SURFACE STRIP COAL MINING HANDBOOK

Compiled By RJ Thompson

2005
This handbook on surface strip coal mining serves as a general introduction to the subject of mining, specifically surface strip coal mining. It is intended as handbook for engineers who are new to the field of mining engineering, or for students who are entering the field with little prior knowledge. Mining and mining engineering are similar but not synonymous terms. Mining consists of the processes, the occupation, and the industry concerned with the extraction of minerals. Mining engineering, on the other hand, is the engineering and science of mining and the operation of mines. The trained professional who relates the two is the mining engineer; being responsible for designing, developing and exploiting and operating mines. Using scientific principles, technological knowledge, and managerial skills, the mining engineer brings a mineral property through the five stages in the life of a mine; prospecting, exploration, development, operation, decommissioning and closure.

Mining engineering fills a special niche within the special engineering profession. In an age of increasing specialisation, mining still requires both breadth and depth. The opportunities of dealing with a naturally variable and initially poorly defined resource presents exciting challenges experienced by few others. The future engineer requires a sound technical base while maintaining the optimism and enthusiasm of the prospector. One of the most famous Mining Engineers, Herbert Hoover once stated:

“To the engineer falls the work of creating from the dry bones of scientific fact the living body of industry. It is those whose intellect and direction bring to the world the comforts and necessities of daily need. Unlike the doctor, theirs is not the constant struggle to save the weak. Unlike the soldier, destruction is not their prime function. Unlike the lawyer, quarrels are not their daily bread. Engineering is the profession of creation and of construction, of stimulation of human effort and accomplishment”.

It is an almost impossible task to be an ‘expert’ in every aspect of surface strip coal mining engineering covered in this book. Operating and management experiences in strip coal mining are not reflected to any great degree in this handbook, rather, the basic generic principles, approaches and criteria of the mining method itself, together with the selection, application and operation of capital equipment for the various strip mining sub-system activities are highlighted.
The methodology adopted in compiling the handbook is one of review and evaluation of key references, texts and recognised state-of-the-art papers in each subject area and to temper these with information supplied and reviewed by practicing mining engineers involved with surface strip coal mining, either from a production, project or planning perspective. A concerted effort has been made to identify the main sources from which these materials have been extracted. Quotes and references “in-text” for each source have been omitted to maintain the readability of the book, thus references to these and other sources have been gathered at the end of the book in a comprehensive bibliography.

Reference to typical costs, capital or operating, have been avoided, due in part to their site-specific nature and also to their likely fluctuation over time which would quickly render the book obsolete. Similarly with legislation governing mining operations, unless central and generic to aspects of mining operations, reference to specific legislation in detail has been avoided and only the regulatory framework mentioned where appropriate.

The knowledge areas covered by the handbook follow a typical mine development and operating system; from exploration, initial method selection, through to operations and the production process of drilling, blasting, excavation and material transport. Finally, the various facets are brought together in an analysis of the planning, scheduling and costing of a strip coal mine. The modules include:

- Introduction to surface strip coal mining
- Geology and exploration
- Surface coal mining methods
- Rotary drilling machines
- Surface strip coal mine blast design
- Loading and hauling systems
- The selection and application of draglines
- Equipment cost estimation techniques
- Mining environment, rehabilitation and closure
- Surface coal mine planning and scheduling
The basic knowledge and principles that you gain from this handbook can be used as the point of departure for more advanced studies, together with application case-studies and current practice reviews. The objective of the book is to enable the reader to understand the method of surface strip coal mining and all the inter-related sub-system activities in each of the five phases of a mine’s life mentioned earlier.

Each module of the handbook addresses a specific set of study themes and is prefaced with specific learning outcomes, as basic knowledge and understanding, application, calculation and evaluation, through to synthesis and design. This is intended to assist students in their studies, in order to acquire the required skills and achieve the learning outcomes effectively.

The statements used to define the learning outcomes are classified in terms of a series of lower- to higher- order cognitive domains, in accordance with Bloom's Taxonomy of Educational Objectives. Each Module is built in the order depicted in the Figure below, from a basic knowledge through application and analysis to eventually enable the student to synthesis, design and evaluate, as required by a practicing mining engineer.
The opinions expressed in this Handbook do not necessarily represent those of the South African Colliery Managers Association. Whilst the author, the SACMA Tertiary Education Handbook Committee Review Panel and publisher have made every effort to ensure the accuracy of the information presented herein, no warranty is given in respect of its accuracy, correctness or suitability. It is the responsibility of the reader to evaluate the appropriateness of particular information and methods or guidelines presented herein, in the context of actual conditions and situations and with due consideration to the necessity to temper this information with site-specific modifications. Author, Tertiary Education Handbook Committee Review Panel and the publisher cannot be held responsible for any typographical or other errors found in this Handbook and accept no liability for any consequences arising out of the use of the information presented herein.
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INTRODUCTION TO SURFACE STRIP COAL MINING

Learning outcomes

Knowledge and understanding of
- Fundamental concepts of resource and reserve evaluation
- Fundamental definitions of resource and reserve
- The role of surface coal mining internationally and locally
- The advantages surface mining offers over underground methods and the typical costs values of capital expenditure (CAPEX) on a new mine and typical operating expenditures (OPEX)
- Fundamental stripping ratio concepts, the break even stripping ratio (BESR) and multiple seam mining incremental stripping ratios
- The components of the basic profit function

Apply, calculate or predict
- Basic production cost, profitability and limiting stripping ratio and BESR
- Stripping ratios and unit profits from selected mining sequences for single or multiple seams
1.1 Fundamental Concepts and Definitions

Initially, any material that is exploited by mining can be sub-divided into ore (or revenue producing materials) and waste, commonly mined to expose the ore. Ore is typically defined as a natural aggregation of one or more solid minerals that can be extracted, processed and sold at a profit.

Although definitions are important to know, it is even more important to know what they mean in the practical sense. The key concept is ‘extraction leading to a profit’. For engineers, profits can be expressed in simple equation form as;

\[ \text{Profits} = \text{Revenues} - \text{Costs} \]

The revenue portion of the equation can be written as;

\[ \text{Revenue} = \text{Ore sold (units)} \times \text{Price/unit} \]

The costs can be similarly expressed as;

\[ \text{Costs} = \text{Ore sold (units)} \times \text{Cost/unit} \]

Combining the equations above yields;

\[ \text{Profits} = \text{Material sold (units)} \times (\text{Price/unit} - \text{Cost/unit}) \]

The price per unit received is more and more being set by worldwide supply and demand. Thus, the price component in the equation is largely determined by others. Where the mining engineer can and does enter is in controlling and managing the unit costs – associated with ore production. To remain profitable over the long term, the mining engineer must continually examine and assess smarter and better site specific ways for reducing costs at the operation. This is done through a better understanding of the deposit itself and the methods and equipment employed or employable in the extraction process. Cost containment/reduction through efficient, safe and environmentally responsive mining practices is serious business today and will be even more important in the future with increasing mining depths and ever more stringent regulations.

The ore itself, be it metalliferous or non-metalliferous as in the case of coal, is located in what is termed a deposit or orebody. It is essential that the various terms used to describe the nature, size and quality of
the deposit be very carefully selected and then used within the limits of well recognized and accepted definitions. A deposit can be further defined in terms of its potential to be economically exploited, on the basis of a resource or a reserve.

Over the years a number of attempts have been made to provide a set of universally accepted definitions for the most important terms. These definitions have evolved somewhat as the technology used to investigate and evaluate orebodies has changed. The South African Code for Reporting Mineral Resources and Mineral Reserves (SAMREC) sets out minimum standards, recommendations and guidelines for Public Reporting of Exploration Results, Mineral Resources and Mineral Reserves in South Africa. The main principles governing the operation and application of the SAMREC Code are transparency, materiality and competence. Public reports dealing with Mineral Resources and/or Mineral Reserves must only use the terms set out in Figure 1.1 below.

Figure 1.1 Mineral Resource and Reserve (following SAMREC, 2000).

Figure 1.1 sets out the framework for classifying tonnage and grade estimates so as to reflect different levels of geoscientific confidence and different degrees of technical and economic evaluation. Mineral resources can be estimated on the basis of geoscientific information with input from relevant disciplines. Mineral reserves, which are a
modified sub-set of the Indicated and Measured Mineral Resources (shown within the dashed outline in Figure 1.1), require consideration of factors affecting extraction, including mining, processing, economic, marketing, legal, environmental, social and governmental factors ('modifying factors'), and should in most instances be estimated with input from a range of disciplines.

A Mineral resource is defined as follows;

“a concentration [or occurrence] of material of economic interest in or on the earth’s crust in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. The location, quantity, grade, continuity and other geological characteristics of a mineral resource are known, estimated from specific geological evidence and knowledge, or interpreted from a well-constrained and portrayed geological model. Mineral resources are subdivided, in order of increasing confidence in respect of geoscientific evidence, into Inferred, Indicated and Measured categories.”

For an inferred resource, estimates are based on geological evidence and assumed continuity in which there is less confidence than for measured and/or indicated resources. Inferred resources may or may not be supported by samples or measurements but the inference must be supported by reasonable geo-scientific (geological, geochemical, geophysical, or other) data.

For an indicated resource, the quantity and/or grade are computed from information similar to that used for measured resources, but the sites for inspection, sampling, and measurements are farther apart or are otherwise less adequately spaced. The degree of assurance, although lower than that for measured resources, is high enough to assume geological continuity between points of observation.

For a measured resource, quantity is computed from dimensions revealed in outcrops, trenches, workings or drill holes; grade and/or quality are computed from the result of detailed sampling. The sites for inspection, sampling and measurement are spaced so closely and the geological character is so well defined that size, shape, depth and mineral content of the resource are well established.

The mineral reserve is thus defined as;

“the economically mineable material derived from a Measured and/or Indicated Mineral Resource. It is inclusive of diluting materials and allows for losses that may occur when the material is mined”.
The term ‘economic’ implies that extraction of the Mineral reserve has been demonstrated to be viable and justifiable under reasonable financial and mining recovery assumptions. A mineral reserve is also reported as inclusive of marginally economic material and diluting material delivered for processing, thus it is clear that a thorough understanding of the expected mining recovery and dilution is necessary. This is both an inherent function of the specific mineral reserve and more importantly, the mining method, equipment and processing route adopted for it’s exploitation.

Although mine planning and scheduling is discussed in more detail later in the book, the exploration, development, and production stages of a mineral deposit can be defined initially as:

Exploration: The search for a mineral deposit (prospecting) and the subsequent investigation of any deposit found until an orebody, if such exists, has been established.

Development: Work done on a mineral deposit, after exploration has disclosed ore in sufficient quantity and quality to justify extraction, in order to make the ore available for mining.

Production: The mining of ores, and as required, the subsequent processing into products ready for marketing.

The first step in the exploitation of a mineral deposit is exploration, and this is most often based on a thorough understanding of the mode of occurrence and geology of the deposit in question.

