alternatively, a calibrated scale in pascals may be incorporated. To calculate the pressure registered by a manometer the following equation can be used.
\[ P = h \times \rho \times g_n \text{ where} \]

\[ P = \text{differential pressure (Pa)} \]
\[ h = \text{manometer reading (m)} \]
\[ \rho = \text{fluid density (kg/m}^3) \]
\[ g_n = \text{acceleration due to gravity } = 9.806 \text{ m/s}^2 \]

Where water is used then the equivalent pressure to one millimetre is -

1 mm H\(_2\)O at 20\(^\circ\) C = 9.789 Pa.

The principle of operation of the pitot tube and manometer is that moving air exerts a pressure. This pressure is usually referred to as the velocity pressure. However, depending upon the manner of connection of the limba of the manometer and the pitot tube, a variety of pressure or head readings may be obtained. These include velocity pressure which represents the kinetic energy of the air while static pressure represents the potential energy of the air. The algebraic sum of the two is the total pressure.
Velocity pressure always has a positive value, while a static pressure difference relative to other air (such as in a duct) may be positive or negative. measuring:

Fig. 18.7 illustrates a Pitot tube and manometer measuring:

- The velocity pressure or velocity head;
- the total head;
- the static head or pressure.

Note that $H_T = H_V + H_S$
CHAPTER 18 MINE VENTILATION – AIR AND AIR MEASUREMENT

18.10

Figure 18.7 Pitot tube and monometer connections

a. velocity head or pressure
b. total head or pressure
c. static head or pressure

Actual positive (boosting) and negative (exhausting) air flows in ducts (static pressure) and methods of obtaining velocity and total pressure reading are illustrated in Fig. 18.8.

Figure 18.8 Static and velocity pressures for positive and negative duct pressures

The pitot tube is held in the airstream so that the central aperture is facing into the airstream.

The readings from the manometer can be used to determine the velocity of air by the following formula.

\[
V = \sqrt{\frac{2}{\rho C} \cdot C_V^2 \cdot \frac{P_V}{P}}
\]

where:
- \( V \) = air velocity (m/s)
- \( P_V \) = velocity pressure (Pa)
- \( \rho \) = air density (kg/m³)
- \( C \) = dimensionless coefficient (approximately unity) for pitot-static tube

From this formula the velocity can be calculated corresponding to any velocity pressure where the density of air is known.

For a rough rule of thumb, air velocity is approximately 40 m/s when the velocity is 1 kPa, (air density at 1.25 kg/m³).
A relationship between air velocity and velocity pressure to bear in mind is that velocity pressure is proportional to the square of the velocity, i.e.

\[ V \propto (P_v)^2 \]

where \( P_v \) is velocity pressure.

The pitot tube provides a basic means of measuring air velocities as well as an accurate method for calibrating other instruments.

Table 18.1 shows a summary of the types and characteristics of instruments used to measure velocity of air in ducts and mine openings.

**PROCEDURES FOR MEASURING VELOCITY**

The velocity of air over the cross section of a duct or airway varies; procedures need to be adopted to measure an average value.

These techniques range from single measurements, multiple measurements to fixed point traversing and continuous traversing.

Single point measurements involve a correction factor. Multiple readings may be made by means of a pitot tube, say, inside a duct. Anemometers are usually continuously traversed across the cross section.

Methods of measurement are illustrated in Fig. 18.9.

![Figure 18.9 Methods of traversing mine airways](image)

a. Continuous traversing  
b. Fixed point traversing
### CHAPTER 18 MINE VENTILATION – AIR AND AIR MEASUREMENT

#### 18.12 Instruments for measuring air velocity in mines

<table>
<thead>
<tr>
<th>Instrument</th>
<th>Accuracy</th>
<th>Sensitivity m/sec</th>
<th>Velocity Range m/sec</th>
</tr>
</thead>
<tbody>
<tr>
<td>Smoke tube</td>
<td>70 – 90%</td>
<td>0.03 – 0.06</td>
<td>0.1 – 0.7 (low)</td>
</tr>
<tr>
<td>Vane</td>
<td>80 – 90%</td>
<td>0.06 – 0.14</td>
<td>0.8 – 11.1 (low to intermediate high)</td>
</tr>
<tr>
<td>Anemometer</td>
<td>95% upper scale</td>
<td>0.28 – 0.56</td>
<td>11.1 – 55.6 (high to high)</td>
</tr>
<tr>
<td>Velometer</td>
<td>90 – 95%</td>
<td>0.03 – 0.06</td>
<td>0.17 – 16.7 (low to high)</td>
</tr>
<tr>
<td>Hot wire anemometer</td>
<td>70 – 90%</td>
<td>0.01 – 0.02</td>
<td>0.06 – 0.14 (low to high)</td>
</tr>
<tr>
<td>Kata thermocouple</td>
<td>90 – 98%</td>
<td>0.06 – 0.14</td>
<td>0.56 – 8.33 (low to high)</td>
</tr>
<tr>
<td>Pitot tube</td>
<td></td>
<td>0.06 – 0.14</td>
<td>4.2 – 55.3 (high)</td>
</tr>
</tbody>
</table>

**Table 18.1 Instruments for measuring air velocity in mines**

### AIR QUANTITY

The quantity of air $Q$ flowing in an airway is not measured directly but is calculated from the average velocity and the cross sectional area of the airway $A$ at the point of measurement.

$$Q = VA \text{ (m}^3/\text{s})$$

In order to attain a reasonable order of accuracy commensurate with the velocity measurement, care should be taken in determining the cross sectional area at the point of measurement.
The value of Q in spot checks can indicate whether or not the ventilation is satisfactory.

**Air flow example**
Determine the air flow Q (quantity) for a mine airway (main level drive) for which a velocity measurement of 7.5 m/sec has been made by anemometer. The drive at the measuring station has a cross section of 3 m × 2.4 m.

\[ Q = V \times A \]
\[ = 7.5 \times 3 \times 2.4 \]
\[ = 54 \text{ cubic metres per sec.} \]

To obtain quantity per minute - multiply this by 60, hence
\[ = 3240 \text{ cubic metres per minute.} \]

**AIR FLOW**
The fundamental principles of air flow are:

- Airflow is induced by a pressure difference between the intake and exhaust openings of a mine.
- The pressure difference is caused by imposing some form of pressure at a point or points within the ventilation system.
- The pressure required to cause air flow must be greater than frictional resistance and other losses.
- Presence of passageways for movement of air.
- Air flows from a point of higher pressure to one of lower pressure.
- Air flow has a square law relationship between volume and pressure.
- Mine ventilation pressures are positive (above atmosphere) where blowing, or negative (below atmospheric pressure) where exhausting.
- Pressure drops in parallel splits will be the same irrespective of the air flow in each split.

Note: 1 mm of water gauge represents a pressure of 9.8 Pa.

These principles will be expanded in developing an overall concept of ventilation.

**REYNOLDS NUMBER**
The flow of a gas or liquid (fluid) is retarded by its internal resistance or viscosity, which is the cohesion between the molecules.

There are two distinct states of flow with a transitional zone between. When a fluid flows slowly movement is laminar, i.e. layered, with different layers flowing over each other. At high velocities the flow becomes turbulent with molecules moving at random, with eddies and greater dissipation of energy. The point at which fluid flow becomes turbulent is the critical velocity.
This critical velocity point is defined by a coefficient referred to as Reynolds Number determined from its formula:

\[ \text{Re} = \frac{V D \rho}{\eta} \]

where \( \text{Re} \) = Reynolds number (dimensionless coefficient)

\( V \) = velocity (m/s)
\( D \) = pipe diameter (m)
\( \rho \) = density (kg/m\(^3\))
\( \eta \) = dynamic viscosity (Ns/m\(^2\))

When the Reynold's number is less than 2000 the flow is laminar; when it is more than 4000 the flow is always turbulent.

In mine openings it is desirable that air flow be in turbulent regime so as to ensure dispersion and removal of contaminants. With few exceptions this is the case, with turbulent flow applying to most mine ventilation.

**ATKINSON'S EQUATION**

When air flows through a duct, whether a pipe, tunnel, shaft or stope, the pressure required to move the air depends upon the internal air friction, size, length and shape of the duct, the roughness of its walls, the nature of obstructions and the velocity and density of the air.

These factors have been incorporated in the flow equation at standard density which is referred to as the Atkinson equation.

\[ P = \frac{K C L V^2}{A} \]

where

\( P \) = pressure loss (Pa)
\( K \) = friction factor (k\(^9\)/m\(^3\))
\( c \) = perimeter (m)
\( L \) = length (m)
\( V \) = velocity of air (m/s)
\( A \) = cross sectional area (m\(^2\))

This can be adapted to the actual air density i.e.

\[ K = K_s \times \frac{\rho}{1.205} \]

(correction for air density other than standard of 1.205 kg/m\(^3\))

Where \( K_s \) is the friction factor of air at standard density.

The friction factor is determined from experimental measurements.

The formula can be adapted to use quantity of airflow

\[ Q = V \times A \]

\[ P = \frac{K C L Q^2}{A^3} \]

This can be equated to

\[ \frac{fp}{2} \times \frac{S_v^2}{A} \quad \text{and} \quad RQ^2 \]
where \( S = C \times L \) rubbing surface area
\( R = \) resistance (kg/m²)
\( f = \) dimensionless coefficient
\( \rho = \) air density (kg/m³)

Using the pressure loss Atkinson formula the duct or mine resistance \((R)\) can then be determined.

\[
R = \frac{KCL}{A^3} = \left( \frac{P}{Q^2} \right)
\]

This can be adapted to the actual air density, i.e.

\[
R = R_s \times \frac{P}{1.205} \quad (R_s \text{ at standard air density})
\]

Where \( R_s \) is the friction factor of air at standard density.

**Friction Factors (kg/m³)**

<table>
<thead>
<tr>
<th>Airway</th>
<th>K</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ventilation tubing</td>
<td>0.003</td>
</tr>
<tr>
<td>Straight rock tunnel</td>
<td>0.01</td>
</tr>
<tr>
<td>Concrete lined shaft</td>
<td>0.05</td>
</tr>
<tr>
<td>Timbered rectangular shaft</td>
<td>0.08</td>
</tr>
</tbody>
</table>

**Examples:**

A. Determine the pressure drop along a 4 x 3 m tunnel, 1500 m long with an air flow of 60 m³/s and an air density of 1.35 kg/m³.

\[
K = 0.01
\]

\[
P = \frac{KCLQ^2}{A^3} \times \frac{P}{1.2}
= \frac{0.01 \times 14 \times 1500 \times 60^2 \times 1.35}{12^3 \times 1.2}
= 492 \text{ Pa}
\]

B. Determine the pressure drop for an airflow of 3 m³/s of air with a density of 1.2 kg/m³, size of duct 400 mm diameter and length 600 metres. \( K = 0.003 \)

The circumference \( C = \pi D = \pi \times 0.4 = 1.257 \)

The area \( A = \pi r = \pi \times (0.2)^2 = 0.126 \)

\[
P = \frac{KCLQ^2}{A^3} \times \frac{P}{1.2}
= \frac{0.003 \times 600 \times 9 \times 1.257}{(0.126)^3}
= 10180 \text{ Pa} = 10.18 \text{ kPa}
\]
B.2 For similar conditions to B.1, using a larger diameter ventilation tubing (500 mm) determine the pressure drop or loss.

This may be derived from the relationship where pressure drop is inversely proportional to the fifth power of the diameter.

\[ P_{500} = P_{400} \times \left( \frac{400}{500} \right)^5 \]

(where \( P_{400} \) is 10180 Pa for B1

\[ = 10180 \times 0.186 \]

\[ = 1892 \text{ Pa} = 1.9 \text{ kPa} \]

This illustrates that a large duct substantially reduces the pressure drop (loss) due to resistance.

**SHOCK LOSSES**

While Atkinson’s equation can be used to calculate the pressure loss due to friction, another pressure loss cause results from shock losses. These are caused by abrupt changes in the velocity of air movements as a result of changes in air direction, or airway area, obstructions and regulation. The total duct or mine loss of head or pressure is then equal to the sum of the friction, loss and the shock loss.

Generally a serious cause of shock loss is the presence of obstructions which cause eddy currents resulting in loss of energy. In order to reduce shock and resistance losses of this type especially in shafts, buntons are of streamlined shape.

Other causes of shock loss are changes in velocity especially where the velocity is suddenly reduced. This occurs both at intake and outlet of a duct system. In order to reduce such shock losses it is common to use a conical enlargement such as a fan evase - (i.e. like an expanding funnel).

**VENTILATION PRESSURE AND AIR POWER**

Air moves from one place to another when a pressure difference exists between the two. The volume of air that will move in unit time depends upon the pressure difference and the resistance to the flow.

Permanent pressure differences in mines are provided by means of fans.

An example of air flow in a mine air way: Air density is 1.35 kg/m\(^3\), the tunnel is 4 x 3 metres, 1500 m long and has a pressure difference of 1 kPa between the entry and exit.

\[ K = 0.01 \]

\[ P = \frac{KCLQ^2}{A} \times \frac{P}{1.2} \]

Where the pressure difference for the same air way is different, i.e. 492 Pa, the new quantity can be determined from the relationship \( Q^2 \propto P \) (\( Q \propto P^{0.5} \))

\[ 1000 = \frac{0.01 \times 14 \times 1500 \times Q^2 \times 1.35}{12^2 \times 1.2} \]

\[ 1000 = 0.1367 Q^2 \]

\[ Q = 85.5 \text{ m}^3/\text{s} \]
Where the pressure difference for the same airway is different, i.e. 492 Pa, the new quantity can be determined from the relationship \( Q^2 \propto P \) \( (Q \propto p^{1/2}) \)

\[
\frac{Q_2}{Q_1} = \sqrt{\frac{P_2}{P_1}}
\]

Where
- \( P = \) frictional pressure loss Pa
- \( Q_1 = 85.5 \) m\(^3\)/s
- \( P_2 = 492 \) Pa
- \( P_1 = 1000 \) Pa

\[
Q_2 = 85.5 \sqrt{\frac{492}{1000}} = 60 \text{ m}^3/\text{s}
\]

In order to ascertain the amount of energy that is required to move air it is necessary to calculate the air power.

**BASIC UNITS**

Work is done when air moves against a resistance, and energy is required to do this work.

Power is the rate of doing work. A joule is the amount of work done when a force of 1 newton moves its point of application by 1 metre.

When a cubic metre of air passes through an opening of one square metre at such a velocity that the pressure loss is one newton per square metre (1 pascal) then the amount of work done is equivalent to one newton metre or one joule.

Where a cubic metre of air moves through a two sq. metre aperture with the same pressure drop the air moves only 0.5 metre.

The amount of work done is independent of the cross-sectional area through which the air passes. It depends on the volume of air which is moved and the amount of pressure or force which is required. Hence where 60 m\(^3\)/s of air is moved through a tunnel of any size and any distance and with the pressure drop of 1 N/m\(^3\) (Pa) then the work done is:

\[
60 \text{ m}^3/\text{s} \times 1 \text{ N/m}^2 = 60 \text{ Nm} = 60 \text{ J}
\]

Where the pressure drop is 492 Pa then the work done for an air flow of 60 m/s is:

\[
60 \text{ m}^3 \times 492 \text{ N/m}^2 = 29520 \text{ Nm (J)}
\]

\[
= 29.5 \text{ kJ}
\]

The air power required to move 60 m\(^3\)/s in a 4 x 3 m tunnel, 1500 m long and an air density of 1.35 kg/m\(^3\) with a pressure drop of 492 Pa is then:

\[
29520 \text{ J/s} = 29520 \text{ watts}
\]

\[
= 29.5 \text{ kW}
\]

A convenient formula for calculating air power is then:
$W_a = \frac{PQ}{1000}$

where $W_a =$ air power (Kw)

$Q =$ volume flow ($\text{m}^3/\text{s}$)

$P =$ pressure (Pa)

**BASIC MINE VENTILATION CIRCUITS**

In mine ventilation, because for any air way or duct, the values of $K$ (friction factor) $C$ (circumference), $L$ (length) and $A$ (area) are constant, it can be ascertained that pressure loss or head loss is proportional to the square of the quantity of air flowing.

i.e. $P \propto Q^2$

This is a basic law of mine ventilation.

This equation can be used to establish the pressure loss for a number of air volumes.

i.e. $\frac{P_1}{P_2} = \left(\frac{Q_1}{Q_2}\right)^2$

hence $P_2 = P_1 \left(\frac{Q_2}{Q_1}\right)^2$

The values derived from this formula can be plotted on a graph i.e. pressure against volume. This allows a graph line to be drawn which is known as the characteristic curve of the duct or airway; it may also be referred to as the system's resistance curve. (See Fig. 18.10)

Characteristic curves are useful in predicting quantities and pressures where ducts and segments of a system are being considered.

**FLOW CIRCUITS**

Two basic combinations of ducts and airways in a ventilation system are possible: series or parallel. These occur in ventilation in combinations and are termed networks. Methods of calculating the pressure heads and quantity for both types of circuit are now outlined.
SERIES FLOW
Series flow occurs when airways are connected end to end. This may be achieved in a multi-level mine where stoping doors are used. In a series circuit the quantity of air flow is the same throughout the circuit.

\[ Q = Q_1 = Q_2 = Q_3 = Q_4 \]

The head or pressure loss equals the sum of the losses in the individual airways.

\[ P = P_1 + P_2 + P_3 + ..... \]

Thus the resistance is cumulative

\[ R = R_1 + R_2 + R_3 + ..... \]

Example: Given five airways in series \( Q = 5 \text{ m}^3/\text{s} \)
where the pressure losses are \( P_1 = 0.50 \text{ kPa}, \ P_2 = 0.75 \text{ kPa}, \ P_3 = 0.25 \text{ kPa}, \ P_4 = 0.50 \text{ kPa}, \ P_5 = 0.25 \text{ kPa}. \)

The total pressure loss is then

\[ P = P_1 + P_2 + P_3 + P_4 + P_5 \]
\[ = 500 + 750 + 250 + 500 + 250 = 2250 \text{ kPa or 2.25 kPa} \]

Note that each section of the circuit has a volume of airflow of 5 m\(^3\)/s.

PARALLEL FLOW
Airways are said to be connected in parallel when the total air flow is divided among them. This is termed splitting. Natural splitting allows the air flow to divide among the branches, of its own accord and without regulation, in inverse relation to the resistance of each airway.

When airways are in parallel the total quantity is the sum of the airflow in the airways:

\[ Q = Q_1 + Q_2 + Q_3 + ..... \]

The total pressure loss is the same across any branch:

\[ P = P_1 = P_2 = P_3 = ..... \]

From the equation \( R = \frac{P}{Q^2} \) and

because the pressure loss is the same in each branch, the relationship \( Q = Q_1 + Q_2 + Q_3 + ..... \) is modified to

\[ \frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} + ..... \]
Solutions of parallel flow circuits involving natural splitting are commonly based on resistance. The quantity of air which will flow in each airway must be determined, knowing the total quantity and the individual resistances. A characteristic of parallel circuits is that the pressure drop is the same for each heading and quantity of air flowing is proportional to the resistance.

\[
Q = \sqrt{P/Q} \quad \text{for each branch.}
\]

then \[ Q \propto \frac{1}{\sqrt{R}}, \quad Q_1 \propto \frac{1}{\sqrt{R_1}}, \quad \text{etc.} \]

this can be expressed as \[
\frac{Q_1}{Q} = \frac{1/\sqrt{R_1}}{1/\sqrt{R}}
\]

\[
Q = Q_1 \frac{1/\sqrt{R}}{1/\sqrt{R}}, \quad \text{etc.}
\]

where \( Q \) = air flow quantity \( \text{m}^3/\text{s} \)

\( P \) = frictional pressure loss \( \text{Pa} \)

\( R \) = resistance \( \text{N.S}^2/\text{m}^8 \)

For a parallel - two-split circuit where the total quantity is \( Q \) and limb quantities are \( Q_1 \) and \( Q_2 \), the equation is then:

\[
Q_1 = \frac{Q A_1}{A_1 \sqrt{S_1 K_1} + A_2 \sqrt{S_2 K_2}}
\]

and \[
Q_2 = \frac{Q A_2}{A_1 \sqrt{S_1 K_1} + A_2 \sqrt{S_2 K_2}}
\]
where \( A \) = cross sectional area of airway
\( c \) = airway circumference (perimeter)
\( S \) = rubbing surface area \((S = C \times 1)\)
\( l \) = length of airway

Thus the quantity of air in each split or limb is proportional to the pressure potential. For a three limb parallel circuit then where \( Q \) is the total quantity and the air in each limb is \( Q_1, Q_2 \) and \( Q_3 \) the equations used are:

\[
Q_1 = \frac{QA_1 \sqrt{\frac{A_1}{S_1K_1}}}{A_1 \sqrt{\frac{A_1}{S_1K_1}} + A_2 \sqrt{\frac{A_2}{S_2K_2}} + A_3 \sqrt{\frac{A_3}{S_3K_3}}}
\]

\[
Q_2 = \frac{QA_2 \sqrt{\frac{A_2}{S_2K_2}}}{A_1 \sqrt{\frac{A_1}{S_1K_1}} + A_2 \sqrt{\frac{A_2}{S_2K_2}} + A_3 \sqrt{\frac{A_3}{S_3K_3}}}
\]

and \( Q_3 = \frac{QA_3 \sqrt{\frac{A_3}{S_3K_3}}}{A_1 \sqrt{\frac{A_1}{S_1K_1}} + A_2 \sqrt{\frac{A_2}{S_2K_2}} + A_3 \sqrt{\frac{A_3}{S_3K_3}}}
\]

where \( A \) is the area of the airway
\( K \) is the friction factor
\( C \) is the rubbing surface of the airway.

**Example 1**

Parallel circuit – an air stream of 22.50 cubic metres per second is split between two airways having the following dimensions:

<table>
<thead>
<tr>
<th></th>
<th>Length</th>
<th>Height</th>
<th>Width</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>260 m</td>
<td>2 m</td>
<td>2 m</td>
</tr>
<tr>
<td>B</td>
<td>150 m</td>
<td>3 m</td>
<td>2 m</td>
</tr>
</tbody>
</table>

Calculating the perimeter, area and swept or surface area we derive:

<table>
<thead>
<tr>
<th></th>
<th>Perimeter (c)</th>
<th>Area A</th>
<th>( S ) ((C \times 1))</th>
<th>( A \sqrt{\frac{A}{S}} )</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>8 m</td>
<td>4 m²</td>
<td>2080 m²</td>
<td>0.1754</td>
</tr>
<tr>
<td>B</td>
<td>10 m</td>
<td>6 m³</td>
<td>1500 m²</td>
<td>0.3014</td>
</tr>
</tbody>
</table>

0.4768
Where \( K \) is the same for each airway with a value of 0.04 tabulated this is used to derive the following.

<table>
<thead>
<tr>
<th>Proportion to each Airway</th>
<th>% of Total</th>
<th>Quantity to each split</th>
<th>Cubic metre per second</th>
</tr>
</thead>
<tbody>
<tr>
<td>A. 0.1754 : 0.4768</td>
<td>36.8</td>
<td>0.368 x 22.5</td>
<td>8.28</td>
</tr>
<tr>
<td>B. 0.3014 : 0.4768</td>
<td>63.2</td>
<td>0.632 x 22.5</td>
<td>14.22</td>
</tr>
</tbody>
</table>

The pressure loss or potential in each limb can be ascertained from

\[
P = \frac{K S Q^2}{A^3}
\]

\[
= 0.04 \times \frac{2080 \times (8.28)^2}{(4)^3}
\]

\[
= 89.1 \text{ Pa}
\]

**Example 2**

Parallel circuit – an air stream of 22.5 cubic metres per second is split into three airways with the dimensions as follows:

<table>
<thead>
<tr>
<th>Airway</th>
<th>Length (m)</th>
<th>Height m</th>
<th>Width</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>330</td>
<td>3</td>
<td>2</td>
</tr>
<tr>
<td>B</td>
<td>100</td>
<td>1.33</td>
<td>2</td>
</tr>
<tr>
<td>C</td>
<td>600</td>
<td>4</td>
<td>2.67</td>
</tr>
</tbody>
</table>

From which the following can be derived

<table>
<thead>
<tr>
<th>Airway</th>
<th>Perimeter (C)</th>
<th>Area (A)</th>
<th>( S (C \times 1) )</th>
<th>( A \sqrt \frac{A}{S} )</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>10</td>
<td>6</td>
<td>3300</td>
<td>0.2558</td>
</tr>
<tr>
<td>B</td>
<td>6.60</td>
<td>2.66</td>
<td>660</td>
<td>0.1681</td>
</tr>
<tr>
<td>C</td>
<td>13.33</td>
<td>10.7</td>
<td>8800</td>
<td>0.3731</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Proportion to each Airway</th>
<th>% of Total</th>
<th>Volume in each Airway (m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A. 0.2558/.7970</td>
<td>32.1</td>
<td>7.22</td>
</tr>
<tr>
<td>B. 0.1681/.7970</td>
<td>21.1</td>
<td>4.75</td>
</tr>
<tr>
<td>C. 0.3731/.7970</td>
<td>46.8</td>
<td>10.53</td>
</tr>
</tbody>
</table>

The value of \( K \) is the same for each airway, with a value of 0.04.
CHAPTER 19 – Mine Ventilation – Natural and Mechanical Ventilation – Temperature – Humidity Control

NATURAL VENTILATION

Natural ventilation or natural draft, is the term used to describe mine air flow resulting from a pressure differential caused by unequal densities or weights of two connected air columns. This unequal density, which is due mainly to the transfer of heat or thermal energy (temperature difference), is analogous to the common chimney or stack effect, where warm air rises and displaces colder air above, producing circulation.