1.2 Surface Coal Mining

The greater proportion of the world’s mineral needs are at present provided by surface mining methods, more or less 60% of the world’s mineral needs without including coal and lignite. When coal and lignite production is also included, the total amount rises to above 70%. Approximately 200 surface mines worldwide have a production capacity of over 3mtpa ore (the commodity mined to produce the income or revenue) per annum, whilst some 140 mines produce in excess of 10Mtpa per annum. Depending on the amount of waste (the rock or overburden removed to expose and exploit the ore) that has to be stripped, the total tonnage can often exceed 70Mt (waste and ore) per annum.
In 2002, opencast mines provided 50 percent of the run-of-mine production of coal in South Africa. The seven largest collieries with an output of more than 10 Mt/a), produced 122 Mt of saleable coal. Six large mines (output of more than 5 Mt/a), produced 42 Mt, nine medium-sized mines (output of more than 2 Mt/a) produced 31 Mt, 11 smaller mines (output more than 2 Mt/a) produced 15 Mt and the 21 smallest mines (output of less than 1 Mt) produced 9Mt.

Whether coal will be mined by surface or underground methods will depend on the relative mining costs. In all instances, the total capital and operating costs must be considered. A correct decision is extremely important in as much as contract commitments are made for product deliveries long before the mine is placed in production. Equipment for surface and underground mining is not interchangeable. Further, investment in equipment for either mining method and pre-production stripping expense for surface mining must be made long before any returns are received and before the method has been proven successful. Thorough engineering studies are therefore essential to help establish the most favourable mining method. In general, surface mining is found to be more economical when the coal reserve is large and the depth of overburden is not excessive.

To plan, develop and bring a new surface mine in South Africa in production takes about 2-6 years at a total investment or capital cost of approximately R500M to R1 400M depending on the volume of production and type of mining method applied. In the case of surface coal mining in the Southern African context, the surface strip mining method is widely used, as coal seams and overlying rock lend themselves to this type of operation.

The predisposition to surface strip mining methods has increased over the last three decades and will continue to increase in the foreseeable future, while underground mining in general will still stay more labour intensive and the number of mines will remain essentially constant. Other factors that make surface strip mining methods more favourable over underground mining methods are:

- Higher productivity – run of mine (ROM – coal produced from the pit before beneficiation) tons of coal or volume of overburden per man year. Average figures are typically 10kt/man-year coal produced (minimum 4 000 and maximum 13 k) and 19kBCM/man-year overburden
- Lower capital cost per ton of ore mined – typically R70/ROM ton per annum
- Lower operating cost per ton of ore mined
The possible exposure of lower grade reserves (because of lower operating cost per ton mined – hence lower grades become profitable)

Improved geological certainty of reserves

Less limitations on size and weight of machines and associated higher efficiencies. Typically this could be equated to approximately 600kBCM/striping equipment, or typically about 12MBCM per annum for a large dragline

Increased recovery of ore – support pillars are unnecessary

Improved safety - loose material can be seen and removed or avoided, and crews can be readily observed at work by supervisors

Larger reserve areas available for mining (at higher annual production rates).

Further, the cost spread between the two methods is growing wider as larger-scale methods are applied to surface mines. However, various factors have also contributed to limit the development of large surface strip coal mines and these are becomingly increasingly important. They can often add significant cost to an operation – although not all apply in Southern Africa, they are nevertheless widely regarded as necessary considerations.

The environmentalists’ and conservationists’ concerns that surface mining especially is a threat to the environment and our existence. This pressure has in some cases lead to extreme limitations on environmental planning and mine permitting – more especially in Europe and the United States.

Few large new deposits are available near established infrastructure. With the extraction of more remote deposits, the costs related to the development of infrastructure systems are often greater than the development cost of the mine itself.

The political instability of many of countries involved (especially third world countries) and the difficulty of obtaining financial support or investments for projects that often exceed many millions of Rands, together with the rapidly changing political situation in these countries means that tax and capital investment regulations can change which would be unfavourable for overseas investors.
In the past, when economic conditions declined, the problem of unit cost increases was reduced through increases in the size of equipment, so as to increase productivity through improved effectiveness (especially in the case of waste handling systems because of waste handling cost sensitivity). For the last decade, the size of equipment has stayed on a plateau and in some instances it has even reduced (for example, the largest machines of the dragline models). Thus it seems that future large scale savings through economies of scale will be unlikely.

Whilst these factors as a whole are important considerations, the most important consideration is that of cost of mining compared to income generated from the sale of the ore, as stated in Module 1.1. This concept can be enumerated from the basic production cost and it forms the basis of determining economic viability, or profitability of mining, and some key terms encountered in surface mining.

1.3 Production Cost and Stripping Ratio Analysis

Surface mining of coal is conducted in a relatively simple sequence of operations (or sub-systems) which includes: 1) preparing the surface, 2) drilling, 3) blasting, 4) overburden removal, 5) loading the deposit, 6) haulage of the mined deposit, and 7) rehabilitation. Mining techniques for a particular region are largely dictated by geologic and topographic conditions. Even where the techniques are generally comparable, economics of alternative equipment choices and utilisation are not easy to generalize.

The surface mining techniques can be broadly classified into:

- strip mining
- terrace mining
- open-pit mining
- quarry mining.

As with any mining project associated with any of the above methods, the planning process for strip mining is based on data collected and the prospect of making a profit. The variables that are involved in the analysis, and the method itself, will be discussed in more detail in later Modules. When considering the fundamental elements of
surface strip mining, we need to consider firstly the basic production cost and secondly, the profit that can be made from mining.

In simple terms, the basic production cost at a surface mine comprises of:

- The unit costs associated with excavating one unit of ore (A) and it’s transport to the processing plant. Ore is the source of income.
- A similar unit cost for the waste material (B). The minimizing of this cost element in most surface strip coal mines becomes very important because it can reduce the profitability of the mine (because of the effect of stripping ratio). Waste mining does not generate any income for the mine – it is solely an expenditure item.

The stripping ratio (S) is defined as:

\[ S = \frac{\text{Units waste}}{\text{Units ore}} \]

And is calculated in similar units for ore and waste (e.g. t/t or m³:m³ or BCM:BCM (Bank Cubic Meter – the volume of ore or waste in situ before it is mined)). Take note that the stripping ratio can change because of different material densities. In a certain coal strip mine, a stripping ratio of 6:1 (t waste mined per t ore recovered) will be equivalent to 3:1 (m³ waste per m³ ore) if the density of waste is 2t/m³ and that of coal (ore) is 1t/m³. It is also common practice in strip coal mines to refer to a stripping ratio at BCM:ROM ton, in which case the stripping ratio would be 3:1. In either case, it is important to refer to the units of the stripping ratio in addition to its value.

The removal of overburden, or stripping, is generally regarded as the most significant component of surface coal mining costs. Variations in stripping ratio affect the scale of the equipment and the efficiency of its operation as far as overburden handling is concerned, while procedures for handling, preparing and marketing coal, and costs associated with these three steps are, in comparison, fixed from mine to mine. The stripping ratio does not necessarily remain constant over the lifetime of the mine but can change according to production schedules, geology of the overburden and/or coal seam, etc. The current or instantaneous stripping ratio is usually indicated by S’. In the case of surface strip coal mining, the variability of the stripping ratio can be due to increasing overburden or coal seam thickness.

Distinction is often made between several different ratio calculations, such as actual or overall stripping ratio, ultimate or break - even stripping ratio, daily or working stripping ratio, and the stripping ratio
developed on the basis of underground mining costs vs. surface mining costs excluding stripping costs. The considerations of various ratios are also dependent on the scale of the operation, the capabilities of equipment and the general economic structure of the company.

Usually the first analysis is the calculation of a stripping ratio at which recovery by surface mining is cost equivalent to recovery by underground methods. Once a surface vs. underground ratio is established, this ratio allows us to determine;

- The amount of reserves that can be mined from surface
- Whatever ore can be recovered at a lower stripping ratio is being mined more economically than is possible by underground

If \( U \) (R/ore unit) represents the cost of underground mining of a unit of coal, \( A \) the unit costs associated with excavating one unit of ore by surface mining and a similar unit cost for the waste material \( B \), it is clear that the economic limit for the stripping ratio \( S_{\text{lim}} \) – when it will become more profitable to mine using underground coal mining methods) is given by;

\[
S_{\text{lim}} = \frac{U - A}{B}
\]

Total design of the mine requires that an ultimate break - even stripping ratio (BESR) be calculated. This ratio is that which establishes the ultimate limits of mining, and is not the overall stripping ratio. The break - even ratio is that ratio at which the total cost of marketing one unit of coal is equal to the price of that unit of coal. The overall ratio must be less than the break - even ratio or no profit would be realised. It forms the basis of the profit function analysis used in the next section.

\[
S_{\text{BESR}} = \frac{rF - A}{B}
\]

Where;

\[
\begin{align*}
F & = \text{coal yield (or recovery – percentage of coal mined that is eventually sold)} \% \\
r & = \text{sales price per unit product coal} \\
A, B & = \text{as previously defined}
\end{align*}
\]
In reality, the BESR (or any stripping ratio) is most commonly a linear polynomial function in several variables, including, but not limited to those as summarised below;

- Processing treatment cost per unit - C
- Rehabilitation cost per unit - R
- Transportation, sales and overhead per sales unit - T

Hence the BESR is further defined as;

\[ S_{BESR} = \frac{(rF - T) - A - C - R}{B} \]

The determination of stripping ratio has been shown to be quite dependent upon and sensitive to a variety of costs. Such a dependence on costs means that the BESR is related to the scale of the operation. The scale and type of operation, economic nature of the company and rehabilitation requirements will affect the development of the BESR and the daily or working stripping ratio of the operation.

The working stripping ratio is generally much less than the BESR. It varies with the depth of cover at the time and provides an overall ratio less than or equal to the BESR. In general, lower stripping ratios imply lower production costs. An objective in planning is often to mine with the contour, i.e. starting where the depth of overburden is least or at the outcrop, if the seam is outcropping. The working stripping ratio then increases with the progress of mining. Such an approach allows higher unit profits (i.e. low production costs) in the first years of life, when the time value of money is highest, and defers high unit production costs to a later point in time. Additionally, if improvements in technology or a higher price for the product is attained at a future date, the BESR can be increased. However, where the topography is severe (steep hill), the BESR is reached in a very small width. Therefore, the entire width is mined to the BESR point and then reclaimed. This concept is further developed in Module 3.

### 1.3.1 Stripping Ratio for Multiple Seams

Thus far, the development of stripping ratio concepts has been based on a single coal orebody, or seam. The analysis of stripping ratio may be complicated by considering the case where several seams are mined from one cut. Multiple-seam planning requires that both
total stripping ratio and the incremental stripping ratios be analysed. A necessary condition is that the cost of stripping all overburden, interburden (waste between coal seams), and partings be less than or equal to the total net value of coal extracted from the seams. However, the total analysis is not, of its own, sufficient to justify multiple-seam mining. Even if it is feasible to remove two seams in terms of the overall stripping ratio, the lower seam will not be mined unless the incremental stripping ratio for the lower seam is also acceptable.

Consider, for example, a situation where the deposit consists of 10m of overburden, 1,0m of coal, 20m interburden, and 0,5m coal. The overall stripping ratio (in terms of meters) is 20:1.

If the BESR for the company were 25:1, then the total ratio would be acceptable, and seemingly both seams should be mined. However, the incremental stripping ratios are 10:1 for the upper seam and 40:1 for the lower seam. While the upper seam is quite favorable for mining, the incremental analysis of the lower seam fails to justify mining both seams.