CHARACTERISTICS OF NATURAL VENTILATION

Natural ventilation depends upon the difference in elevation of the surface and the mine workings and the differences in air temperatures inside and outside the mine. In underground workings the temperature remains relatively constant throughout the year, therefore the temperature at the surface and its fluctuation will determine the direction and magnitude of natural ventilation.

Deep mines do not necessarily have good natural ventilation; only deep mines which are hot will generally possess strong natural ventilation. Mines in hilly terrain may have strong natural ventilation because of the differences in elevation of the openings.

The direction of flow of air is seldom constant with natural ventilation, especially in shallow mines (500 metres). Temperature variations can occur daily and/or seasonally which will either cause natural ventilation, reduce it to zero, or reverse its direction.

It is therefore concluded that natural ventilation fluctuates, is unstable and unreliable. It is usually strongest in winter and weakest in summer. Therefore natural ventilation cannot normally be relied upon as the only source of ventilation of a mine but it must be considered. It should assist artificial means (fans) when strongest and work against the fan at its weakest.

EXAMPLE OF NATURAL VENTILATION

In a mine having a shaft and an adit into a hillside (see Fig. 19.1) air will travel down the shaft during summer and up the shaft during the winter because the temperature of the air in the mine shaft stays comparatively constant throughout the year because of the consistency of the rock temperature, while the temperature of the outside air changes with the seasons. During the summer when the outside air is hot, it is too light to balance the cool column of air in the shaft and consequently air moves down the shaft and out of the adit. In winter the cold outside air is heavier than that in the shaft and consequently air enters the adit and up-casts through the shaft.

The amount of natural ventilation pressure generated by thermal energy is usually less than 120 Pa and seldom exceeds 735 Pa, except in extreme cases.
CHAPTER 19 MINE VENTILATION – NATURAL AND MECHANICAL VENTILATION – TEMPERATURE – HUMIDITY CONTROL

MECHANICAL VENTILATION

Natural ventilation is seldom constant or directionally stable therefore it is necessary to provide artificial ventilation or mechanical ventilation in all underground mines. This is achieved by using ventilation devices including all powered machines such as fans but it can also be achieved by using injectors.

<table>
<thead>
<tr>
<th>Need to induce</th>
<th>Left</th>
<th>Centre</th>
<th>Right</th>
</tr>
</thead>
<tbody>
<tr>
<td>Direction winter</td>
<td>Yes</td>
<td>No</td>
<td>No</td>
</tr>
<tr>
<td>Direction summer</td>
<td>None</td>
<td>Left to right</td>
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FANS

Basically a fan is an air pump, a machine which creates a pressure difference in a duct or air way and thus causes air flow.

FAN TYPES

There are two main types of fan:

- axial flow;
- centrifugal or radial flow.

An axial flow fan consists of a shaft with a hub or boss to which is attached a number of blades.

These blades may be fixed at a certain angle or may be adjustable so that their pitch can be altered. Some fans have variable pitch blades allowing blade angle to be altered whilst in operation. (See Fig 19.3)

When the axial flow fan is revolved the blades scoop up the air on the one side of the impeller and push it to the other side moving the air parallel to the rotating shaft. A spiral
motion is imparted to the air: to counteract this and to improve the efficiency of the fan, stationary blades are installed either on the inlet or outlet side of the blades.

Axial flow fans have casings, usually of three parts: the cylindrical section around the impeller, an inlet section - with a bell mouth to reduce intake losses, and a diverging outlet section or evase to reduce exit shock losses. The drive motor may be external to the fan casing and drive either the impeller shaft by belt and pulley or be a centrally mounted electric motor which is common in small to medium sized fans.

The centrifugal fan works on a different principle from the axial flow fan. Centrifugal means fleeing from the centre. The intake air is therefore at the hub of a cylindrical fan blade assembly and leaves from the blades through a discharge opening in the casing. (See Fig. 19.4) The actual blade configuration is one of three possible types:

- radial tip (perpendicular to the tangent of the impeller);
- forward curved;
- backward curved (See Fig. 19.5).
The impeller of a centrifugal fan consists of two rings with blades fitted between them. When the fan is revolved, air is drawn in parallel to the shaft into the open end or ends of the impeller and expelled in a radial direction through the blades. Fans may be either single inlet or double inlet depending on whether one or both sides of the impeller act as inlets.

Figure 19.5 Centrifugal fan and blade configuration

**FAN CHARACTERISTICS**

Fan characteristic curves indicate how much air a fan can deliver at any particular pressure and how much power is required to drive the fan in each case.

The most important operating variables are head, quantity, power and efficiency. The customary form of these curves plots all these variables against quantity. Fig. 19.6 illustrates a Characteristic Curve. Curves are plotted for a given fan diameter, constant speed, and standard air density.
Example from Curves Fig. 19.6:

The curves are applicable to the particular fan when driven at a speed of 10 r/s and when it is handling air at a density of 1.2 kg/m³.

The pressure-curve marked 'static pressure' indicates the fan will handle 175 m³/sec at no pressure, i.e. to ducting or in a mine, it will handle less air at a higher pressure. The higher the resistance of the mine, the less will be the quantity of air delivered and the greater the pressure put into it. At point A the fan delivers 92 m³/sec at a pressure of 1.6 kPa. If the mine resistance is increased, the fan will deliver still less air at a lower pressure – see point B. This illustrates the stall zone at which there is insufficient air to maintain uniform flow.

The fan input power curve (input power) indicates the fan shaft power required for the different air volumes being delivered. The shape of the input curve indicates the load characteristics of the fan. In this case it is non-overloading where the motor is adequate for any duty of the fan. This is typical of axial flow fans and backward bladed centrifugal fans. Radial bladed centrifugal fans however have overloading characteristics. In this case the motor can handle normal design duty but when the air volume is allowed to increase the motor will overload (See Fig. 19.7).

The efficiency curve is derived from the other two curves. The efficiency of a machine is defined as the ratio of the useful work output to the energy input and is usually expressed as a percentage.

$$\text{Hence fan efficiency (\%) = } \frac{\text{Air power} \times 100}{\text{Fan input power}}$$
CHAPTER 19 MINE VENTILATION – NATURAL AND MECHANICAL VENTILATION – TEMPERATURE – HUMIDITY CONTROL

Efficiency is zero where there is no air flow or at zero pressure, i.e. free delivery.

At the design point the fan delivers 115 m³/s at 1.37 kPa requiring power of 225 kW.

\[
\text{Air power} = 1.37 \times 115 = 157.6 \text{ kW} \\
\text{Efficiency} = \frac{157.6}{225} \times 100 = 70\%
\]

When the fan operates near the design point it is fairly efficient; however at say, half the delivery volume, efficiency is low.

**FAN LAWS**

The behaviour of a fan under changing head quantity conditions is predictable from its characteristic curves. However, there are certain variables other than the float and the resistance of the system which exert a considerable effect on fan performance.

![Figure 19.7 Overload characteristics of a radial bladed centrifugal fan](image)

Fan laws which are applicable to conditions of varying speed but with constant air density are:

- Air quantity varies directly as fan speed quantity is independent of air density, i.e. twice the volume requires twice the speed.
  \[
  Q_2 = Q_1 \times \left(\frac{S_2}{S_1}\right)
  \]

- Pressure induced varies directly as fan speed is squared, and directly as density, i.e. twice the volume develops four times the pressure.
  \[
  P_2 = P_1 \times \left(\frac{S_2}{S_1}\right)^2
  \]

- The fan power-input varies directly as the fan speed is cubed and directly as the air density. i.e. twice the volume requires eight times the power.
  \[
  \text{power}_2 = \text{power}_1 \times \left(\frac{S_2}{S_1}\right)^3
  \]

- The mechanical efficiency of the fan is independent of fan speed and air density.
  \[
  \text{Eff}_2 = \text{Eff}_1
  \]
Using these relationships and a set of operational characteristics of a fan under certain conditions, new characteristics can be calculated.

The effect of air density change is important because it is often responsible for day to day fluctuations in fan characteristics. Likewise fan performance changes as elevation changes because air density changes.

Hence a re-statement of laws with varying density but with constant speed:

- Quantity remains constant \( Q_2 = Q_1 \)
- Pressure varies directly as the density \( P_2 = P_1 \times \left( \frac{W_2}{W_1} \right) \)
- Power varies directly as the density \( \text{power}_2 = \text{power}_1 \times \left( \frac{W_2}{W_1} \right) \)
- Efficiency is constant \( \text{Eff}_2 = \text{Eff}_1 \)

Example of use of fan laws:

Using characteristics of the fan in Fig. 19.6 which is a fan running at 10 r/s and handling air at a density of 1.2 kg/m\(^3\), determine the new operation points at a speed of 13.3 r/s and handling air at a density of 1.04 kg/m\(^3\).

At its design point the fan delivers 115 m\(^3\)/s at 1.37 kPa using 225 kW at an operating efficiency of 70.0 per cent. (Speed 10 r/s and density 1.2 kg/m\(^3\)).

At new speed 13.3 r/s and density 1.04 kg/m\(^3\) the laws which apply are:

New quantity \( = 115 \times \frac{13.3}{10} \)
\( = 153 \text{ m}^3/\text{s} \)

New pressure \( = 1.37 \times \left( \frac{13.3}{10} \right)^2 \times \left( \frac{W_2}{W_1} \right) \)
\( = 1.37 \times \left( \frac{13.3}{10} \right)^2 \times \left( \frac{1.04}{1.2} \right) \)
\( = 2.1 \text{ kPa} \).

New input power \( = 225 \times \left( \frac{13.3}{10} \right)^3 \times \frac{1.04}{1.2} \)
\( = 459 \text{ kW} \)

New efficiency \( = 70 \times 1 \)
\( = 70\% \)

Efficiency check \( = \frac{153}{459} \times 100 = 70.0\% \)

Similarly other points on the curve can be recalculated and plotted onto a graph to establish the new characteristic curve (See Fig. 19.8).
FAN PERFORMANCE
When a fan is installed in a mine (or system) its performance will be determined by the point at which the mine (or system) characteristics cut the fan characteristic. The mine or system characteristic often referred to as mine resistance is a curve showing how the pressure drop across the mine or system varies as the quantity of air varies. In Fig. 19.8 a mine characteristic curve is shown which cuts the 18 r/s fan characteristic at the design point i.e. 115 m$^3$/s at 1.37 kPa.

The same mine characteristic curve cuts the 13.3 r/s fan characteristic at 114 m$^3$/s. However this fan characteristic is drawn for a density of 1.04 kg/m$^3$ while the mine characteristic is drawn for a density of 1.2 kg/m$^3$. If the 13.3 r/s fan curve is re-drawn for a density of 1.2 kg/m$^3$, it will give a value of 156 m$^3$/sec.

GENERAL NOTE
The information presented on fan curves illustrates, for simplicity, static pressure only. Fans impart static and velocity pressure to air; the algebraic sum of these two is the total air pressure.

FAN SELECTION
An important aspect of mechanical ventilation underground is not only finding a fan to meet mine pressure and quantity requirements but one of selecting a fan which would be best suited and most economical.

To ventilate a mine adequately, a certain quantity of air flow at a certain head and density is required. This is known as the mine head (pressure) and quantity. A number of fans will be available whose head quantity characteristics pass through the desired operating point or can be made to by changing speed, pitch or diameter.
Determining the operating point of the system is a relatively straightforward problem with a single fan, but becomes more complicated when two or more fans are installed in the same mine in such a way that they have an effect on each other.

A significant factor in mine fan operation is the operating cost because fans run continuously. In general the operating cost of a fan exceeds the purchase cost within a two year period. Therefore its mechanical efficiency becomes a significant factor in yearly operating costs.

Example of operating costs and efficiency:
A mine requires a fan to handle 150 m$^3$/s at a pressure of 0.66 kPa over a period of 15 years. A new fan gives a performance at 75% efficiency with a capital cost of $30,000 installed and uses 132 kW costing some $25,000 a year to run. However a fan unit is already available, but to install it would result in an efficiency of 45%, absorbing some 220 kW at an operating cost of $41,000. It can be seen that the increased operating cost alone within a two year period would offset the purchase of the new fan.

Because mine workings are not stationary but continually changing, advancing and retreating, ventilation requirements are not constant. Because of changes in mine layout, production and work force the mine resistance (characteristics) changes. Therefore fan selection should be within a fluctuation range of application. In general a fan has a useful life of 15 to 20 years and matching consideration is usually made within this criteria.

The fan with the lower overall cost per year will be the most economical one to select and the best suited, all other factors being equal.

**Fans in Series and Parallel**
When two fans are so installed, one behind the other, that they handle the same air, they are said to be in series. When two fans are installed side by side in such a way that they draw air from the same source and deliver to the same destination they are said to be in parallel. (See Fig 19.9)

When two fans are in series the same volume of air passes through each and each adds a certain amount of pressure to it in the process. The advantage of fans in series is that where resistance to flow is high more efficient distribution and better operating conditions may be obtained.