A complication might exist when considering more than two seams: A situation where mining the upper two seams is not feasible while mining three or more seams is feasible. Consider a situation as above, but in which 10m of interburden lie over a third, lowest seam, 2,5m thick. The incremental analysis would become 10:1 and 40:1 as above for the upper and middle seams, and 12,2:1 for the lowest seam (without considering the middle seam) or 10:1 (with middle seam as a by-product).

It would then be advisable to mine to the lowest seams. Of course, since the lowest seam is to be mined, and since the stripping ratio considering the second seam as “by-product” is better than mining the lowest alone, the second seam will also be recovered. Unique cases may exist when the particular economics of coal handling and cleaning may indicate that the middle seam should be treated as interburden. However, with stringent requirements for the total utilisation of coal reserves, the burial of coal in waste overburden and conservation of natural resources, the practice may not be justified.

The above stripping ratio analyses are clearly a simplification, the definitive calculation is also dependent on other factors such as the coal quality, specific market and income structures, recovery, processing and other costs. When these factors are considered, then the profit function can be derived.
1.4 Basic Profit Function Analysis

Consider a flat-lying coal seam overlain by waste which increases in thickness. As mining proceeds, more waste has to be mined to expose the same amount of coal. Also, a point will be reached where the cost of waste mining equals the income from coal sales, i.e. the profit will be zero. This point is referred to as the BESR as previously.

For surface strip coal mining, consider the following cost and income variables:

- $S'$ = incremental or periodic stripping ratio
- $A$ = cost of stripping of 1 unit waste
- $B$ = cost of mining of 1 unit coal
- $C$ = coal treatment cost per unit coal
- $F$ = coal yield (or recovery – percentage of coal mined that is eventually sold) (%)
- $r$ = sales price per unit coal
- $V$ = rehabilitation cost per unit coal mined
- $T$ = transportation (port and rail) per sales unit
- $O$ = cost of sales and overheads per sales unit
- $R$ = royalties per sales unit
- $D$ = administration and management costs per unit
- $P$ = profit per coal unit

Note that coal ore units are typically tons, thus cost of mining (ore or waste) would be R/t, processing costs R/t, and selling price thus R/t. Hence profit in R/t. Also – since we include a yield, we refer to ROM tons (tons mined before beneficiation or washing).

The profit function is thus:

$$P = (r - O - R - T)F - S'B - A - C - V - D$$

Figure 1.2 illustrates a typical breakdown of these costs for a large surface mining operation.
Thus it can be seen how critical the control of waste mining costs are to both the profitability of a surface strip coal mine, and in respect of the ultimate depth of surface workings.

The mine planning process would have determined these measures of economy prior to mining – in fact, the decision whether or not to mine a certain coal deposit is made along these lines. However, the data required to estimate these various costs, especially for a mine in the planning stages – form part of the mine planning process and are dependant on a number of method - and equipment-related choices in the planning process. Before surface strip coal mine planning and costing is discussed, it is necessary to gain an understanding of these various method and equipment options.
# GEOLOGY AND EXPLORATION

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Evaluate or design
- Marketability of a given type of coal
- Ore resource to reserve delineation technique for a given coal resource
2.1 Geology of South African Coal Resources

Coal mining is one of South Africa’s basic industries and represents the country’s major source of energy. It has contributed to significant industrial development, from its initial discovery and exploitation at Breyten in the eighteenth century. Today, South Africa has in excess of 55.3 billion tons of recoverable coal reserves (representing 10% of the world’s known reserves), which are mined by both surface and underground methods. In general, within limited areas of several hundred and, in some localities, several thousand square kilometers, individual seams (which may number several dozen, but usually six or less) show remarkable consistency and, in contrast to the predominant conditions in the northern hemisphere, are relatively flat-lying and undisturbed. This greatly facilitates their mining and leads to a lower cost structure than exists in many other countries. Exactly which mining method is chosen depends largely on the financial and geological suitability of the reserve to either underground or surface exploitation. This concept will be covered in more detail later in the Module, suffice to say that whichever method is finally selected, it should be the one which delivers lowest cost per ton mined, both to provide a financial return and to maximise economic exploitation.

Most major coal deposits occur in the Middle Ecca group, with overburden thickness varying from 30-150m, in a series of seams, some of which are not necessarily economic or of suitable quality to mine. The rock or waste overlying these seams is typically sandstones and shales of variable thickness. It is the thickness of this waste, combined with the thickness of the coal seam(s) below that dictate to a large degree if surface strip mining can be used. Major reserves of coal and operating surface strip coal mines are located throughout South Africa, but centered mostly in the provinces of Mpumulanga, (northern) Free State, Gauteng and Limpopo.

2.1.1 Coal Formation

South African coals have been formed from a wide variety of plants varying from the simplest forms through ferns to large trees, but rather different from the plants of today. Fossils of a wide variety of plants are found in the coal seams and the associated shale beds. The dominant plants contributing to the formation of coal in South Africa were rather different from those of the northern hemisphere.
For the formation of coal seams, peat (where plants grow in swampy conditions, although decay also occurs, the biochemically altered plant material does not disappear but forms peat) had to subside and become covered with water; and then layers of clay, silt and sand were deposited above the peat. Over time, more and more sediments covered the peat beds, and under the influence of pressure, enhanced temperature, the second stage of alteration from the original plant material began. In the peat stage of transformation, usually referred to as the biochemical stage, the rate of reaction was rapid and a wide variety of end products could be obtained, dependent on the original vegetation present, the conditions in the swamp, and the number and variety of fungi, bacteria and other parasitic organisms present. Soon after burial, bacterial activity ceased, and the further transformation proceeded at a very much slower pace. The main net effects of the second, or metamorphic, stage are the reduction in the amount of water held by the coal, and a decrease in the oxygen content of the coal.

The coal measure strata consist of coal, shale, and sandstone, with lesser amounts of grit and occasional conglomerates. The classical succession of deposition consists of seat earth (nearly always absent in South Africa), coal seam, and shale roof which grades into sandstone, followed by similar successions for each coal seam. These deposits are formed due to the gradual subsidence of the strata, so that new beds form by deposition from inflowing water.

In South Africa, coal seams usually have a shale or sandstone floor, but grits and conglomerates (and even Dwyka tillite below the bottom seam) are also found. The same rocks, apart from Dwyka tillite, can also form the roof of the seam. Conglomerate formed by tillite which has been washed down from higher ground is restricted almost entirely to the roof of the bottom seam in the Orange Free State. For some seams either shale or sandstone forms the typical floor or roof, but in other seams the roof or floor may gradually change laterally from one rock type to the other. Some of the thicker seams in the upper portions show a gradual transition from inferior coal to shale. In general, sandstones form the dominant rock type of the coal measures.

The coalfields of South Africa were laid down in horizontal beds, and all the important coalfields have virtually remained so. There are minor rolls in the floor causing local dip variations, and some shallow basins occur, either because of being so deposited, or due to later consolidation of underlying beds through the pressure of the overburden. The slopes in these areas seldom exceed 6°-10°. In certain parts of the coalfields extensive faulting has occurred, nearly always as a result of the intrusion of molten igneous rocks (dolerite) into the coal measures. These intrusions cause displacement of the
strata from a few metres to over a hundred metres; the strata on each side of the fault remain essentially horizontal.

The coalfields are limited by areas where conditions were unfavourable for the formation of coal, by abutment against elevated masses of older rocks forming the borders of the original peat swamps, and by erosion of the coal measures where these occur in elevated positions.

2.1.2 **Stages in the Formation of Coal**

The following Table 2.1 is a brief description of important stages in the formation of coal. In reality the stages are continuous with no sharp distinction transformation between them.

<table>
<thead>
<tr>
<th>Formation</th>
<th>Type of coal</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>Young</td>
<td>Peat</td>
<td>Peat is an unconsolidated product of the decomposition of vegetation found in bogs or marshes. It is characterised by a very high moisture content, and has a low heating value. Peat is not coal, but merely its forerunner.</td>
</tr>
<tr>
<td>Lignite</td>
<td>This is the first stage in true coalification. It contains less moisture than peat, and has a higher heating value.</td>
<td></td>
</tr>
<tr>
<td>Sub-bituminous coal</td>
<td>This is a black coal, but does not contain the glistening bright bands found in bituminous coal. It is lower in moisture content and higher in heating value than lignite</td>
<td></td>
</tr>
<tr>
<td>Bituminous coal</td>
<td>This stage is subdivided into high, medium and low volatile groups, indicating increase in rank. The heating value increases with decreasing moisture content, and for the higher rank bituminous coals up to anthracite, remains fairly constant. Coking properties are present in the bright coals over most of the range of rank, being absent in the least mature coals, and in South Africa also in low volatile bituminous coals</td>
<td></td>
</tr>
<tr>
<td>Lean coal</td>
<td>This is a high rank coal with volatile matter content between those of low volatile bituminous coal and anthracite. This coal burns with very little smoke.</td>
<td></td>
</tr>
<tr>
<td>Mature</td>
<td>Anthracite</td>
<td>This is normally the final stage of coal metamorphosis. Anthracite burns without smoke, has volatile matter below about 10%.</td>
</tr>
</tbody>
</table>
2.1.3 Principal Coal Seams of South Africa

The coalfields occur mainly in the Mpumulanga, Gauteng, Limpopo and Kwazulu-Natal provinces, and are spread over an area some 600 km from north to south, and 500 km from east to west. As a rule, there is a general increase in coal rank (i.e. carbon content) from west to east across the coalfields, as well as a decreasing number of seams and a thinning of the stratigraphic thickness within which they occur. Figure 2.1 shows the location of the most significant reserves in South Africa, sub-divided into 19 principal coalfields, each representing a separate basin of deposition. A short description is given of some of the major coal seams in South Africa. It does not address specifically each coalfield shown in Figure 2.1 but rather the fields in which larger surface strip or terrace coal mines are operating.

In South Africa the following scheme of classification (Table 2.2) is used to describe the various classes of coal associated with coalfields and seams described next.

Table 2.2 General characteristics and descriptions of coal types

<table>
<thead>
<tr>
<th>Percentage bright coal (%)</th>
<th>Density (kg/m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal shale</td>
<td>&lt;33</td>
</tr>
<tr>
<td>Dull coal</td>
<td>33</td>
</tr>
<tr>
<td>Mixed (mainly dull)</td>
<td>33-50</td>
</tr>
<tr>
<td>Mixed (mainly bright)</td>
<td>50-67</td>
</tr>
<tr>
<td>Bright coal</td>
<td>&gt;67</td>
</tr>
</tbody>
</table>

WITBANK COALFIELD
This coalfield extends a considerable distance southwards towards the Free State border. The coal seams are numbered from No. 1 to No.5 upwards and identified by these numbers.

The No.1 Seam consists typically of dull coal, but in certain areas contains a fair amount of bright coal. At its best development the seam is about 2m thick and has less than 15% ash. The seam is
sometimes thin or absent, and can deteriorate in quality to well over 20% ash.

The No.2 Seam generally contains bright coal at the bottom, and has bright bands at other levels, but still consists mainly of dull coal. It is normally over 6m thick, and is the most widespread seam of the coalfield. It is undoubtedly the main economic seam, and has been
very extensively worked. The seam as mined varies in ash content from about 13% to 18%, except for some areas where it is mined for power production. Sometimes a lower split, called the 2A seam, is present.