The flow conditions for series operations are determined by mine resistance and the combined pressure characteristics of the fans, obtained by plotting the sum of the pressures for the same quantity. (See Figs 19.10 and 19.11)
Note however, that unless the two fans are properly selected for the work to be done in series operation, they will not work together at maximum efficiency, and will give poor results.

When two fans are installed in parallel they do not necessarily handle the same volume of air, but they must always produce the same pressure because they draw from a common point and deliver to a common point. In practice fans in parallel may draw from separate intakes sharing a common discharge. The combined performance characteristic of fans operating in parallel at the same point in the system is obtained by plotting the sum of the quantities handled at the same pressures against the pressures. The intersection of the mine or system characteristic with the combined characteristic determines the pressure because the pressure must be the same for both fans: this pressure determines the relative quantities each fan will produce.

However, should the mine characteristic (resistance curve) not intersect the combined characteristic the higher pressure fan will handle more air alone than the two fans together, and if operation in parallel is attempted the high pressure fan will reverse air through the low-pressure fan. (See Fig. 19.11)
Fans in Combination

In actual mine practice fan combinations are not always as simple as series and parallel operations. By combined operation each fan, although it may share an air way with another fan, has an individual zone of influence with intake and outlet. Consequently fans in multi-fan ventilation systems are partially in series when most of the air from one fan passes through the other but some air is added or removed between the two. Fans are partially in parallel when they draw air from the same place and discharge to the same place but instead of being near to the fans these common points are some distance away.

The procedure which can be adopted to ascertain the likely air flows is to use a residual fan pressure curve. This procedure is not dealt with in this preliminary coverage.

Effect of Natural Ventilation Pressure on Fan Characteristics

Natural ventilation pressure can be regarded as an additional fan usually in series with existing fans. N.V.P. depends upon the density difference between downcast and upcast air and on the depth of the mine. Graphically it can be presented as a straight line because it is only indirectly affected by the volume of air flowing.

When the N.V.P. tends to move the air in the same direction as the fan it is added to the fan pressure and when it tends to move the air in the opposite direction to the fan it is subtracted from the fan pressure as shown in Fig. 19.12.
Example:
Fan handles $75 \text{ m}^3/\text{s}$ at a pressure of $0.8 \text{ kPa}$ with zero N.V.P., some $83 \text{ m}^3/\text{s}$ when assisted by $0.3 \text{ kPa}$ N.V.P. and $65 \text{ m}^3/\text{s}$ at $0.9 \text{ kPa}$ when $0.3 \text{ kPa}$ N.V.P. acts against the fan. This explains why a mine fan often handles more air at lower pressure in winter than in summer, because the N.V.P. in winter is at its maximum value.

VENTILATION SYSTEMS

A mine ventilation system consists of the following essential components:

- pressure source (fan);
- connected ducts (mine openings);
- control devices (stoppings, doors, regulators etc.).

The chief function of mine ventilation is quantity control, which is concerned with the air movement, its direction and magnitude. Supplying the air in the desired amounts to the various working places in a mine is referred to as air distribution. It is accomplished by adopting a ventilation method and plan suitable for the mining method to be employed in exploiting the mineral deposit; both ventilation and mining method should be selected to complement each other.

![Figure 19.12 Effect of N.V.P. (natural ventilation pressure) on characteristic curves](image)
Effective distribution signifies that both direction and quantity of air are controlled. Location of pressure sources and control devices and interconnections of air ways will determine direction, and intensity of source and magnitude of resistance of openings will determine quantity. Means of control are afforded by modifying the pressure source and/or the mine openings.

**VENTILATION CONTROLS**

Effective air distribution is accomplished by use of various control devices in mine air ways. Their use and location should however constitute a minimum interference to the flow of mine traffic.

**STOPPINGS**

A stopping blocks off a mine opening to prevent flow of air. These may be temporary or permanent, depending upon length of surface, hence affecting the design and construction.

Temporary stoppings are of light-frame construction and moderately air tight. They are removable and if possible re-usable. Permanent stoppings may be equipped with access doors, and may be constructed of steel plate or masonry, i.e. brick work or concrete. These must be as air tight as possible because they may also serve the purpose of fire bulkheads in coal mines.

**DOORS**

A door is essentially a stopping with a movable partition to permit passage of personnel and equipment. Doors are often arranged in pairs to provide an air lock so that one will always be closed while the other is open.

The operation of doors are often manual opening but equipped with self closing mechanisms. Large doors may have pneumatic cylinders linked with automatic controls.

**CROSSINGS**

A crossing is a device to permit the passing of two streams of air at an intersection without mixing. These air bridges, especially overcasts, are used in coal mines. Crossings are erected to interfere with travel in one direction only, being placed across the entry or heading that is least frequently used.

**REGULATORS**

Regulators are used to control and redistribute the quantity of flow in each split. This is done by creating shock loss and restricting the passage of air through an airway. Regulators are often set in ventilation doors, consisting of a sliding panel and are usually placed on return air ways.

**AUXILIARY VENTILATION**

Where the main ventilation air stream is inadequate or unavailable, reliance must be placed on supplemented means of supplying air. The practice of augmenting the main ventilation system is termed auxiliary ventilation.

Auxiliary ventilation requirements are most evident in working places which are being advanced beyond the main ventilation air stream. Blind or dead end workings are characteristic of development headings and characterized by room and pillar and some other mining methods.

The common applications of auxiliary ventilation fall into three categories.
1. Supplying air to dead-end working places, both development and production (quantity control).

2. Supplying uncontaminated air to faces of working places in contaminated environments (quality control).

3. Supplying conditioned air to faces of working places in uncomfortably hot or cold environments (temperature humility control).

The ventilation of dead-end working places is the most frequent and important application of auxiliary ventilation. Drives, rises, shafts and winzes require auxiliary ventilation as do entries in coal mines which proceed beyond the last connecting crosscut.

Coal and uranium mines are examples in which auxiliary ventilation is required for quality control. Coal mines invariably liberate methane gas, and uranium produces radon gas. These strata gases can be maintained within allowable limits by dilution, but auxiliary ventilation means are the most practicable. In coal mines, reliance is placed on line brattices to direct sufficient quantities of diluting air to the working faces. Extremely dusty operations are also sometimes controlled through dilution by auxiliary ventilation.

Temperature-humidity control in hot deep mines may be achieved by auxiliary ventilation especially for development headings where high virgin rock temperatures are encountered.

**METHODS OF AUXILIARY VENTILATION**

Auxiliary ventilation discussed here relates mainly to dead end workings. However, multiple heading methods of development openings especially in bedded deposits can greatly alleviate the problem.

The three methods of ventilating faces of dead-end working places are

- line brattice;
- compressed air line or equipment exhaust;
- fan (or injector) and ventilation pipe or tubing.

The first is the usual practice in coal mines; the other two are common to metalliferous and non-metallic mines. However, the use of fan and vent tubing has increasing application in coal mines especially for continuous mining.

**Line Brattice**

The erection of a curtain longitudinally in an entry or a room effectively divides the opening in two. The line brattice is usually erected from the last through cross-cut to within a few metres of the working face, ventilating air can then be directed to the face along one side of the brattice and returned along the other side. This method has particular application to coal face ventilation but the maximum distance of advance past the last cross-cut is usually stated by mine legislation. (See Fig. 19.13)

Line brattice usually consists of fire resistant cloth-material; plastic sheeting is also used. Line brattice can be at best only a deflection device and leakage can be considerable. Liberation of strata gas or contaminants occurs mainly at the working face and it is essential that the auxiliary air stream is delivered as close as possible to the face so the air can sweep away impurities. Velocity decrease is critical especially as the brattice end moves away from the face.

Brattice is used extensively in coal mines, with little application to metal mines because of concussion damage from blasting.
Compressed Air and Injectors

The widespread use of compressed air for power in metal mines suggests it as an attractive supplement and substitute for ventilation. Exhaust from pneumatic drills, slushers and mucking machines helps the environment in hot humid and/or dusty working places. However, compressed air exhaust should never be accepted as a substitute for ventilation because these machines run only for a portion of a shift and air quantity is insufficient to remove air-borne contaminants.

While compressed air is frequently available in metal mines it is relatively expensive and even when used to operate fans or injectors it is a multiple of the cost of electricity (= IOx) as a source of power. Ventilation directly from a compressed air line is seldom warranted, however it has been used to purge rises and winzes to clear smoke and fumes immediately after blasting. Compressed air-water atomizers have particular value in allaying dust at points of dust generation but have limited effect on air-borne dust. Direct ventilation from an air line incurs a very high cost.
Fig. 19.14 illustrates types of compressed air injectors. Compressed air injectors have low head and produce low quantity and have low efficiency (≈ 10 - 15%). Advantages are zero maintenance, simple installation and ability to operate under adverse conditions. Injectors are limited to relatively short vent-pipe installations or those without vent tubing.

Fan and Ventilation Pipe
The use of fans attached to vent pipes or tubing, if the installation is properly maintained, is the most desirable method of auxiliary ventilation for dead end workings. Small electric-powered fans, while not comparable in efficiency to large fans, are still considerably more efficient than compressed air-injectors or fans or line brattices.

For developmental work in metal mining, a single length of vent pipe or tubing is suspended in the working place from hangers and attached to a small auxiliary fan. The location of the fan defines the operation term: if located so that the fan is at the inlet of the system it is a blower; if intermediate between intake and exhaust it is a booster; if at the exit it is an exhauster. Fig. 19.15 illustrates auxiliary fans blowing and exhausting headings. Decrease in air velocity beyond the vent pipe is significant. Note that blowing air is far superior to exhausting in creating a positive circulation and a safe comfortable environment at the face.

Rigid vent piping is required for exhausting systems to prevent tube collapse because it works on negative pressures.

Auxiliary fans should be protected against concussion damage from blasting by being switched off during actual blasting.

MINING METHOD AND VENTILATION SYSTEMS
In relating mining method and ventilation to ensure adequate amounts of fresh air at the face of all working places, certain principles are suggested:

- Co-ordinate mining and ventilation systems rather than superimpose ventilation on a pre-determined mining method.
- Utilize primary development openings as main air ways for ventilation; use other development as secondary air ways.
- Restrict the number of working places on one split, e.g. each stope having a separate intake and outlet.
Two groups of systems are distinctive enough to warrant further attention: these are horizontal, as associated with coal mining, and vertical and combined systems as associated with metal mines.

**HORIZONTAL SYSTEMS**

Horizontal systems are applied to mines whose workings lie essentially in a single horizon. Because almost all development headings lie within the deposit, horizontal systems generally use multiple parallel openings.

Room and pillar mining for coal mining is illustrated as an example of horizontal ventilation systems.

Room and pillar development consists of driving multiple parallel openings connected at intervals by cross cuts. This affords a systematic layout and efficient distribution for air to working faces both during development and exploitation.

Using multiple openings either uni-directional or bi-directional distribution may be employed. In uni-directional flow the air in adjacent openings flows in the same direction and is entirely fresh or exhaust air; while in bi-directional flow, the air in adjacent openings flows in opposite directions. To maintain separate flow in bi-directional distribution, stoppages must be erected in connecting cross cuts. Leakage is a major problem with bi-directional flow, hence the predominant use of uni-directional flow for horizontal systems. A neutral heading can be maintained for a conveyor installation. Fig. 19.16 shows single split and double split for entry driving.
VERTICAL SYSTEMS

Vertical ventilation systems are applied to mines whose workings lie essentially in a single plane. The majority of the metallic minerals fall in this category. Multiple headings are not frequently used in metal mines because of the expense of a second heading, which would have to be placed in waste or country rock. If multiple headings are driven within the deposit, the lower one on each level is usually used for haulage and the upper as a grizzly or an air way.

In steeply inclined deposits, mining generally originates from levels (or horizons) spaced at equal intervals down the dip and parallel to the strike. Alternate levels are used as intakes and returns using uni-directional distribution. Because it is usually required to have more than one surface connection, alternate shafts can act as downcasts and upcasts to effect uni-directional flow with balanced resistance. Ideally a system has a minimum of three surface connections: the central opening is made as intake and the peripheral openings are exhausts. (See Fig. 19.17)

Fundamentally, there are three ventilation plans for vertical mining methods depending upon the number of access openings into each working place and the availability of inter-level connections. Fig 19.18, Ventilation of working places, illustrates the basic choices available.

Multiple heading ventilation using a single split (line brattice and curtain stopping)

Multiple heading ventilation using double split (line brattice and curtain stopping)

Figure 19.16 Single and double split ventilation of multiple (coal) headings

Figure 19.17 Vertical plane ventilation depicting an ideal retreating mining method
Method (1) consists of stopes with inter-level connections.

Method (2) consists of stopes with no connections between levels but having two access openings.

Method (3) consists of stopes with no connections between levels and having only one access opening.

In the first method air is admitted to the working place from the level below and exhausted to the level above. This is the preferred approach because each stope receives a separate split of air. However separate exhaust openings may be necessary to prevent contaminated air from entering other stopes especially where advancing stoping outward from the downcast shaft is practised.

In the second method air is admitted to the working place from the level below and is returned to it at downstream. This is not desirable because each stope cannot be on a separate split where there is more than one active stope per level.

In the third method, auxiliary ventilation usually consisting of a blower fan and vent tubing is employed to direct air to the working place. This is the least desirable method because the main ventilation stream does not reach the working face and requires similar stopes to share the same air.

There are two unwanted problems in ventilating metal mines. One is locating and maintaining shafts and air ways in ground which will not be caved and lost. The other is preventing short circuitry and dust hazard created by the leakage of air up or down ore passes or chutes when partially filled or empty.

Actual ventilation layout will largely be variants of the methods mentioned above depending upon the mining method.
COMBINED VERTICAL – HORIZONTAL SYSTEMS
Mines with extensive workings on levels as well as being between levels are ventilated by combined vertical – horizontal systems. Massive and irregularly shaped deposits are typical of this system. The objective is to supply fresh air to each level from a centrally located downcast shaft, radiating outward and upward through the working places to exhaust air ways leading to upcast shafts.

TEMPERATURE – HUMIDITY CONTROL
Temperature - humidity control is one of the three functions of air conditioning, and is essentially heat control. Air conditioning is the simultaneous control with prescribed limits, of the quality, quantity, and temperature - humidity of the air within a designated space such as a mine. Temperature - humidity control is akin to quality control in that it pertains to the physical quality of the air, while quality control pertains to the chemical quality of the air.

The usual reason for employing temperature -humidity air conditioning in mines is for comfort rather than product purposes. The heat content of the mine air is maintained within prescribed limits for the comfort and working efficiency of human beings.

Some special circumstances such as occur in rock salt mines where moisture absorption can affect stability, may require air conditioning.

A major limitation to mining is depth and a factor associated with this is heat. Therefore the process of cooling and dehumidification (temperature – humidity) control is important.

SOURCES OF HEAT
A number of sources of heat are recognized:

- adiabatic compression,
- surrounding strata,
- machinery and equipment,
- heat produced by workmen,
- chemical processes (blasting, oxidation),
- rock movement.

Adiabatic compression is an unavoidable heat source associated with the pressure increases with air descending a shaft (often termed auto-compression) and is of the order of 2° per 300 metres of vertical depth.

Strata surrounding the mine opening is a major and unlimited heat source affecting underground air temperatures. Geothermal gradient varies from place to place but signifies an increase in rock temperature with depth.

Machinery equipment generates heat which is the equivalent of the difference in total energy and energy absorbed in useful work. A large proportion of work underground involves friction hence energy appears as heat. Where possible, heat from machines such as diesel equipment needs to be directed to return airways.

Heat from workmen arises from the metabolism processes within the human body. Chemical and oxidation processes produce waste heat; it is the mechanisms of heat liberation that regulate the body temperature.
Chemical processes such as the oxidation of minerals and timber can contribute heat and, like blasting, should be directed into return airways.

Rock movement may cause heat which is likely to be absorbed by the rock mass with little direct transfer to mine air.

**PHYSIOLOGICAL EFFECTS OF HEAT AND HUMIDITY**

The human body has very intricate and very efficient heat-regulating mechanisms, which strive to keep the body temperature at about 37° C.

In order for any object to maintain a constant temperature, its heat gain and heat loss must be equal. The human body is constantly gaining heat from its internal metabolism, from the intake of hot food and drink and from the surrounding atmosphere and objects. To balance these heat gains the body loses heat mainly to the surrounding atmosphere.

A major factor in the production of metabolic heat is the degree of physical exertion undertaken. A person doing heavy work can produce up to 450 W of waste heat. This heat may be transferred to surrounding air and objects by the heat-transfer processes of convection, radiation and/or evaporation.

Acclimatization of miners to hard work under hot conditions is essential to avoid the possible dangers and ill effects of heat such as dehydration, exhaustion and possible heat stroke.

There are associated undesirable effects which indirectly occur because of excessively high temperature – humidity environments. Also because of the inability of a miner to produce at his normal rate in hot, humid surroundings, efficiency of work decreases. There is also an attendant increase in carelessness and accidents because miners become less alert. Both these effects are costly in terms of production, medical fees, safety and worker morale.

**PSYCHROMETRY**

The determination of the psychrometric properties of air at a given point is termed psychrometry. A sound knowledge of the basis of this is required in temperature humidity control where the properties of air undergo changes through the processes of heating, cooling, drying and humidification. Hence psychrometry is the thermodynamics of air – vapour mixtures.

The determination of the properties of air at given conditions is termed the state point and is the pre-requisite to the solution of problems involving temperature humidity control processes. Two aids are used to find these properties: psychrometric tables and psychrometric charts. Because the mass of air is moving and the density is not constant because it changes with heat/temperature changes, the quantity of flow is not a reliable reference. Therefore flow rates are calculated on a flow mass, hence the total heat involved can be determined. A detailed coverage on psychrometric processes is not appropriate in this text (Mining 1).

**MINE AIR CONDITIONING**

The improvement of underground environment conditions can lead to an increase in worker efficiency and reduce possible health hazards through improved mining practices and ventilation systems. The economics of any such system needs of course to be well researched prior to a definite decision being made.
Mine ventilation improvements which may postpone or eliminate the need for mine refrigeration are:

- increasing air volume,
- insulating airways,
- changing the mine layout.

Increasing air volume can reduce air-strata contact time and diffuse the heat throughout a larger air mass. However, because power requirements vary as the cube of the air volume, it may be necessary to try to reduce mine resistance by streamlining shafts and reducing leakage. Control of water intake in a mine may help reduce the wet bulb temperature – humidity in a mine.

Ventilation tubing can be insulated in hot areas to reduce heat absorption prior to ventilation of development faces.

A significant improvement in ventilation environment can be achieved by directing heat into return airways especially from heat sources such as mechanical equipment installations.

**MINE AIR COOLING**

Mine air can be cooled on the surface before it enters the downcast shaft, or underground at or near the bottom of the downcast shaft, or in the working places.

A mine air conditioning system consists of a series or combination of devices which remove heat from the mine atmosphere and transfer it to isolated areas e.g. to return airways or water as an exchange medium to waste at the surface. The individual components and component arrangement vary from system to system but usually include combinations of cooling towers, mechanical-refrigeration units, underground heat exchanges and air coolers.

Surface air cooling is the cheapest and easiest application of refrigeration because maintenance and supervision is easy with ready disposal of waste heat. Although effective cooling is limited to factors of heat absorption due to temperature difference, it is offset by a large amount of contact with the rock surface as air travels to the work face.

Underground cooling may be handled by bulk underground cooling or working place cooling. Bulk underground cooling may be done at the bottom of the mine and allows the plant to run at full load through the year, whereas a surface plant achieves limited effective cooling in winter. Waste heat can be a problem to dispose of underground. Working place cooling utilizes a central underground plant with cool water circulated to work places through a heat exchanger. A fan blows air through the cooler hence cooling the work face. A disadvantage is the high capital cost of reticulation.

Other approaches have been to chill the service water which can be as much as the mass of ore extracted. However this method may only at best account for between 10 and 30% of the required cooling in deep hot mines. Surface or underground bulk plants are still required.

Whatever the approach to mine air conditioning, the objective of a satisfactory, safe atmospheric environment for human workers, must never be lost sight of. Its role must be related to the overall requirements of in-mine environment.
VENTILATION REGULATIONS

Metalliferous mines in Western Australia are required to comply with the minimum requirements for ventilation as set down by the Mines Regulation Act and Regulations. Part 8 - Ventilation and Control of Dust and Atmospheric Contaminants of Division A - Regulations to be observed in all mines are the major topics for attention.

Note the requirement of Regulation 8.12 for underground working. The difference between wet and dry bulb temperature up to a dry bulb temperature of 26.5° should be at least one degree whilst above this the dry bulb reading should rise by one for each half a degree rise in the wet bulb.

Regulation 8.20 permits inspectors to require installation of auxiliary ventilation into work faces. Development headings and the dispersal of fumes require ventilation installation as specified in Regulation 8.24(1); however in rises, shafts and winzes, compressed air purge lines may be used.

The provision of independent airways from the lowest level and the routing of ventilation as directly as practicable to the return airway is covered under Regulation 8.22.

The provisions of separate spits for each level workings, with exhausts to a return airway, is specified under Reg. 8.28.

The use of air doors, stoppings, cross-overs and brattice as approved for use in mines and the manner of ventilation of stopes are detailed in Regulations 8.29 and 8.30.

Each Mine Manager is required to keep at the mine, plans and sections on which the direction, course and quantity of air and the position of all ventilation equipment devices are recorded.

The Ventilation of Coal Mines within Western Australia falls under the legislation of the Coal Mines Regulation Act and Regulations. Division VIII – Ventilation Regulations 116 to 130 cover the minimum requirements for mine ventilation.

These Regulations cover provisions for the division of underground mines into ventilation districts or splits, provision for clean air intake into a mine, the use of stopings, air crossings and brattice.

Intake airways and return airways are to be provided with ventilation doors which form an air lock when entering or leaving.

Provisions for regular pre-shift inspection by deputies for the presence of gas are stated as well as the inspection of areas outside normal Deputies' Districts.
CHAPTER 20 – Mining Regulations – Mine Safety

MINES REGULATIONS (METALLIFEROUS)

The current legislation in Western Australia which deals specifically with mining safety, hygiene and welfare of workers in metalliferous and quarrying operations is the Mines Regulation Act 1946 - 1974 and Regulations.

Every person directly involved in working in a mine should be familiar with at least the basic requirements of the regulation. Management has particular commitments under requirements of the Act. Particular emphasis is placed on safety and fundamentally each individual is responsible for his own safety and is required to report to a person in authority or management any unsafe conditions or equipment likely to endanger himself or others.

THE MINES REGULATION ACT

The Mines Regulation Act is divided into the following Divisions:

Division I  - Preliminary, ss 4-5
Division 2  - Inspection, ss 6-23
Division 2A  - Health, ss 23A-23F
Division 3  - Management, ss 24-30
Division 4  - Accidents, ss 31-35
Division 5  - Employment, ss 36-41
Division 6  - Sunday Labour, ss 42-45
Division 7  - Engine Drivers and Machinery Operators, ss 46-46A
Division 8  - Plans, ss 47-48
Division 9  - Miscellaneous, ss 49-61.

Division 1 – Preliminary, is a list of definitions applicable to the Act and Regulations.

Division 2 – Inspection, identifies the types of inspectors, their powers and requirements such as an inspection record book.

Division 2 – Health, relates specifically to occupational health especially associated with ventilation – environmental atmosphere and respiratory ailments.

Division 3 – Management. This specifies important categories of management requirements for mines and quarries.

A registered manager is required for underground or quarry workings with 25 or more persons employed. An underground manager, or a quarry manager respectively, is required for mines or quarries with more than 25 men, but he may also be the registered manager.

The Underground Manager for underground operations where more than 25 men are employed is required to be the holder of a First Class Mine Manager's Certificate of Competency.
The quarry manager is required to be either the holder of a First Class Mine Manager's Certificate of Competency or a Quarry Manager's Certificate of Competency.

For underground mines with less than 25 men underground, the underground manager is required to be the holder of an Underground Supervisor's Certificate of Competency.

For quarry operations with less than 25 men employed, the quarry manager is required to be either the holder of a First Class Mine Manager's Certificate, Quarry Manager's Certificate or a restricted Quarry Manager's Certificate of Competency or Service.

The duties and responsibilities of management are defined with reference to the Act and Regulations.

Division 4 – Accidents, specifies the conditions of accident notification to both the District Inspector of Mines and the union to which the employee belongs.

Accidents or injury such as fuming or arising out of explosive or blasting agents must be treated as serious. A record book must be maintained which notes the occurrence of mishaps as well as accidents. Ground subsidence, underground fires, flooding, failure of hoisting equipment, accidental ignition of explosives or similar occurrences need to be recorded. Where injury arises, notice must be given to the District Inspector.

Division 5 – Employment, states the permissible hours of employment duration, both daily and weekly. Employment underground is normally limited to a maximum of six, seven-and-half-hour shifts per week.

The minimum age for normal full-time employment underground is 18 years.

Division 6 – Sunday Labour Underground, restricts Sunday work to essential maintenance and emergencies and prohibits productive work on that day.

In order to work on Sundays underground it is necessary to obtain a permit of authorization from the District (Mines) Inspector.

Division 7 – Engine Drivers and Machinery Operators, requires that winder drivers be certificated under the Machinery Act. It also has provisions for specific equipment within the Regulation such as vehicles, and shovel operation.