The No. 3 Seam is not of economic importance. It is often much less than one metre thick, and consists of mainly bright coal. It has been found in mineable quantities at one opencast mine.

The No.4 Seam is normally about 3m thick or more, and consists mainly of dull coal, with a few bright bands. The ash content is generally over 15% and can be over 20%. In the Bethal and Standerton districts the No.4 seam is the most important economically. Splits are often present above the main seam and these are called the Upper 4 and 4A seams but are not always economically important.

The No.5 Seam seldom exceeds 2m in thickness, and can be less than one metre thick. It consists of mainly bright coal, sometimes with duller coal towards the top. It is restricted to the higher ground, and is mined mostly for blend-coking coal. In the outlying areas of the coalfield the rank of the coal is lower with consequent loss of coking properties.

**FREE STATE COALFIELD**

The seams occurring south of Vereeniging, where the full development occurs, will be described. There are three main seams, named No.1, No. 2 and No.3 from the bottom.

The No. 1 Seam is variable in thickness and quality. The coal is mainly dull in appearance. In the north the seam can attain great thicknesses, and up to 15m has been worked. In the north the quality is also at its best and ash contents as low as 15% have been obtained over restricted areas. Generally however ash contents tend to be about 20% or more. The seam tends to thin towards the south, where it may be less than 5m thick.

The No.2 Seam again tends to be variable in thickness but is often 6m or more thick with about 20% ash or more. The coal is essentially dull, with very little bright coal. A shale band of about one metre often separates the seam into upper and lower portions. This band varies in position but is often near the middle of the seam.

The No.3 Seam is normally 2 to 3m thick, and is uniformly dull. Ash contents tend to be over 20%.
WATERBERG COALFIELD
This coalfield is characterised by coal bearing zones in the shallower part of the field and discrete seams in the deeper portions. The zones, up to 30m thick, consist of narrow coal seams closely interspersed with waste bands. Up to twelve such seams may be identified. The raw ash content of the whole zone can be up to 50% and the ash content of the coal is high, as it contains closely intergrown mineral matter. On washing the zone, to get blend coking coal of 10% ash, the yield is low. In the seams below the zones, much higher yields, of 60-70%, of low ash coal are found, with generally less well developed coking properties.

2.1.4 Coal Quality
The coalification process involves the transformation of plant matter into coal. This is a process of metamorphism and the degree of change is referred to as the rank of the coal; low rank has undergone only a slight change, whilst high rank coal is greatly changed in nature and chemical composition. Heat is an important factor in the metamorphosis of coal. In parts of the coalfields extensive igneous intrusions are found which, if within reasonable distance from the coal seams, caused an increase in temperature of the coal and therefore an increase in rank. These effects are fairly localised where the dolerite has passed through the coal as a dyke, but where thick sills lie parallel with the coal seams the effect can be widespread and consistent. All the medium and low volatile bituminous coal, as well as the lean coal and anthracite, of the northern Kwazulu-Natal coalfields are due to the effects of igneous intrusions. The maturing effect of dolerite can be beneficial, as is often the case in Kwazulu-Natal, or harmful. In the latter case the matured coal becomes soft and powdery, decreasing its value to a very large extent, and there is often an associated weakening of the roof or floor of the seam, which makes mining difficult (such material is "burnt coal"; if it is hard, it is called "mineral coke").

In general, the rank of coal is measured by a number of analytically determined values, since no single parameter can cover the whole range from peat to anthracite. As rank increases, the air dried moisture tends to decrease, and the percentage carbon in the coal increases. Changes also occur in the petrographic characteristics of the coal.

When coal is ignited, it does not burn completely away. There is always some material in the coal which is inert to combustion. This material consists of water (moisture) and mineral matter which remains in an altered form as ash. Coal always contains water,
which is referred to as its moisture content and the higher the moisture content, the less material is available for combustion.

Two types of moisture content are recognised;

- Inherent moisture is contained in the coal micropores. It cannot be removed by draining but can be removed by heating above the boiling point of water. The value for a given coal is determined by the rank of the coal and the atmospheric humidity.

- Extraneous moisture occurs on the surface of coal, and in cracks and joints. Its sources are water which percolates into the mine workings. Extraneous moisture is easily removed.

Mineral matter is the inert solid material in coal, and like moisture it reduces the heating value of coal by dilution. On burning coal, the mineral matter remains behind in a slightly altered form as ash.

Three types are recognised;

- Inherent mineral matter is mineral matter intimately mixed with the coal. It consists of the minerals present in the original vegetation from which the coal was formed, and finely divided clays and similar materials. All South African coals contain varying quantities of such intimately mixed clays.

- Extraneous mineral matter consists of waste bands and lenses in the seam (interburden and parting), and shales sandstones and intermediate rocks introduced into the mined product from the roof, floor and parting of the seam. Most of this material is easily removed by coal preparation techniques.

- Other forms of extraneous mineral matter are pyrites (fool's gold), and ankerite or calcite (thin white flakes often found in the joints of coal, and sometimes in the bedding plane). These are secondary minerals, deposited in the coal seam after its formation.

The properties which are generally taken into account in assessing the rank of coals are the following;

- **Moisture content** - the loss in mass on heating to 105 - 110°C.

- **Volatile matter content** - the loss in mass on heating to 900°C, less moisture.
- Ash content - the residue after burning the coal under controlled conditions.

- Fixed carbon – the amount of carbon in the coal, expressed as the difference between sample mass and mass of moisture, volatiles and ash combined.

The above properties are often referred to as the proximate analysis results. In addition, a general analysis would include, amongst others;

- Ash fusion temperature - low fusion temperatures cause difficulties in boilers and other appliances.

- Calorific value (C.V.) - generally the most important and is the amount of heat energy produced by the combustion of a specified mass of coal under specified conditions. In South Africa it is expressed as megajoules per kilogram (MJ/kg).

- Coking properties which are largely based on the degree of melting of the coal, or swelling characteristics (swell index) of the coal on heating. Also determined as the Roga Index.

- Sulphur content – important from an air pollution and metallurgical perspective.

- Grindability – to determine the extent to which a coal can be pulverised, usually part of the power generation process.

- Abrasiveness – to determine the propensity of a coal to wear the mill during pulverisation and transport. A critical parameter in many processes including power generation and often as a result of sandstone or pyrite in the coal.

Also of importance are weathering and liability to spontaneous combustion, the latter being an important consideration in the mining method selection.

Rank may also be assessed based on the Fuel Ratio, i.e., the ratio of fixed carbon to volatile matter on a dry, ash-free (daf) basis. The volatile matter is taken as the gas and vapour, less water. Thus;

\[
\text{Fuel Ratio} = \frac{\text{Percentage fixed carbon}}{\text{Percentage volatile matter}}
\]

Bituminous (Humic) coals include household, coking and steam coals and are important industrially in South Africa. They are characteristically dark, brittle and banded; dense, well-jointed and break conchoidally; original vegetable matter is not detectable by
eye; the fixed carbon ranges from 47% to 84%, the moisture is low (3 - 11%) and volatile matter, high (12 - 40%).

The macroscopic constituents (visible by naked eye) of bituminous coal comprise:

- Vitrain which forms the black, shiny bands in the coal that are up to 1 cm wide. So-called bright coals have a high proportion of vitrain.

- Durain is black or grey, the dull grey durain is often a dominant constituent of South African coals, especially when seams are thick. Coals that have a high proportion of durain are hard and tough and can be handled without deterioration. They are good stream coals but may contain high levels of ash.

- Clarain has a satiny lustre and consists of alternating thin bands of vitrain and black durain. The ash content tends to be high. Coals dominant in clarain have high volatiles, ignite easily and are therefore good household coals. Clarain is not an important component of South Afflean coals.

- Fusain is soft and charcoal-like. The ash content is high and the coal is typically friable and has a tendency to cause dust.

A coking coal, when heated slowly, produces a light, porous material with high carbon content and low ash. Bituminous coal of medium to high rank, slightly devolatised is also suitable, but in South Africa, the devolatilisation is rarely achieved and these coals are in great demand.

Anthracite represents the end product of the coalification process and has the highest rank. It is a jet-black, brittle hard coal that has a high lustre. It ignites slowly, burns with a short blue flame and has a high heating value. In South Africa, there are no real anthracites due to the absence of lateral pressure during coalification, only pseudo-anthracite associated with the intrusion of dolerite into the coal measures. As a result of their porosity, groundwater soaking through the coal beds tends to deposit secondary minerals giving rise to high ash contents.
2.1.5 Coal Production and Markets

The grade of coal supplied depends on the market and South Africa mines in the order of 285Mt “run-of-mine” (ROM) coal to produce some 220Mt product for various markets. This product is typically 2Mt pseudo-anthracite and 218Mt bituminous coal, 66Mt of which was export coal. Currently, mined coal in South Africa is consumed by power production (mainly Escom, 59%), petrochemical manufacture (Sasol, 26%), various industrial (8%), metallurgical use (largely Iscor, 4%), and domestic sales (3%). Figure 2.2 illustrates the role of coal in South Africa’s energy market (size of primary supply relative to market share) and the various resource end-uses.

Consumers require consistent quality coal of high CV, low ash and in precise size ranges. These needs, when set out against the actual properties of coal as mined, mean that some sort of coal preparation is usually required. This preparation must be engineered to meet the difference between the quality of the coal produced from the mining operation – run-of-mine (ROM) and the needs of the consumers supplied by the mine. There is no preparation pattern that can be applied to every mine, and poorly controlled mining makes preparation of the coal problematic and inefficient.

The major local consumer of coal in South Africa are power stations. They require small coal, usually less than 25mm in size, with not too much fines content. If the coal has to be transported considerable distances, e.g. to the Cape, then high CV coals must be supplied, to

Figure 2.2 Coal resource end-uses in South Africa’s energy market, primary supply showing relative contribution to energy market, (after Surridge, 2002)
avoid high transport charges on incombustible material. On the other hand, a captive mine, supplying an immediately adjacent power station via a conveyor belt, can supply a lower CV fuel; though the coal must conform to the size grading required for convenient handling and storage. This is because the transport charges form only a small part of the delivered cost of the fuel; much less than the cost of preparing the coal to a higher standard. Usually, only a part of the coal may need to be washed (to remove poor quality coal fractions) and most of the coal is only crushed to meet the size requirements of the power station.

In general terms, power station coal should have a raw coal calorific value up to 26 MJ/kg - coal with a higher calorific value has a higher value in other markets. The volatile matter should not exceed about 20% and sulphur should be less than 1% (to avoid excessive air pollution). A minimum ash deformation point of 1200°C is desirable in order to avoid clogging of burners, together with other specific requirements such as abrasiveness limits, swelling index, etc.

Industrial users of coal other than power stations require coals that are in fairly close size ranges and of high calorific value. The volatile matter must also be of a certain value (though this cannot, to a great extent, be controlled by the washing process). Hence mines supplying such markets must have either raw coal of the required CV or (more likely) a preparation (washing) plant.

To meet the specifications of straight coking coal (i.e. unblended) most South African coking coals have to be washed (processed) and this should give a yield of about 50%. The ash content should preferably be less than 9%, sulphur should be reasonably low and the phosphorus content must not exceed 0.15%. The swelling index should be as high as possible and not less than 4, and the Roga index at least 60 (the Roga index is a measure of coking power and is really a measure of the resistance of a coke button to abrasion). Volatile matter should lie between 19 and 30%.

The calorific value of coal for petrochemical use is not so important and ash up to 35% can be tolerated since the Sasol process essentially involves hydrogenation and the coal is simply treated as a source of carbon. However, sulphur should be lower than 0.65% and nitrogen less than 1.5%, while the minimum deformation temperature is 1380°C.

Table 2.3 summarises the size ranges of coals sold whilst Table 2.4 gives the quality of coals sold on the South African market. The figures are only indicative of average sales weighted values, and certain grades may be marketed which are outside the values given.
### Table 2.3  Typical coal size ranges noted in Table 2.4

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<thead>
<tr>
<th></th>
<th>Bulk size range (mm)</th>
<th>Max fines (&gt;4mm) (%)</th>
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<tbody>
<tr>
<td>Large cobbles</td>
<td>-150 to +31,5</td>
<td>7,5</td>
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<tr>
<td>Cobbles</td>
<td>-100 to +31,5</td>
<td>7,5</td>
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<td>Large nuts</td>
<td>-71 to +31,5</td>
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<td>Nuts</td>
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<td>Peas</td>
<td>-25 to +6,3</td>
<td>12,5</td>
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<tr>
<td>Smalls</td>
<td>-25</td>
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<tr>
<td>Duff</td>
<td>-6,3</td>
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### Table 2.4  Typical coal qualities and specifications

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<tr>
<th>Consumer</th>
<th>Product (size in Table 2.2)</th>
<th>Ash %</th>
<th>Volatile matter %</th>
<th>Calorific Value MJ(kg)</th>
<th>Swelling Index</th>
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</thead>
<tbody>
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<td>Nuts</td>
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<td>Peas</td>
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<tr>
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<td></td>
<td>Duff</td>
<td>12-23</td>
<td>21-31</td>
<td>22-29</td>
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<td>Captive power station</td>
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<td>21-26</td>
<td>19-25</td>
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<tr>
<td>Blend Coking coal</td>
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<td>30-34</td>
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<tr>
<td>Straight coking coal</td>
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<td>Export steam coal</td>
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<td>Low ash export coal</td>
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Notes

PSS – power station smalls
Ultimately, to evaluate a resource in terms of its marketability, and more importantly, to establish the possibility of mining such a resource at a profit, requires an exploration and geological or ore resource evaluation system, operating in parallel with a mine planning and techno-economic assessment methodology.

2.2 Generalised Approach to Ore Resource to Reserve Delineation

To determine the suitability of a particular mineral resource for surface mining, the starting point is an exploration program which will determine if and where a deposit may exist in the first instance. Once a deposit is recognised, further detailed exploration is required to fully define the deposit and, with increasing knowledge and data reliability, select a mining method and other modifying factors that will eventually redefine the resource as an economically mineable material. Figure 2.3 illustrates the generalised approach, from exploration to reserve delineation.

Figure 2.3  Approach to delineating an ore reserve
Exploration includes activities and evaluations necessary to facilitate the discovery and acquisition of coal deposits that may be extracted and processed economically, now or in the future. The extent of exploration programs will vary according to company practices and the area under investigation (a potential new mine or an existing mine for reserve expansion). The program should go to the point of detail where additional study would not provide greater reliability. Typical investigation techniques are:

- **Background study**
  - Literature search
  - Mining history
  - Photo geology

- **Surface mapping**
  - Photo geology & remote sensing
  - Detailed geological mapping

- **Subsurface investigation**
  - Rotary drilling
  - Core drilling
  - Geophysical logging
  - Survey old workings

- **Geophysical investigation**
  - Seismic reflection
  - Seismic refraction
  - Seismic cross hole
  - Electrical methods
  - Gravity methods
  - Magnetic methods

- **Field testing and instrumentation**
  - In-situ stress
- Modulus values
- Groundwater testing
- Vibration

- **Laboratory testing**
  - Stress-strain
  - Creep
  - Strength
  - Weathering rate
  - Diggability

- **Geophysical**
  - Geochemical
  - Soil testing
  - Water quality
  - Reactivity
  - CV (MJ/kg)
  - Proximate analysis

It must also be recognised that exploration is a sequential process where after each exploration stage a decision must be made to either continue or abandon the exploration program. Figure 2.3 implies a succession from resource to reserve through increasingly detailed studies, data reliability and modifying factor determination. In reality, the process must be evaluated at each level of exploration study, as illustrated in Figure 2.4.
Areas subjected to exploration procedures may be broadly classified into areas where coal seams are known to exist and virgin areas. The coal exploration programs may be defined as follows:

- **Geological prospecting with minimum drilling.** This type of exploration is common where outcrops are abundant and geologic formations are well known.

- **Random drilling.** This method is applied to previously explored regions with high geologic predictability.

- **Grid drilling and statistical approaches.** A program of this type is necessary in virgin areas or in areas of high geological variability.

The most common type of field exploration is a drilling program. The purpose of a drilling program is to provide a comprehensive data base covering the location of the coal and the overburden. Therefore, drilling must be planned to yield maximum information. Drill hole patterns will vary according to the terrain, geological
formations, the presence of coal and the type of exploration program and end-use of the data in the mine planning process.

When very large areas are being studied, hole spacing will vary greatly and will probably not be in any set pattern. When the program is narrowed to a regional study, a grid pattern is most common. Detailed exploration is performed in target areas where coal is known to exist, and therefore, closer spaced drill patterns are required.

One of the main advantages of a drilling program is that it can provide physical samples of the coal and the overburden for chemical and physical analyses. This is accomplished by coring. In most programs, a percentage of the holes are cored for overburden and coal. Usually, this ranges from 10 to 25 percent but is dependant on the structural and sedimentologically complexity of the area. Holes should be cored from the surface, through the coal, and then below it in order to maximize information about waste (spoil) quality and pollution potential.

Depending on the top size of the product, large-diameter drilling may be necessary to evaluate product quality, coal washing, product yields, preliminary coal processing plant design and preliminary coal utilisation aspects.

2.3 Exploration and the Mine Planning Process

The exploration process is allied closely to the phases of a systematic mine planning process as discussed in a later Module. The decision points shown in Figure 2.4 are evaluated not just on the basis of geological information alone, but must also be supported by techno-economic assessments too.

Mine planning is an engineering discipline in which decisions are made using known data (or data which has certain probability of being correct) to arrive at economically optimal solutions. Within the exploration and prospecting phases, the job of the geologist is to supply a representation of the orebody - initially making little or no judgments as to economic viability or potential mining constraints until these are examined by mine planners. Detailed geological evaluation is clearly directed at the most economic targets, however, the decision on the economic criteria to use is one for the mine planning engineer.
In general, as the mine planning phases increase in detail, so must the exploration and prospecting geological data. Typical mine planning phases and the associated geological data and study requirements are:

- Conceptual mine plan with geological target generation study
- Pre-feasibility plan with regional geological, project area, preliminary reserve and planning for detailed target area evaluation studies and limits assessment (geological and economic)
- Cost ranking and feasibility plan with detailed target area evaluation studies based on a high density exploration drilling program

The geoscientific data gathering and analysis should, in addition to supplying information from which a geological model of the resource can be developed, also generate data to enable mine planning staff to conduct a techno-economic analysis of the various factors which impact the selected surface mining methods, for example in strip coal mining:

- Highwall stability
- Spoil stability (geotechnical and chemical)
- Road design
- Water handling
- Excavation system
- Mined-out areas
- Spontaneous combustion

and also those factors which impact on the required surface infrastructure for the mine, i.e.;

- Siting
- Structures
- Storage and stock piles
- Cut and fill works
- Transport systems
- Impoundments and waste dumps

Table 2.5 shows the relationship between the exploration techniques mentioned in section 2.2 with the mining considerations listed above.

The geophysical factors which impact on the mining considerations listed above are classified in Table 2.6 according to their importance. This tabulation clearly shows the importance of local geology, surface characteristics and rock and soil property exploration data in determining the mining considerations and thereby reducing the level of planning risk at each stage of the planning process. Especially relevant at pre- and full feasibility study phases are hazards and associated risks that could impact on safety, health, environment and the technical and economic integrity of the mining operation.

### 2.3.1 Conceptual mine plan with geological target generation study

A geological target generation study is the initial geological evaluation aimed at identifying a particular coal deposit or a number of coal deposits, with sufficient physical and coal quality continuation, that will satisfy the identified market requirements. This phase usually consists of a desktop study, including a literature review of all available data relative to the area under consideration. Limited fieldwork, where applicable, is recommended for the evaluation of a coal occurrence that may be suitable for the establishment of a mining project.

The initial step in all exploration programs is a literature search to obtain all public information and previous exploration data concerning a target area. This will assist in establishing the geological setting of the area, identify any previous geological exploration work and mining activities performed in the area and establish surface and mineral ownership and title. From this information, a more reliable assessment of the level of detail required in any future field mapping can be determined. In some areas, regional mapping may be useful during the reconnaissance phase and provide the framework for the successive evaluation phases. Field mapping may include the tracing of marker beds, the measuring of stratigraphic sections and the location of major faults,
igneous intrusions and any other features that may affect the continuity of coal seams.

Table 2.5  Relationship between exploration techniques and mining considerations.

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<thead>
<tr>
<th>Investigation techniques</th>
<th>Mining considerations</th>
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2-22
Table 2.6  Relationship between geotechnical factors which impact on mining.

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<th>Surface Facilities</th>
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Once a detailed target generation study and the associated conceptual economic assessment reveals an economic mineral deposit may exist, exploration is then defined prospecting and is an integral part of the mine planning process. The stages involved from this point onwards are:

- Securing a prospecting license for prospecting and test work
Prospecting for the purposes of a detailed pre-feasibility study

Additional prospecting for a (bankable) feasibility study

If the conceptual economic assessment indicates that the proposed target area warrants further geological exploration and test work, a prospecting permit in terms of Sections 6 and 8 of the Minerals Act, 1991 (Act 50 of 1991) must first be obtained from the Director, Mineral Development, Department of Minerals and Energy. Each application for permission to prospect must be supported and accompanied by the documentation listed in Figure 2.5.

Figure 2.5 Procedures and documentation requirements for a Prospecting Permit in terms of Sections 6 and 8 of the Minerals Act, 1991.
2.3.2 Pre-feasibility mine plan with regional geological, project area, preliminary reserve and planning for detailed target area evaluation studies

A pre-feasibility study is defined following the South African Guide to the Systematic Evaluation of Coal Resources and Reserves (SABS 0320: 2000) and the SAMREC Code as:

“the first evaluation, planning and design study following a successful geological exploration campaign. It provides a preliminary assessment of the economic viability of a coal deposit and forms the basis for justifying further investigations. It summarises all geological, mining, coal processing, engineering, environmental, marketing, legal and economic information accumulated on the project”.