Division 8 – Plans, requires Management to furnish mine plans generally on a six monthly basis but at least once, by March each year. Plans need to be certified by an Authorized Mine Surveyor prior to submission.

Division 9 – Miscellaneous, Section 49 is significant because it identifies the certain responsibility of safety to the individual mine/quarry employee. Section 61 provides the points under the Act to make regulations relevant to the Act.

REGULATIONS
The Mines Regulation Act Regulations currently in force came into existence in April 1976. It is these regulations that specify the requirements and manner of administrating the provisions of the Act.

The regulations are divided into a number of parts and divisions applicable to certain areas of operation under the control of the Act.
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The most relevant Divisions applicable to Mining I involved in underground mining are A and B. Division C will be of relevance where the student is in surface mining operations and not underground.
PART 1 – PRELIMINARY
Regulation 1.7 relates to regulations which come under provisions of the Inspection of Machinery Act. A printed copy of Regulations (poster) is required to be displayed in the mine office or similar place on all operating mines.

Division A – Regulations to be Observed in All Mines.

PART 2 – INSPECTION
The Regulations in this part cover the appointment of Inspectors of Mines and their responsibility to the Minister of Mines in detail.

Workmen’s Inspectors are elected by mines employees under the 1967 Electoral Act. Workmen’s inspectors are responsible for the examination and enquiry into whether provisions of the Act and Regulations are being maintained where men are employed.

PART 3 – MANAGEMENT AND SUPERVISION
Provisions under this part deal with the types of Statutory Certification for: First Class Mine Manager’s Certificate of Competency; Quarry Manager’s Certificate of Competency; Underground Supervisor’s Certificate of Competency; Restricted Quarry Manager’s Certificate of Competency.

Initially some of these certifications could be obtained by conditions of service but this no longer applies.

PART 4 – GENERAL SAFETY REGULATIONS
This part specifies requirements for provision of adequate First Aid equipment and facilities, dependent upon the number of employees involved.

All mines are required to provide rescue equipment and breathing apparatus unless given exemption by the District Inspector.

All employees have to be provided with and wear safety helmets where operations are:

• underground in a mine,
• crushing or treatment,
• quarry excavation, or other construction site.

Provisions for safety (harness belts) require them to be used where there is danger of falling from a height.

PART 5 – ELECTRICITY IN MINES
This part specifies requirements for electricity supply, usage and protection from electrical hazards in mines. Electrical installations within mines come under the control of the State Energy Commission of Western Australia as well as under the provisions of the Mines Regulations on voltage and types of cables.

PART 6 – MACHINERY IN MINES
Fundamentally all mechanical equipment within mines is required to be kept in good repair and to be operated in a safe manner. Certain types of machinery that requires appropriate certificates for authorized operation includes: winders, cranes, power shovels, underground locomotives and diesel engine vehicles underground. In particular, machinery operators on leaving machinery unattended, are required to render it safe.
PART 7 – EXPLOSIVES AND BLASTING AGENTS
Specific definitions relative to types of explosive and terms are given. Explosives come under the control of the Chief Inspector of Explosives of the Explosives Branch of the Mines Department, who is appointed under the Explosives and Dangerous Goods Act. Uses of explosives in mining have to comply with both the provisions of that legislation and Part 7 of the Mines Regulations.

Significant aspects of this Part are:
- Minimum age of explosives handlers is 18 years;
- Explosive users are to be competent to handle explosives;
- Separate storage of detonators from explosives;
- Detonating fuse (cord) is classified as an explosive;
- Minimum length for safety fuse and manner of lighting of such;
- Drilling precautions in headings where butts are present;
- Firing times applicable to underground and surface operations;
- Electrical firing – manner and type of circuit and exploder devices;
- The safety precautions to be taken when misfires occur or are suspected to have occurred.
- Minimum re-approach times to fired faces.

PART 8 – VENTILATION, AND CONTROL OF DUST AND ATMOSPHERIC CONTAMINANTS
This part is supplementary to the provisions of the Clean Air Act 1964.

Operating mines are required to have a ventilation officer either where diesel equipment operates underground, or where required to appoint one by the District Inspector.

Provisions are laid down for:
- Duties of ventilation officers;
- Standards of air purity;
- Air in working places – standards of temperature;
- Suppression of dust in and about mines;
- Ventilation for installations.

Also of significance are the requirements to wash down headings prior to working on re-entry and maintenance of ventilation in development headings.

Ventilation plans are required to be kept with details of:
- the direction and course of air flow volume,
- positions of ventilation and doors.

PART 9 – OCCUPATIONAL DISEASES
Certain mines fall within different categories of possible hazards to respiratory health. Fundamentally, however, all mine workers are required to have a Mine Worker’s Health Certificate. This involves physical examination and chest X-Ray to detect possible respiratory diseases such as pneumonoconiosis.
PART 10 – SURVEYS AND PLANS
These have particular relevance to Mine Surveyors and indicate the provision of obtaining Authorization as a Mine Surveyor and the areas applicable to such certification. The types of plans required and order of accuracy of survey work is specified.

PART 11 – HYGIENE AND SANITATION
This deals with requirements on sanitation and water supply to underground mines. Specific requirements for change house conditions are specified.

Division B - Regulations to be Observed in Underground Mining.

PART 12 – SAFETY AND PROTECTION (UNDERGROUND)
A significant provision within this part is the requirement that all mine workers underground be instructed in work duty and are competent to carry out that work.

Other provisions of note are:
- Lamps for underground workers;
- Dual access to mine workings;
- Stopes to have two travelling ways;
- Where a potential danger exists with a mine or section the workmen will be withdrawn;
- Shaft entrances to have safety fencing and gates;
- Dangerous accumulations of water should be tested for by advance probe drilling;
- Travelling way requirements – protection and ladder installations.

PART 13 – LOADING AND TRANSPORT (UNDERGROUND)
This part deals with excavation and haulage equipment underground. The increased use of trackless diesel equipment attracts particular attention to aspects such as: operating condition - maintenance, speed limits, driver certification and manner of operation. Underground locomotives also require driver certification as well as battery charging stations. Locomotive track gradients are not permitted to exceed 1 in 12. Excessive gradients generally greater than 1 in 7, such as 1 in 5 for trackless haulage require approval from the District Inspector. Minimum equipment clearances are specified in haulage drives.

PART 14 – DIESEL ENGINES (UNDERGROUND)
Only compression ignition diesel engines are permitted to be used underground in mines. Each engine is required to have a permit and is subject to testing prior to approval. Stringent ventilation requirements on the volume of air required for dilution of exhaust fumes and the permissible levels of noxious or hazardous gases are specified. Fuelling and servicing of diesel equipment likewise attracts specific control requirements.

PART 15 – WINDING, WINDING ROPES AND SIGNALS
Head I – Regulations of General Application.

Fundamentally all winder drivers require Certification under provisions of the Machinery Act. The mode of operation and control features of winders are specified.
Also of importance is the use of a Code of Signals for winder communication. Every person employed underground in a mine should be acquainted with the Code of Signals where shaft hoisting is practised.

Basic Code: See Table 20.1

There are other basic code signals for specific operations such as for shaft sinking.

Hoist or winding rope safety factors are quoted for the different types of installations, i.e. drum winders and friction hoists. In-shaft conveyances and attachments must meet certain tests and standards. Although travelling with material in a conveyance is not permitted, the surveyor may travel, with his instruments, in an empty conveyance.

Head II – Drum winding operations.
Requirements on deriving rope safety factors and drum winder installations are specified. Drum winder installations require detaching hooks and safety dogs unless exempted. Where exempted non-destructive rope testing on a periodic basis is required.

Head 111 – Friction Winding Operations.

This part applies to Koepe type – friction hoists.

PART 16 – SHAFT SINKING
Shaft sinking operations require approval, prior to commencement, in writing from the Senior Inspector. Equipment usage, installations and protection such as pentices for sinking below levels being worked are specified.

Division C – Regulations to be observed in and about Quarries

PART 17 – SAFETY AND PROTECTION (QUARRIES)
This part has provision for the employment of persons within quarries, certification of power shovel operators, motor vehicle drivers and requirements for vehicle operational safety. Loading and dumping as well as lighting are particularly emphasised.

Permissible vertical quarry heights of 20 metres may only be exceeded with written permission after inspection by the District Inspector.

Sand pits have requirements for safety fencing in built up areas as well as limits for face heights.
## CODE OF SIGNALS

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<th>Shall Signify</th>
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<tr>
<td>1</td>
<td>Stop – Signal to be returned by driver when conveyance is or has been brought to rest.</td>
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<td>Lower.</td>
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<td>Raise.</td>
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<td>Hoist to surface.</td>
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<td>5</td>
<td>Danger Signal – The conveyance must not be moved until the release signal 8 has been given.</td>
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<td>6</td>
<td>Materials or equipment to be conveyed (cautionary signal). Signal to be returned by driver before a command signal is given.</td>
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<tr>
<td>7</td>
<td>Firing warning.</td>
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<td>8</td>
<td>Release from ‘danger’ signal 5. Signal to be returned by driver before a command signal is given.</td>
</tr>
<tr>
<td>12</td>
<td>Accident signal – to be followed after a pause by the signal for the level where the conveyance is required.</td>
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### COMMAND - LEVEL SIGNALS (examples)

- 1 pause 1 to No. 1 level
- 1 pause 2 to No. 2 level
- 2 pause 1 to No. 6 level
- 2 pause 2 to No. 7 level
- 8 pause 1 to No. 36 level

*Table 20.1 Code of signals*

In-pit haulage, gradients, road widths, and safety signposting are required to be sufficient to the satisfaction of the Mines Inspector.

### MINES REGULATIONS (COAL)

The legislation in Western Australia applicable to the safety, hygiene and welfare of workers in coal mining is the Coal Mines Regulation Act, 1946-1976 and Regulations.

Personnel directly associated with coal mining should be familiar with the basic requirements of these regulations. Conditions experienced at Collie - the only productive coal field in Western Australia at present - are significantly different from those of most coal mining fields. However stringent safety conditions are still required.

The Coal Mines Regulation Act is divided into the following Divisions:

- Division 1 – Preliminary, ss 1-5
- Division 2 – Inspectors, ss 6-18
- Division 3 – Management, ss 19-22
- Division 4 – Accidents, ss 23
- Division 5 – Employment, ss 24-33
Division 6 – Sunday Labour ss 34
Division 7 – Plans, ss 35-37
Division 8 – Accident Relief and Superannuation, ss 38
Division 9 – Miscellaneous, ss 39-69.

Division 1 – Preliminary, is a listing of definitions applicable to the Act and Regulations.

Division 2 – Inspectors, identifies the types of inspectors, powers and duties.

Division 3 – Management, this categorises the types of management for coal mines.

A manager of a mine (underground and coal) must possess a First Class Mine Manager’s Certificate of Competency (Coal).

A manager of an open cut (coal) must possess a First Class Mine Manager’s Certificate of Competency (coal) or an Open Cut Mine Manager’s Certificate of Competency.

It is permissible for a superintendent to have control of two or more mines provided they have managers. This superintendent must however possess either a First Class Manager’s Certificate where the mines are underground or an Open Cut Manager’s Certificate if the operations are open cut.

The manager is relieved by an Under manager who has to possess a First Class Certificate or a Second Class Mine Manager’s Certificate of competency for underground mines.

For open cut mines the under manager has to be the holder of either a First Class Certificate or Second Class Certificate or Open Cut Manager’s Certificate.

A further deputization within management for coal mines occurs with the provision for Mine Deputies. A mine deputy underground must possess a First Class Manager’s Certificate, a Second Class Manager’s Certificate or a Third Class or Deputy’s Certificate of Competency.

For an open cut, the deputy must possess the above statutory certificates or an Open Cut Manager’s Certificate or a Deputy’s (open cut) Certificate of Competency.

The duties and responsibilities of management are defined with reference to the Act and Regulations.

Division 4 – Accidents, requires that all accidents are reported and inspection and inquiry into accidents conducted.

Division 5 – Employment - states the permissible hours of employment for mine workers. The maximum permissible time per week underground is 42 hours, with a limit of 7 hour hours per shift. The minimum age for employment in or about a mine is 15 years. Piece work payment systems on a weight basis have particular constraints, while Ministerial approval for incentive payment systems is necessary.

Division 6 – Sunday Labour - Sunday work is restricted to essential maintenance and emergencies with production work being prohibited on that day.

Division 7 – Returns, Plan, Notices and Abandonment - Production returns for each month must be furnished by the mine-owner or manager to the District Inspector, including names of owner, manager, overman, the number of workers employed, the quantity of material produced and its value. Accurate up to date working plans are required to be kept at the mine office. Provided with this must be an overlay which details surface features, rivers, swamps, installation and buildings.
All seams or mines on abandonment require the management to supply the District Inspector with plans and sections detailing workings and their boundaries.

Plans need to be certified by an Authorized Coal Mine Surveyor prior to submission.