In terms of the above definition, the pre-feasibility study should at least be based on indicated coal resources. Indicated coal resources can be defined as that part of an in situ coal resource for which tonnage, densities, shape, physical characteristics and coal quality can be estimated with a moderate level of confidence. It is based on exploration, sampling and testing information gathered by means of appropriate techniques from locations such as outcrops, trenches, pits, workings and boreholes. To be classified as an “indicated coal resource”, the borehole density must support a reasonable estimate of the coal resource, bearing in mind complications which may arise in structurally and sedimentologically complex areas and areas with significant variability in the coal quality. Minimum requirements are:

- four boreholes per 100 ha (approximately 500 meter spacing) for multiple-seam deposit types, or
- one cored borehole with quality data per 100 ha (approximately 1 000 meter spacing) for thick interbedded-seam deposit types.

2.3.3 Cost ranking and feasibility mine plan with detailed target area evaluation studies based on a high density exploration drilling program

A feasibility study assesses in detail the technical soundness and economic viability of a mining project. It is seldom undertaken unless there is a reasonable assurance that the proposal is viable and represents a detailed economic evaluation that serves as a basis for the investment decision and allows for the preparation of a
bankable document for project financing. The study constitutes an audit of all geological, geotechnical, mining, engineering, coal processing, environmental, marketing, legal and economic information accumulated on the project. (SAMREC Code and SABS 0320:2000.)

For the geological investigations that form part of the feasibility study, the cored borehole spacing must be reduced to a minimum of eight cored boreholes per 100 hectares (approximately 350 meter spacing) for all coal deposit types in order to define the coal deposit as a measured or indicated coal resource. The feasibility study may not be undertaken on inferred coal resources. A “measured coal resource” is that part of an in-situ coal resource for which tonnage, densities, shape, physical characteristics and coal quality can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered by means of appropriate techniques from locations such as outcrops, trenches, pits, workings and boreholes. The locations are spaced closely enough to confirm physical and coal quality continuity.

The outcome of the exploration phases, together with the techno-economic mine planning analysis, will determine if the coal resource can be classified as a coal reserve. From this point onwards, a bankable feasibility document is produced for the purposes of informing potential investors of the exploration results and mining prospect. Thereafter, once funding is in place and a mining authorisation is secured, mine establishment and construction would begin.

2.4 Delineation of Land Acquisition Requirements

Allied to the exploration and resource to reserve delineation exercise are the property acquisitions that would be required (as surface and/or mineral rights). Considerations include the type of land required, the priority of the acquisitions and the acquisition schedule itself.

The procedure begins with the determination of the economic mineral resources available to the project through the preliminary exploration target identification process. At this point, the land ownership identification would also begin. This would obviously include the mineral rights and surface rights, but must also be extended to cover;
- **Mining areas.** Including the actual portions of the mineral deposit that will be exploited during the mining operations. In the case of surface mining, the effect on the property is obvious. Land requirements could also include areas intended to provide a ‘buffer’ between the active operations and the mine’s neighbours.

- **Access.** Provided for both personnel and suppliers. The nearest public road is usually the start point and the mining plan identifies the location within the project area to which the access corridor must lead. Additionally, haul road, rail or conveyor access may be required. In all cases, these corridors will have to be purchased at relatively high cost (especially in the case where bridges, cutting and diversions are required, or the use of local infrastructure requires some upgrading).

- **Facilities.** Includes offices, workshops, warehouses, equipment service facilities, outside storage, explosive magazines, and processing plants including waste disposal areas. The preferred locations for these facilities will be determined during the planning process.

- **Utilities.** Including incoming electrical power, internal power distribution, water supply and distribution, sewage treatment and disposal, and communications lines, etc.

The acquisition process begins with a determination of the strategic importance of a parcel of land, based on its economic impact on the project if not acquired. A parcel will be strategically important due to the mineral resource itself, but also possibly due to topographical suitability for the construction of mine structures, or it may provide access to a number of other parcels. Typical priorities are:

- The bulk of the mineral resource properties.

- Properties required to avoid costly alternative or sub-optimal mining or mineral resource utilisation plans. The economic impact on the project without these parcels will be strongly negative, since alternatives will be much more costly.

- Surface rights that could be acquired through similarly priced alternatives, without a significant impact on the project viability.

- Low priority parcels not absolutely necessary and may only be desired as slight enhancements or for their “nuisance value” to the project.
Determination of the appropriate level of priority to place on an individual piece of property will often require an economic analysis. Occasionally, this analysis will compare the acquisition of a particular property with the acquisition or use of an alternative property that could serve the same purpose.

In addition to priority, the timing of the acquisition can impact on project cash-flow. The time-schedule priorities are often:

- Surface or mineral resource properties that have a high strategic importance and thus serve to lower the level of risk to the project
- Properties required to proceed with licensing
- Properties required before the commencement of construction and initial mining activities
- Properties required sometime later that may provide for future expansion or extension of the mine if the economic operating climate extends favourably into the future.

The purchase of mineral or surface rights entails determination of the level of control required. Alternative levels of control that can be considered are:

- Options to lease or to purchase (usually only considered in the early stages of exploration and target identification).
- Outright ownership of surface and minerals.
- Lease of surface and/or minerals
- A combination of ownership and leasing.

Surface and mineral rights acquisition is central to the development of a mining project, since not only is the value of the land and the underlying minerals often the basis upon which project financing is extended, but without the legal right to affect the properties involved, licenses required for mining projects will not be granted. If acquisition is poorly planned, the incorrect parcels of surface and mineral rights may be acquired or it would impact on project cash-flows (acquired too early or late or at excessive cost).
# SURFACE COAL MINING METHODS

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<td>Terrace mining method and layout</td>
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**Apply, calculate or predict**

- Strip mine bench layout for specified coal production and strip widths
- Dilution due to previously mined areas
- Strip ratio variation for specific terrace panel layouts

**Evaluate or design**

- Ore body characteristics for method selection
- Equipment requirements for strip mining
- Cash flows for various box cut orientations
- Methods for accessing previously mined reserves
- Likelihood of a coal type self combusting
3.1 Surface Mining Method Selection

The ore deposits being mined by surface mining techniques today vary considerably in size, shape, orientation and depth below the surface. The initial surface topographies can vary from plains, gentle undulations to hills and valleys. In spite of this, there are a number of geometry based design and operating considerations fundamental to them all. These are the focus of this section of the Module.

Ore reserves suitable for surface mining can be classified initially as:

- Relatively horizontal stratified reserves with a thin or thick covering of overburden – this is most typical of the Middle Ecca series coal-seams in South Africa
- Stratified vein-type deposits with an inclination steeper than the natural angle of repose of the material so that waste cannot be tipped inside the pit
- Massive deposits, deep and very large laterally such that dumping of the waste within the pit is not possible.

Of all the variations of surface mining methods available, since the majority of near surface coal seams in southern Africa are amenable to exploitation by either strip mining or terrace mining, these methods will be covered in more detail.

Figure 3.1 refers to the classification of these ore reserves and the selection of mining methods. Note how the general classification system is applied, from stratified (or layered) types of deposits, typical of coal-seams, through the type of layering, the thickness of overburden and finally the means by which overburden waste is handled; specifically, in-pit or ex-pit waste handling sub-systems. It can be seen from Figure 3.1 that a further consideration is the type of material (waste or ore) handling systems that can be used in each type of mining operation, namely cyclic (discontinuous) or continuous systems. Refer to Modules six and seven for more information on this concept.

This simplified classification is based on ore reserve and waste geomorphology only. As will be seen when the various surface mining methods are introduced in more detail, additional considerations apply, related to specific ore and waste geometrics, topography, equipment selection and unit operations.
The selection of a specific surface mining method forms an integral part of the mine planning process and the evaluation of various method options is based on the objective of most cost-effective mining method commensurate with maximum utilisation of available coal reserves, safety and health, during the expected life cycle of the mine. Whilst the evaluation system is described in detail in Module ten, the basic selection methodology is discussed here.

Figure 3.1 Classification of surface mining methods

The basic steps in the selection methodology are shown in Figure 3.2 and listed below;

1. Identification of physical limits and constraints, such as surface structures to be protected, mine boundaries, geological features, servitudes, rivers, dams and other water courses.
2. Identify and list the required design criteria for the mining method selection process.

3. Data collection for method selection process, including;

3.1 Surface plans indicating existing infrastructure, e.g. roads, railway lines, water, supply lines, power lines, infrastructure that needs to be protected, location of rivers, streams, dams and potential flood buffer lines, surface contour lines (digital terrain model), land utilisation, e.g. extent and intensity of agricultural, industrial and other economic activities, townships, dwellings and other residential areas, sensitive ecological and environmental areas.

3.2 Mining area plans indicating property boundaries, extent of mineral/coal rights and reserves and coal outcrops or sub-outcrops.

3.3 Geological information pertaining to depth of overburden contours, coal seam thickness contours, coal quality contours, reactivity and spontaneous combustion, inter- and in-seam parting thickness, geological structures, e.g. sills, dykes, faults, burnt coal, hydrological areas, etc. and previously mining areas. The results of a geotechnical and rock mechanics investigation defining the extent and location of the different geotechnical areas.


3.5 Alternative mining methods that can be considered: Identify and list possible mining methods that can be considered for the coal deposit under investigation.

3.6 Available technology: Identify and list the latest technologies and developments that could influence the mining method selection process, for example mining equipment available, mining operational procedures and management and control strategies.

4. Method unit operating costs: Typical unit operating costs for different mining methods and processes, to establish an approximate cost of mining. Unit operations are introduced in the next section, but operating costs estimates would include;
4.1 cost of removing a unit of waste, and all of the associated operations performed on it (such as drilling, blasting etc.), and

4.2 cost of mining and transporting a unit of coal, and of performing the associated operations on it, and

4.3 cost of beneficiating coal and transporting and marketing the product, and

4.4 the percentage recovery of the coal, after providing for method-related pit losses, dilution, plant yield, moisture changes etc.

5. Identify and list all the significant hazards and risks associated with human activities, natural processes, mining and engineering activities during the operating and closure phases of a surface mining method option. Identify procedures, systems and/or design principles that will eliminate, reduce or control the identified risks and highlight these procedures for preliminary surface mine design specifications.

Figure 3.2 The basic steps of the mining method selection methodology
3.2 Unit Operations for Surface Mining Methods

A number of unit operations are common to any surface mining method and as shown in an earlier Module, the cost of these individual operations can be summed to form an estimate of the cost of mining. Surface mining of coal is conducted in a relatively simple sequence of operations which includes:

- preparing the surface
- drilling
- blasting
- overburden removal
- loading the deposit
- haulage of the mined deposit
- rehabilitation.

A typical equipment inventory for a 4Mtpa ROM surface coal mine is shown in Table 3.1, for strip mining and terrace mining. A short discussion of these unit operations in which the equipment is utilised follows and more specific details are presented in the Modules dealing specifically with that unit operation and/or equipment.

3.2.1 Preparing the Surface

Preparing the surface includes the removal of all vegetative cover in preparation for other mining operations. While this operation is not always necessary, it may be necessary to clear the land of trees or other obstructions.

Regulations in South Africa require that topsoil be removed and stored for later use. It is not easy to universally define what is topsoil though it is generally understood to be the soft layers of soil over which the current vegetation has established its roots. These layers or horizons are contrasting layers of soil lying one below the other, parallel or nearly parallel to the land surface. Soil horizons are differentiated on the basis of field characteristics and laboratory data. The three major soil horizons are:
Table 3.1 Typical equipment inventory for a 4Mtpa ROM, 3BCM/t strip ratio surface strip (with dragline) or terrace (without dragline) coal mine

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3-7
1. **A horizon.** The uppermost layer in the soil profile often called the surface soil. It is the part of the soil in which organic matter is most abundant, and where leaching of soluble or suspended particles is the greatest.

2. **B horizon.** The layer immediately beneath the A horizon and often called the subsoil. This middle layer commonly contains more clay, iron, or aluminum than the A or C horizons.

3. **C horizon.** The deepest layer of the soil profile. It consists of loose material or weathered rock that is relatively unaffected by biological activity.

### 3.2.2 Drilling and Blasting

Once the weathered overburden has been exposed following topsoil removal, to allow easier handling by stripping equipment later the unit operations of drilling and blasting to fracture the rock mass may or may not be required. Where the ground cover is hard, it is usually drilled and blasted. Soft strata may often be directly excavated – termed “free-digging” and with the advent of increasingly powerful bucket breakout forces on hydraulic shovels, the definition of soft – or “diggability” is shifting.

### 3.2.3 Overburden Removal

Overburden removal is the most important aspect of the mining system. The equipment and methods used to remove overburden must be carefully chosen to provide the required production at the minimum cost, due to the multiplier effect of the stripping ratio as discussed in Module one. The stability of the highwall on which the excavation equipment stands (in the case of a dragline) or directly below it (in the case of coal or overburden loading shovels) and the spoil pile stability is an important consideration in pit design.

### 3.2.4 Loading and Haulage of the Deposit

Coal loading is usually carried out by a loading shovel, either a front-end loader, rope shovel or hydraulic front or back-hoe. The specific selection will depend on the characteristics of the mining method, available pit room, coal geology (especially the presence of parting) and other production and equipment constraints. Transport out of
the pit is achieved mostly by large haul truck – rear or bottom dump types.

### 3.2.5 Rehabilitation

Rehabilitation includes backfilling, regrading, surface stabilisation, revegetation and restoration operations. It is a legal requirement at the start of mining operations to specify how mining will impact the environment and what measures the mine will take to restore the land to its pre-mining capability. These details are defined in the mine’s environmental impact assessment and environmental management program report (EMPR) or management plan.

Backfilling is achieved through the use of virtually any of the equipment used in overburden handling, the most cost-effective option being to excavate and dump the waste in its final backfill location. However, approaches to segregation of material, specific burial, and layering and compacting can be distinguished as distinct backfilling steps. Once back-filled, regrading is typically achieved by wheeled or tracked dozers. The amount of regrading depends upon the care with which backfill is done, and on the degree of topsoil restoration and grading, which is either desired or practicable.

Considerations of surface stabilisation include;

- provision for water quality maintenance
- compaction and layering
- sealing the spoil layers.

It also encompasses amendments and therapeutic actions which revitalise the surface or prepare it for revegetation, such as mulching, fertilizing, liming, stabilising, diskng, harrowing, etc. One phase is usually the spreading of topsoil on the regraded soil. Scrapers are often used to reclaim soil from the stockpile, transporting it, and spread it on the spoil. Seeding is actually the final stage of rehabilitation following which revegetation and restoration takes place over a longer time period. Closure will eventually result at which stage all the criteria as defined in the EMPR have been met in order to minimise the impact of the mine on health, safety and environment. These concepts are discussed in detail in a later Module.
3.3 Strip Mining

Also referred to as opencast mining, the method constitutes the selective extraction of overburden, interburden and the coal deposit, and is used for mining relatively shallow and low angle of dip tabular deposits. A unique attribute of strip mining operations is the systematic sequence in which the various mining activities are performed.

The utilisation of strip mining techniques has long been accepted by the international coal mining industry as both an effective and efficient method of coal recovery and as a cost-effective alternative to underground methods. One of this method’s primary advantages over underground mining is the cost-effective high-percentage extraction of coal seams too shallow for underground methods, together with, in ideal conditions, total extraction of the in-situ coal seam. Other advantages of strip mining include the generally lower working costs and higher productivity. For these reasons it can be anticipated that strip mines will be an integral part of the international and domestic coal mining industry for many years to come.

Strip mining is ideally applied where the surface of the ground and the orebody itself are relatively horizontal and not too deep under the surface, and a wide area is available to be mined in a series of strips or cuts. Typical examples of this type of mining are the larger tonnage coal mining operations in Mpumulanga.

Favourable conditions are:

- Relatively thin overburden (0-50m maximum otherwise stripping ratio and cost of stripping can become too high – although the exact limit would depend on the BESR as discussed previously).

- Regular and constant surface topography and coal layers (not more than 20° variation from horizontal on the coal seam – topography can vary more since pre-stripping can be used to level it – but this is expensive to apply).

- Extensive area of reserves (to give adequate life of mine (LOM) and to cover all capital loan repayments – typically more than 20 years life at 4-14mt per annum production).

Walking draglines have been for many years the most popular machine for overburden excavation in this type of mining due to their flexibility, utility and availability, but more importantly, their low operating costs for waste mining (R/t or R/BCM). The dragline is a typical combined cyclic excavator and material carrier since it both excavates material and dumps it directly (without the use of trucks or conveyor belts). The dragline sits above the waste or overburden
strip block, usually 50m or so wide, on the highwall side and excavates the material in front of itself to uncover the top of the coal seam, and then dumps the overburden on the low-wall or spoil side of the strip, as shown in Figures 3.3 and 3.4 which also shows the terminology used. Figure 3.5 and 3.6 illustrate the typical method of strip mining, showing in addition to the dragline overburden stripping unit, coal loading and hauling using spoil-side (low-wall) ramp roads (Figure 3.5) and low- and high-wall ramp roads in Figure 3.6.

Figure 3.3 Walking dragline for strip mine overburden handling

In a dragline operation, two long walls are formed in the initial box-cut. One of these, the low wall, remains and is eventually covered by spoil. The other, the highwall, is progressively excavated and occupies a new position with each strip. The maximum highwall height is usually of the order of 45-50m, with a variable depth of chopping or prestripping above the dragline working level.

For maximum productivity, a long strip is required (ideally over 2km in length) to reduce excessive “dead-heading” time and to accommodate the various in- and ex-pit activities (shown in Figure 3.7) and floor stocks or buffers (the amount of production stock available between mining cycle activities – typically drilled, blasted, exposed, coal-on-floor, drilled coal and blasted coal). Whilst longer pits increase the floor stocks or buffers and facilitate easier unit operations, they also increase the risks of time dependant slope failure in both the highwall and the (waste) low-wall, may be more prone to spontaneous combustion problems and take up large surface areas that can cause rehabilitation and transport problems. If mixing of coal is important (to meet sales specifications) then long
strip lengths are also problematic in terms of the active mining fronts available for blending the coal. Where highwall or low-wall stability is problematic, it becomes necessary to monitor the stability of the pit extensively.

Nowadays, several large strip mines operate in areas that were previously mined by underground methods. In such cases it is difficult to anticipate the stability of the overburden and geotechnical surveys are required especially where underground rooms are required to be blasted (by collapsing the pillars between) prior to using a dragline on these areas. This aspect is dealt with in more detail later in this Module.

Figure 3.4  Strip mining terminology
3.3.1 Strip Mine Layout

The fundamental objective of any commercial mining operation is the extraction of the mineral deposit at the lowest overall cost in order to maximize profits and resource utilisation. In strip mining the selection of cut parameters, both in terms of direction and width, contributes significantly to the overall efficiency and profitability of the operation. The initial decision to be taken, following selection of the method itself, is with regard to the location of the first strip (the box-cut) and cut direction.

**BOX CUT ORIENTATION AND DIRECTION OF ADVANCE**

Box-cut orientation is frequently determined with due consideration to excavation volumes and strip ratios, costs, economic returns, property boundaries, transport gradients or pit drainage and important geotechnical constraints. The selected cut direction usually has a major influence on the highwall stability, as its choice can cause particular joint sets, shear planes or faults to be favourably or unfavourably orientated in terms of slope stability. Ignoring geotechnical constraints may temporarily produce large economic returns, but the complex interrelation of lithology,
structures, weathering elements, ground water and slope geometry invariably result in geotechnical hazards causing losses of coal production which ultimately reduce the overall profitability of the mining operation.

Figure 3.6  Strip mining with draglines (on overburden) and high-wall and low-wall access ramp roads

The primary cut directions generally considered in a gently dipping coal deposit as illustrated in Figure 3.8 are;

1. Parallel to the coal outcrop limit or strike, advancing in a down-dip direction.
2. Parallel to the strike of the coal deposit, advancing in an up-dip direction.
3. Parallel to the dip of the coal deposit, advancing along strike.
4. Oblique cuts.
Since the initial financial outlay in developing any strip mining operation is large, with most of the capital cost being accounted by mining equipment purchases, it is desirable to attain an early start to production, hence an early return on the capital investment. This criterion is generally met by positioning the initial mining cuts parallel to and as close as possible to the outcrop limit of the coal deposit and proceeding down-dip. Because of the initially low overburden...
cover in the shallower coal deposits, the overburden stripping costs are reduced considerably. In addition, a relatively small initial box-cut is required, which poses no serious problems to siting of the waste material generated from this initial excavation. However, having selected the initial mining equipment, thereby fixing the stripping capacity, the increase in overburden volumes down-dip, with time, will result in a decrease in coal exposure rate and hence coal output. This will entail purchasing of additional mining equipment to cater for the increase in overburden volumes in order to maintain a constant coal output.

When selecting a strike box-cut with up-dip advance, the large volume of waste material from the initial box cut requires a larger spoil pile area of correspondingly greater height, and, given the same rate of coal production, would require an initially larger overburden stripping capacity.

Economically, cuts positioned parallel to the dip of the coal deposit are less favourable than strike cuts with down-dip advance, but their selection can be justified by overriding geotechnical and environmental considerations. Coal output from such an operation is constant over the project life even though short term fluctuations in coal exposure rates will be experienced, depending on the relative position of the dragline along the cut.

In practice, site boundaries often dictate the orientation of the initial box-cut, and may result in oblique cuts. In this case, the specific advantages that would accrue, compared to the disadvantages of fluctuating coal exposure rates, are very much dependant on the box-cut orientation with respect to coal seam and overburden.

In order to assess the economic influence of the box-cut directions mentioned above (1-3) for a particular coal deposit, a detailed investigation in terms of cash flow projections, based on equipment purchasing cost, box-cut completion, initiation of additional stripping capacity purchases and completion of additional stripping capacity purchases related to coal and overburden geometry and production rates is required. Typical results are shown in Figure 3.9.

Traditionally, the practice in South African surface strip coal mines has been to commence operations in the area of lowest overburden to coal ratio. This has the advantage of achieving full production earlier and providing a faster cash flow for improved economics. Draglines, by the nature of their direct casting technique, are normally forced to commence stripping at the sub-outcrop of the coal. Fortunately the sub-outcrop is often the area of lowest stripping ratio.
Figure 3.9  Cash flow comparison for various box-cut directions defined in Figure 3.6 (modified after Atkinson, 1989).