Division 8 – Accident relief and superannuation - has provisions for a Coal Mines Accident Relief Fund, to which all employees contribute, which provides a fund for payment of compensation arising out of injury or contraction of disease attributed to coal mining.

Division 9 – Miscellaneous - A significant feature of Division 9 is the details on statutory certification for:

- First Class Mine Manager’s Certificate of Competency.
- Second Class Mine Manager’s Certificate of Competency.
- Third Class or Deputy’s Certificate of Competency.
- Open Cut Mine Manager’s Certificate of Competency.
- Deputy’s (Open Cut) Certificate of Competency.

Another significant feature within the Act is the provision for personnel to be able to gain their Statutory Certification by passing prescribed examinations set through the Board of Examiners (coal).

There are specific requirements for Arbitration should disputes arise between management and employees.

Section 49 provides the power under the Act to make regulations relevant to the Act.

Each mine may be subject to Special Rules related to the particular circumstances for such mines.

REGULATIONS

The existing Coal Mines Regulation Act Regulations originally came into force in 1946. Amendments are published in the Government Gazette from time to time. It is these regulations which specify the requirements and manner of administering the Act. The regulations have a number of major divisions within which are regulations applicable to those specific areas.

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Division I – Preliminary - The Coal Mine Regulations do not affect the provisions of the Machinery Act which applies to machinery within or about a mine.

A printed copy of the Regulations (poster) is required to be displayed at the mine site.

Division II – Inspection - This division covers the requirements of Departmental Mines Inspectors and Workmen's Inspectors.

Division III – Conduct of examinations - This group of regulations covers the requirements and conduct of statutory examinations for:

- Third Class or Deputy's Certificate of Competency,
- First Class Mine Manager's Certificate of Competency.
- Second Class Mine Manager's Certificate of Competency.
  and
- First Class Open Cut Mine Manager's Certificate of Competency.
- Deputy's (Open Cut) Certificate of Competency.

Division IV – Explosives - Explosive usage, handling and storage comes under the control of the Chief inspector of Explosives of the Explosives Branch of the Mines Department. Provisions of the Explosives and Dangerous Goods Act apply as well as those specified by the Coal Mines Regulations.

The duties and responsibilities of shot-firers in coal mining are defined. Unlike metalliferous regulation requirements where firing times are restricted to specific times, unless special firing to clear obstructions, face blasting in coal mining may be done at any time provided safety precautions are observed.

Division V – Safety and Protection – Shafts, Tunnels or Outlets

Significant provisions within this division cover aspects such as:

- Mine access - where possible two alternative routes – shafts or outlets;
- Support of roofs and sides – timbering;
• Provision of lamps for underground use;
• Withdrawal of workers in case of damage;
• Likely encounter of accumulations of water;
• Personnel safety gear provisions.

Division VI - Ladders and Travelling Ways - The type and installation of ladders must meet the approval of the inspector. Travelling ways have minimum specified dimensions, and are to have provisions for personnel refuge – manholes for clearance from vehicles passing.

Division VII – Winding and Signals - Where shaft or inclined planes are used specific requirements on equipment and operation are given.

Division VIII – Ventilation - The quantity of air required per man underground is specified. The minimum oxygen content and permissible levels of contaminants are stated. The use of brattices, ventilation doors, and intake and return airways is subject to particular requirements.

Division IX – Sanitation and Hygiene - Requirements for water drainage, crib places, drinking water, waste removal and sanitation are specified.

Division X – Change Houses - The provision of change houses, construction, and provisions are specified.

Division XI – Electricity in Mines - The voltages permitted, safety requirements and the requirements for electricians to have charge of electrical machinery and apparatus in and about a mine are specified.

Division XII – Plans and Surveys - These have particular relevance to Coal Mine Surveyors and cover the provisions for authorization to sign mine plans. The type of plans required and order of accuracy of survey work are specified.

Division XIII – Accident relief fund - This division covers the provisions of such a fund for coal mine works.

Division XIV – Miscellaneous - Provisions for first aid facilities are specified. A significant feature of this group of regulations is Reg. 241 which requires coal of greater thickness than 1.5 metres to be worked on the panel system by bord and pillar method.

The minimum pillar sizes are specified; the permissible coal recovery percentage on first working of coal may be determined by a select committee.

Monthly returns stating the lease, mine official, men employed, mined area, coal produced and the estimated value must be submitted on the prescribed schedule form.

Division XV – Trolley wire locomotive - Covers the approval, operation and safety provisions for electric locomotives underground. Where fire damp/ (methane) or inflammable gas is detected in excess of 1.25% then the operation of locomotives is prohibited.
CHAPTER 21 – Mineralization Assessment (Sampling)

INTRODUCTION
The evaluation of a mineral deposit is done by means of sampling procedures. Three
dimensional sampling and location of mineralization below the ground surface (at depth)
is done by drilling e.g. diamond core drilling. Cores or chip samples obtained can be
examined and sampled for analysis or assay. Initial field testing of this nature is termed
pilot or reconnaissance sampling. This type of testing may involve systematic sample
testing for trace elements, i.e. at fixed intervals of depth as well as to identify any
mineralization intersected. Mineralization or a mineral deposit does not become known as
an ore body unless it has the potential to be exploited economically by mining. The cost of
mining and extraction (treatment) must be weighed against recoverable value.

To prove ore or to delineate the boundaries of the mineralization more closely, sampling
procedures are intensified; i.e. more holes are put down on a grid pattern so that a greater
number of samples/assays can be made to delineate any block of ground. A block is
generally a rectangular solid section of specified dimensions. It consists of a quantity of
ground within a square or rectangular grid with a vertical projection either from the
surface to specified depth or thickness of ore in the block.

SAMPLE
A sample is a small fraction of material, prepared with care and skill, which contains the
same content or proportion of the wanted material as occurs in the entire original bulk of
material from which the sample is taken. The sample is therefore representative of a certain
bulk of material from a certain location. A sample is identified by a code or number which
establishes its location relative to the material mass.

SPECIMEN
A specimen is not a sample because it is not representative of the bulk but is a selected
portion and is not usually identifiable by location coding.

RELATION OF SAMPLE FINENESS TO BULK
Sampling is done to establish the potential worth of a mineral deposit.
The greater the number of samples taken, the more accurate the final assessment of the
value of the whole will be. Systematic sampling programmes are implemented and
samples are collected by a variety of techniques. As an example, a fast moving conveyor
belt carrying a crushed product may have a scraper device which removes \( \frac{1}{12} \) of the
material on the belt, hence one tonne is detained for every 12 tonnes conveyed. Generally
one tonne is too great for handling so the sample is mixed and subsequent reductions are
done to reduce the bulk probably down to about \( \frac{1}{12} \) of a tonne.

The sample so obtained is only accurate if the material is well mixed and finely ground. If
coarse fragments of variable composition are part of the whole, the valuable material
content in the fraction or sample may not be representative of the content in the original
bulk from which the sample was taken.
The significant factors of sampling are thus accuracy of analysis of the material and the particle size of the material being sampled. The smaller the sample quantity in relation to the total amount, the finer the material has to be.

A gold ore sample to be assayed is about 30 grams. For ordinary ores, the gold obtained is likely to be 10 mg or less. This gold is recovered in the assay procedure (say in the case of fire assay) and weighed and its proportion is reported in terms of grams of gold to the tonne of ore. Traditionally this was done in terms of troy ozs per ton of ore.

Note 14 troy ozs = 1 pound (Avoirdupois)
20 pennyweights = 1 troy oz.

Also it was important to know the type of ton referred to, i.e. whether a short ton of 2000 pounds or a long ton of 2240 pounds (lbs). In metric terms precious metals are reported in grams per metric tonne. (1 troy oz being equivalent to 31.10 grams)

Since grades of gold ores are profitable even at concentrations of around 5 grams per tonne, very fine grinding is necessary to ensure gold liberation and to ensure the sample obtained is representative. The above grade represents a ratio of 1 : 200,000 i.e. gold to gangue (waste). Note grams per tonne is numerically equal to parts per million.

Fineness of grinding for accurate sampling is dependent on the amount of valuable material in proportion to the waste material. For gold ores, silver, platinum and tin minerals which may exist only as fractional percentages then very fine grinding of the sample prior to final assay is necessary.

For ores of nickel (metal content $0.6 - 3\%$), copper ($0.5 - 3.5\%$), zinc ($2 - 6\%$), aluminium ($20 - 50\%$) and iron ($50 - 60\%$), fineness of grinding is replaced by crushing sizes. Only in final analysis is fine size reduction necessary.

**SAMPLING ALLUVIAL MATERIALS**

Alluvial clays and silts which contain valuable materials like gold, tin or gemstones are sampled without sample size reduction (grinding).

The method of assaying is to take a large bulk as a sample and pass it through concentrating devices after the scalping off (removal) of very coarse particles. The amount collected in the sluices, jigs or concentrating tables is then proportioned out over unit volume of the deposit so sampled. With sufficient numbers of determinations of this sort, the likely recoverable commodity for an area may be gauged. For example, take one cubic metre of material and recover the valuable material. Weigh the valuable material recovered and use this as a measure of a large area. This then is repeated, obtaining samples from grid locations usually at specified uniform intervals. Hence a composite average can be obtained and applied to the total mass sampled.

Gemstones which occur as alluvials or placers are tested in this manner. Size reduction processes need to be controlled to avoid large valuable (prize) stones from being crushed.

**SAMPLING A WORKING FACE**

As soon as the sample has been obtained from a working face/exposure of an ore body it is identified by a number, with its location and other detail recorded by the sampler. The assay of the sample (sample grade or value) is thus meaningful or translatable to that particular location with the mine. Sample/assay results may be completed in an assay ledger as well as being plotted on an assay plan.
The more frequent the assay samples then the greater is the accuracy of the sample result. Obviously the ultimate sample result is the value after treatment of recovered material. However this is not practicable or applicable and a compromise must be made. Sampling is generally made at each face or round for an underground mine development heading if driving on ore.

Close sampling is a systematic procedure done at the exposed surfaces of the ore body.

**SAMPLING PROCEDURES – TYPES**

**Groove Samples or samples collected by channelling**
A moil is a chisel-like tool made of drill steel, shaped by forging. The point may be diamond shaped for hard rock or chisel shaped for soft friable rocks.

The procedure adopted is to channel a groove at right angles to the footwall and hanging wall sides of the exposure of the ore. Sample width varies according to local practice. A single sample may be taken from a narrow vein or a number of samples may be taken where the ore is thick (wide). The material from the chipping operation is collected by an assistant or on a drop sheet and bagged.

**Pick or chip sampling**
In practice this is more commonly used because of the slow labour intensive nature of channel/groove sampling. In chip sampling, a miner’s pick or geologist’s pick is used in one hand and a gathering sheet rolled into a cone or funnel is held in the other to collect the chipped material. Care is required to avoid bias due to collecting friable material.

**DEVELOPMENT SAMPLING**

Underground development of a block of ore consists of blocking out by levels and rises. Face samples are taken at various intervals in these development headings. These sample results can then provide a basis for calculating developed reserves (bore blocks) or confirming drill hole estimation of ore grade reserves.

![Figure 21.1 Development face sampling position end view](image)

The width of drive and ore intersection as well as the subsequent mining method will determine the type and manner of sampling. While a sample may be taken only across the ore body or vein, in mining, dilution occurs from the inclusion of country rock or waste
from the hanging wall and footwall. Hence the mined grade will be lower than the sample assay.

Samples are referenced in relation to survey stations, with an appropriate record of the sample statistics such as width and location in relation to the face.

![Figure 12.2 Stope preparation cross cut– sampling exposure (not true width)](image)

![Figure 12.3 Cut and fill stope ore exposure – dilution effect of inclusion of waste with ore to maintain width](image)

Development sampling therefore involves the evaluation of ore blocks and subsequent mining prospects, and because of the substantial cost of underground development, considerable reliance is placed today on drill hole results for grades and reserve calculations.

Underground sampling is important in terms of re-development of mines which have ceased production and where records are either not in existence or suspect.

**PRODUCTION SAMPLING**

Sampling in production serves the purpose of checking extraction from stopes, and directing production control on produced grade by selecting blending ratios and avoiding pulling waste/low grade from stopes and mixing with ore. Mine production departments
are budgeted/scheduled to achieve certain tonnages and grade in relation to the developed reserves.

Production sampling consists of chip sampling of stope faces and samples from chutes and trucks, the latter are grab type samples.

**GRAB SAMPLES**
A grab sample is obtained by taking a shovelful or handfuls of material from a chute or truck.

These may be used as a single sample or used with others to form a composite sample for a shift. Grab samples are subject to considerable bias, i.e. selection of fines, however over a period they are useful for supervision and control purposes.

**OTHER SAMPLE TYPES**

**ROCK DRILL SAMPLES**
Holes drilled into working faces or into sidewall ore yield chip and sludge material which may be collected and assayed. While driving on ore it is often standard procedure to drill the wall rocks in case there is parallel mineralization. This may be done at varying intervals, however where mineralization is disseminated, i.e. spread finely in the host or country rock, wall rocks may carry values, and wall drilling will be at frequent and close intervals.

To obtain good hole flushing without excessive fines, holes are often angled upwards. Various receptacles are used to collect sludge samples. In sulfide mineralization it is possible to judge the degree of mineralization by the colour. Dark colours show sulfides, light colours show waste rock.

Extension drill steel enables probe holes, deeper than conventional steel, used in headings, of 1.8 – 2.4 metres, to be made. Much depends on available power and the capacity of the drill, but the deeper holes may be around 3 – 6 metres. Samples may represent each metre of hole drilled.

**DIAMOND DRILL SAMPLING**
Diamond drilling usually produces a core representative of the ground which the hole has traversed. These cores provide visual evidence of the type of rock, structural features and any mineralization through which the hole passes, Therefore, to be reliable a high core recovery is essential: where core loss is greater than 20% it is generally necessary to collect sludge samples.

Diamond drill cores are split either by a core splitting device or cut longitudinally by a rock (diamond) saw. One half is retained for permanent reference in the core tray (stored in the core farm bulk drill core store and display area); the other half is sent for assay.

**BULK SAMPLING**
Bulk sampling is where a large mass of material is taken and treated often by a mini-treatment plant or pilot plant. The purpose of these tests is often to determine the metallurgical and in-plant characteristics of the material in question. Bulk samples may come from under-ground headings such as crosscuts through an ore body, test pits or costeans for near surface mineralization.
FRACTIONAL SAMPLING
This is akin to bulk sampling but is generally integrated to monitoring run of mine ore such as on conveyor belt or plant feed. These types of samples are usually taken automatically either on a time lapse-interval basis or on a through-put tonnage basis.

Most modern treatment plants incorporate radio-active isotope devices capable of monitoring relative levels of mineral concentration or grade. These may be linked to automatic - computerized control. Sampling is conducted to check control performance as well as to serve the important function of calibration.

COMPOSITE SAMPLING
Where a mine level has six producing stopes and 100 sample/assays have been made of the face values in each stope, the mean assay value can be calculated as well as the computed tonnage from a face survey. A check on the Computation of grade can be obtained by collecting a representative portion of each of the samples presented for assay. In this case these fractions would need to be of the same mass. The whole bulk collected would then be a composite sample which can then be further prepared for assay. The assay of the composite should closely tally with the mean calculated from the individual sample/assays.

Composite samples may also be useful in pilot plant studies because the sample material is a collection of small portions of a larger bulk rather than a single point bulk sample.

ERRORS IN SAMPLING
Errors in sampling may arise from a variety of causes such as:

• Salting: intentional interference with samples for the purpose of fraud;
• Variation of texture and hardness in the ore allowing more of the softer (friable) material to be chipped and collected in the sample;
• Lack of care, and negligence on the part of the person sampling;
• Poor procedure for gathering sample - lack of training and understanding of application of proper procedures;
• Loss of portions due to flying chips or dropping;
• Contamination from sample bags not properly washed where cloth bags are used. Likewise contamination can occur in the assay reduction process;
• Not cleaning down the face prior to sampling (cleaning down is washing and roughly cleaning off loose particles);
• Mixing up or incorrectly labelling sample identifications;
• Sample - assay preparation and practice procedures, which result in unreliable results.

Normal practice in mine sampling is to use duplicate samples for checks as well as re-sampling of faces to check sampler performance and sampling technique. Dummy samples of known grade may be presented for assay; these will often highlight possible occurrence of contamination. High grade samples, especially gold, can cause contamination to subsequent samples where fine grinding is involved by particles adhering to the grinding device. Very high grade faces which show free gold in the exposure may often not be sampled because the likely sample result will be very high and subject to bias from particles of free gold collected in the sample.
CRUSHING AND GRINDING APPLIANCES

In the preparation of samples in the assay sample room the material collected underground is crushed, reduced in bulk and mixed. This series of operations is repeated until a finely ground quantity of the sample is handled in the analysis section or laboratory. The quantity required will depend upon the method of determination. Fine assay procedures require considerably more material than, say, wet chemical analysis or atomic absorption spectroscopic determination.

Preliminary sample reduction is done by a small jaw type crusher which is cleaned by compressed air after each sample. Dust extraction equipment is used in preparation rooms. The crushed material is mixed and reduced in bulk either by riffling such as by a splitting device or by a hand method known as coning and quartering. (See Fig. 21.4)

After the bulk has been reduced to a convenient size the material is ground in a pulverizer, a mechanical device which crushes the material to a fine size. A common fine grinder is one consisting of one fixed plate and another which rotates or spins in close contact with the other. These plates have radial grooves on the faces of the discs. The material produced after grinding is of a sieve size of around 100 mesh (≈0.150 mm).

The number of passes and the set or closeness of the discs controls the resultant particle size. The material produced from fine grinding is subsequently sent for final analysis.

ASSAY PLANS

When the assay values of the samples are made available from the assay office they are plotted on to mine plans and sections in relation to their position of origin. These may be kept up to date by the sampler, or by survey or geological staff using current mine plans. Grab sample results are recorded only on an extraction – production report.

Where face positions are sampled in stopes, it is possible to derive an extraction record of the amount of ore removed (by survey measurement) and calculate the grade from face values/widths.

COMPUTATION OF QUANTITY AND QUALITY (grade and volume mass)

The quantity of ore within an ore body may be determined by bore hole intersection calculations. Each ore body intersection in a drill hole is plotted both on a hole section and in plan. The intersection width and grade is then attributed to an area of influence which may be a square, rectangle or polygon depending upon the frequency of holes and bore hole drilling pattern.

It is necessary to know the rock or mineral density (kg/m$^3$) relative to the values intersected. Where gold occurs in quartz the gold present has little bearing on the quartz density, however for sulfides the effect on rock density is significant.

The procedure used to calculate a bore hole intersection is illustrated.

Example:
Determine the average (mean) value of a bore hole intersection of nickel sulfide mineralization.

<table>
<thead>
<tr>
<th>Hole No.</th>
<th>Intersection depth (m)</th>
<th>Interval (m)</th>
<th>Grade % NiFeS</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>X1034</td>
<td>475.0 – 475.5</td>
<td>0.5</td>
<td>4.1</td>
<td>Massive sulfide</td>
</tr>
<tr>
<td></td>
<td>475.5 – 476.1</td>
<td>0.6</td>
<td>1.3</td>
<td>Disseminated</td>
</tr>
</tbody>
</table>
The procedure adopted is to weight the intersection width \( W \) with the assay \( G \) then calculate the weighted average i.e.

<table>
<thead>
<tr>
<th>( W ) (m)</th>
<th>( G % )</th>
<th>( W \times G )</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5</td>
<td>4.1</td>
<td>2.05</td>
</tr>
<tr>
<td>0.6</td>
<td>1.3</td>
<td>0.78</td>
</tr>
<tr>
<td>0.9</td>
<td>3.2</td>
<td>2.88</td>
</tr>
</tbody>
</table>

\[ \Sigma 5.71 \]

Hence total width is \( \Sigma \) (sum of) intersection widths 2.0 metres, the weighted grade is:

21.8
The determination of a block value of quantity and grade using corner bore hole values is obtained in the following manner:

The intersection values and widths are weighted together to determine the average mineralization width and weight grade. The width and grade density is then used with the block dimensions to calculate volume/tonnage.

Example:

Grid spacing is on a square pattern with an interval of 50 metres. The hole intersection width and values are as follows:

<table>
<thead>
<tr>
<th>Hole No.</th>
<th>Width (W)</th>
<th>Grade g/tonne</th>
<th>W x G</th>
</tr>
</thead>
<tbody>
<tr>
<td>X-10</td>
<td>1.8</td>
<td>7.2</td>
<td>12.96</td>
</tr>
<tr>
<td>X-11</td>
<td>2.2</td>
<td>8.3</td>
<td>18.26</td>
</tr>
<tr>
<td>Y-10</td>
<td>1.9</td>
<td>6.9</td>
<td>13.11</td>
</tr>
<tr>
<td>Y-11</td>
<td>2.3</td>
<td>7.6</td>
<td>17.48</td>
</tr>
<tr>
<td>Σ</td>
<td>8.2</td>
<td>Σ 61.81</td>
<td></td>
</tr>
</tbody>
</table>

average width = \( \frac{\Sigma W}{4} = \frac{8.2}{4} = 2.05 \) metres

average (weighted) grade = \( \frac{\Sigma W \times G}{\Sigma W} = \frac{61.81}{8.2} = 7.56 \) gm/tonne

Taking the hole intersections as true widths, then the volume \( V \) of the block is:

\[ V = L \times B \times (W) \]
\[ = 50 \times 50 \times 2.05 = 5125 \text{ m}^3 \]

The density of ore is 2700 kg/m\(^3\): host material is a quartz reef.

Quantity of ore: \( V \times \) density/1000 (tonnes)
\[ = 5125 \times 2.7 = 13837.5 \text{ tonnes} \]

The contained mineral is then the total tonnage multiplied by the average grade,

i.e. 13837.5 tonnes \( \times \) 7.56 gm/tonne
\[ = 104611.5 \text{ g of precious metal} \]
\[ = 104.61 \text{ kg} \]

The grade of ore in an ore body is estimated by averaging the assay results of the sample.

In the example below four samples have been taken along a leading stope and have the values of:

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Width (m)</th>
<th>Assay Grade %</th>
<th>WG</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.98</td>
<td>6.2</td>
<td>6.089</td>
</tr>
<tr>
<td>2</td>
<td>1.95</td>
<td>7.3</td>
<td>14.24</td>
</tr>
<tr>
<td>3</td>
<td>1.62</td>
<td>8.5</td>
<td>13.77</td>
</tr>
</tbody>
</table>
Plan of leading stope values  
Average width: 1.30 m  
Average grade: 7.36%

Figure 21.5 Stop values (insitu ore)

The effect of maintaining a minimum mining width makes it necessary to mine some of the foot wall or hanging wall which may be valueless: this results in dilution of the ore grade. An adjusted calculation is as follows: (See Fig. 21.6)

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Stoping width (m)</th>
<th>Width of ore (m)</th>
<th>Assay Grade %</th>
<th>W x G</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1.53</td>
<td>0.98</td>
<td>6.2</td>
<td>6.08</td>
</tr>
<tr>
<td>2</td>
<td>1.95</td>
<td>1.95</td>
<td>7.3</td>
<td>14.24</td>
</tr>
<tr>
<td>3</td>
<td>1.62</td>
<td>1.62</td>
<td>8.5</td>
<td>13.77</td>
</tr>
<tr>
<td>4</td>
<td>1.53</td>
<td>0.64</td>
<td>6.4</td>
<td>4.10</td>
</tr>
<tr>
<td></td>
<td>6.63</td>
<td>Σ 5.19</td>
<td></td>
<td>Σ 38.19</td>
</tr>
</tbody>
</table>

Average stoping grade: \( \frac{38.19}{6.63} = 5.76\% \)

Note that in this example the average ore grade is 7.36% but because the probable minimum stoping width is 1.53 metres then the stoping grade is 5.76%. The assumption here, is that the distance between the sample position is equidistant. Where this was not the case then the distance between the sample positions would need to be used in weighting the grade/width calculations.
In the computation of grades excessively high values especially for precious metal ores may need special treatment to avoid bias.

These abnormally high values may require cutting to a pre-determined value or they may be disregarded. However it may be practicable to re-sample the face if it is still accessible.

The computed grade and tonnes are compiled to form the basis of ore reserves. Depending upon the reliability of the data so the classification into proved, probable and inferred ore reserves is made.

Accurate ore reserves are essential for budgeting in mining extraction. Periodic checks on extraction need to be made for stope reserves. These are required for control purposes and the reconciliation or balancing of reserves. The information on ground mined comes from mine surveys and forms the basis of calculating the mine recovery. Valuable ore either left in a stope or lost in mine fill needs to be known and allowances made for it.

**COAL SAMPLING**

Samples are collected from fresh exposures by vertical channelling to obtain 8 – 9 kg of seam. This sample is then mixed and quartered then sealed in an air tight container to retain bed moisture and sent for analysis.

Coal quality is determined by a number of laboratory tests. Proximate analysis includes percentage of water in composition, volatile matter, ash and fixed carbon. Ultimate analysis determines hydrogen, carbon, nitrogen, oxygen, sulfur and ash percentages, along with heating value, swelling characteristics and other properties. Final analysis will depend upon the type of coal and its intended use.

Fundamentally, high grade coals are metallurgical coal in that they can be used to produce coke in blast furnace operations. Low grades are steaming coals suitable for burning thermal coal-fired power stations.
DESCRIPTION
Topics include: Mining Law; Blasting and Blasting Practices; Mining Methods; Mine Hoisting; Mine Services; Mine Ventilation; Mining Regulations. Study Guide 32-243 Mining 1 is used with this Text.

CATEGORY
Engineering, Mechanical, Metals and Electrical