There can be little argument as to the wisdom of this approach, however, an increasing number of new mines are finding that the mining cannot commence in the area of lowest strip ratio for any of the following reasons:

- Mine boundaries have been drawn so as not to allow disposal of initial box-cut material adjacent to it.
- Thick coal seams thin out towards the sub-outcrop without becoming shallower - resulting in high stripping ratios for the first several strips.
- The quality of the coal nearer the sub-outcrop may be poorer – hence the yield falls. The stripping ratio per ton ROM would still be favourable, but the ratio in terms of ton coal sales may be high (or profitability low).
- A multiseam dragline mine often must commence at the sub-outcrop of the lower seam — which may be at a higher ratio than the two seam situation several strips further on.
- The increasing concern about leaving a huge final void at the completion of the mine life may require a different approach to mine design consistent with progressively more backfilling.
The above factors notwithstanding, there is seldom a more attractive alternative than commencing at the shallowest, lowest strip ratio position and progressing down dip. Strips should preferably be straight so that the refill shape can be designed to make maximum use of available spoil room, and the volume of void carried along in the operation is kept to a minimum.

BENCH LAYOUT AND WORKING FACE LENGTH

Undoubtedly the most important factor in determining production rates achievable from a surface strip coal mine is the available working area for equipment to operate. Furthermore, the more complex an operation, the greater the amount of equipment needed to operate in it, and the greater is the requirement for working area for each of these individual operations. Required working face lengths are therefore related to scheduling criteria and pit geometry.

However, a number of simplifications can be used to quickly assess the constraints. Normally, efficient equipment use requires continuous operations with as little “deadheading” as possible between one block or cut and the next (equipment walks from end of current cut to start of next). For dragline operations, a three month period is a desirable minimum time for continuous excavation without excessive lost time due to deadheading.

At 5 million ton ROM production per year, an area of 125 ha of coal is uncovered and mined each year for a 3m thick seam (using a coal density of 1,33t/m$^3$). For a 60 m wide pit, this represents a total yearly face length of 20km. Based on a 3 month strip cycle time, a site face length of at least (20x3/12) or approximately 5km is therefore required.

As in-pit operations become more complex (for instance, multiple seams or partings), the required working area must increase if production rates are to be maintained. This is readily achieved in a multiseam (shovel/truck) terrace operation by arranging the bench layout for simultaneous excavation of seams. In a dragline operation, only sequential excavation of seams within the pit can be undertaken and to exacerbate the situation, pit widths cannot normally be increased to provide more working room. High production rates in complex multiseam dragline operations are therefore only achievable if long cut lengths can be opened up. Typically at least 1.5 km to 2 km of cut length per million tons of ROM coal per annum is required in a complex mining situation. For a dragline operation at 5Mtpa, a minimum cut length of 7.5 km to 10 km is therefore required; or the same effect could be achieved in a
shovel/truck operation by opening up 3 benches (in 3 seams) over a total length of 2.5 km to 3 km.

PIT WIDTH AND COAL HAULING RAMP ACCESS
The pit (or block) width is usually determined from dragline size selection, based on the reach and operating radius required and is usually made in the planning stage with little or no leeway for change once mining starts. The variations in coal exposure rate as compared with the required coal mining rate is especially important in strip mining, where depths of overburden above the coal vary widely along the length of the cut, since coal exposure rate will vary and cut width variation (to balance exposure rates) is not a viable option.

A wide pit, of say 50m, is less congested and easier to operate as regards coal loading and hauling than a narrow pit of say 25m width. One of the primary considerations is an efficient coal hauling operation. For a single seam dragline operation haul trucks do not have to pass the stripping equipment in the cut. Other less important factors might be the possible location of sumps, dewatering pumps and catering for coal drilling and blasting operations.

On the low-wall side, the average distance between ramps is important, as is the presence or absence of access at the end of the cut. In general a distance of approximately 1000m –1500m between ramps is satisfactory. Ramps cause overburden storage loss and this loss is only recovered in about 1000m pit length without narrowing the cut width. Also it is usually found advantageous wherever possible to have access at one or both ends of the cut. This allows the dragline deadheading to be kept to a minimum, if no access is available at the end of the cut then the first lateral ramp should be close enough to the end of the cut to provide a not unreasonable deadheading distance. An alternative option, which as advantageous from the point of view of rehabilitation and improved pit spoiling room, is to access the coal seam using highwall ramps. Whilst these are more disruptive to the dragline stripping operations, and require rebuilding at every strip turnover, their advantages may be relevant in certain conditions, often associated with multi-seam mining operations. Figures 3.10 and 3.11 illustrate the strip layouts associated with these ramp options.
Figure 3.10  Strip mine layout (in plan) with low wall (spoil side) access ramps to coal seam

Figure 3.11  Strip mine layout (in plan) with high wall access ramps to coal seam
3.3.2 Strip Mining Over Mined-out Areas

Since the start of coal mining in South Africa in the eighteenth century, mining methods have progressed from handgot bord and pillar methods. The bord and pillar methods, unless further extracted by secondary means, cause pillars to be left in situ and in the case of thick seams, the additional remaining top and/or bottom coal represents a major loss of coal reserves and thus a substantial financial loss to the mining company and to the national reserves. One factor which has been common to all older underground mines is that due to economic reasons, the mining wherever possible has been restricted to shallow seams. This has resulted in partially extracted seams that are economic to mine using the typically high volume, low unit-cost method of surface mining. However, in doing this, certain additional mining method-related considerations arise;

- Stability of the underground workings
- Stability of the highwall and the direction of mining
- Blasting techniques to preserve or collapse the workings

STABILITY OF THE UNDERGROUND WORKINGS

The interaction between strip mine workings and underground workings will largely be governed by the stability of the underground workings. Factors to consider are;

- the position and size of the pillars
- mining heights
- the condition of the roof and floor
- the floor elevation of the workings
- the extent of rock falls, sudden changes in the elevation of the workings, and significant slips.

This data can be compiled into a hazard plan to indicate areas of underground working that present a subsidence hazard to typically (but no exclusively) the dragline.


**STABILITY OF THE HIGHWALL AND DIRECTION OF MINING.**

The strip mining direction has a large influence on stability, as its choice can cause particular joint sets, shear planes or faults to be favourably or unfavourable orientated in terms of slope stability.

The principle geotechnical considerations in suggesting a particular mining direction is the effect of underground workings. If the strip is oriented parallel to the workings, then the highwall will sometimes intersect bords with no pillar support. If the highwall runs at an angle to the workings, there will always be some pillars at the face. In practice, it is usually recommended that the strip direction should be at an angle of 30° to the direction of the underground mining.

**BLASTING TECHNIQUES TO PRESERVE OR COLLAPSE THE WORKINGS**

Overburden blasting techniques may be classified as either;

- Blasting to collapse the bords
- Blasting to collapse the pillars
- Blasting to preserve the workings

In each case, the extent of mined-out coal and remaining top- and bottom-coal, together with the mining method and equipment and production-related constraints will dictate which methods is most suitable.

Collapsing the bords during overburden blasting will prevent subsidence from taking place underneath the dragline, but will result in dilution of the coal. This method will work only if all the bords collapse during blasting when the dead load of the blasted overburden should cause any top coal over the bords to fail. In addition, toe damage at the bottom of the blast holes and cratering should also assist in the collapse of the top coal into the workings. The sequence of mining operations is shown schematically in Figure 3.12. Dilution, calculated according to the dimensions given in Figure 3.13, is given by;

\[
Dilution (D) = \frac{\text{volume waste}}{\text{volume waste} + \text{volume coal}}
\]
Figure 3.12 Strip mining method with bord collapse (after Morris, 1983) showing (1) overburden drilling, (2) dragline stripping, (3) dozer cleaning to top of coal, (4) drilling pillars and (5) ripping bottom coal.

Figure 3.13 Bord collapse in previously mined seam and definition of symbols for calculation of dilution.

where volumes are expressed as broken rock, or loose cubic meters.

\[ D_t = \frac{E_x(H - T_c S_w)}{H(1 + S_w) - E_x S_w(T_c + H)} \]

where;
Collapsing the pillars during overburden blasting would result in a subsidence free bench for the dragline operation. However, it would be necessary to accurately survey the location of the pillars to enable overburden drilling to intersect the pillars. This method of blasting the pillars would produce a large amount of fines, which may be undesirable, together with some dilution due to variations in overburden strength in the vicinity of the pillars. The sequence of operations is shown in Figure 3.14.

The main purpose of attempting to preserve the workings would be to minimise waste dilution of coal. A calculation of roof beam thickness necessary to support the dead weight of blasted overburden would be necessary. This beam could then be ripped once the overburden is removed – but not without considerable risk of collapse of the top-coal and damage to equipment. Alternatively, the beam could be removed by hydraulic excavator separately from the coal as shown in Figure 3.15.
Figure 3.15 Strip mining method with workings preserved (after Morris, 1983) showing (1) overburden drilling to top of roof beam, (2) dragline stripping, (3) dozer ripping beam (4) drill, blast and loading top, pillar and bottom coal

3.4 Terrace Mining

A dragline is the least expensive method to strip overburden where simple side casting is used. However, deep overburden, dipping seams, faulted seams, multiple seams and thin interburden make dragline strip mining procedures very complex, and in any of these situations it is questionable whether dragline stripping is the best mining method.

In strip mining, the dragline excavates overburden and dumps it as waste directly over the mined-out area. In deeper or more geologically complex deposits, other mining equipment such as shovels, bucket wheel excavators, scrapers, and intermediate cyclic or continuous transport (e.g. trucks or conveyors) systems are used to transport the overburden to where it can be tipped back into the previously mined void. Such equipment is best used where benches or terraces must be constructed to reach a deeper or thicker, more complex coal seam. Figure 3.16 shows a typical truck and shovel terrace mining method in which three overburden benches are mined to expose a thick coal seam, divided into top, middle and lower seam products. The overburden benches are level and trucks haul the spoil around the pit to be back-dumped. In this way, the pit is continuously backfilled and reclaimed as the mine progresses.
The coal is accessed by removing overburden from the top down in a series of horizontal layers of various thickness called benches. Mining starts with the top bench and after a sufficient floor area has been exposed, mining of the next bench can begin. The process continues until the bottom bench elevation is reached and the coal removed. From that point onwards, the benches advance laterally, thus it becomes a multi-benched sideways-moving method, the whole mine moves over the coal reserve from one end to the other, but not necessarily in a single bench or single panel. The number of benches used is usually a function of the excavation depth and type of machinery used (typically between 10-15m bench height and 2-10 benches in the terrace).

**TERRACE MINING GEOMETRY AND LAYOUT**

If the terrain is flat and the coal seam has practically no dip, a long and narrow, rectangular deposit can be mined in two panels with one turnaround. The turnaround allows a better face length and places the second panel’s final void close to the first panel’s box-cut stockpile. The final void can then be completely backfilled with the box-cut material. In a rectangular mine, the faces should be advanced in parallel. For example, if the mine needs to produce 10 million tons of coal per year for 20 years with a 14,5m total seam
thickness (or 18.1t/m² coal exposure), a 1100 hectare property is needed. If the property is two sections wide and eight long, the above two-panel technique can be used, with a panel width of 830m. But if it is four sections square, four panels may be necessary to keep an optimal face length and still place the final void near the original box-cut stockpile.

Determining the optimum face length is difficult because it depends on many variables and often property boundaries do not allow the use of the optimum face length, so a compromise is necessary. A integer number of panels must be planned to fit into the property, and an even number of panels is preferred because it positions the final voids near the original box-cut stockpile. In a case of an irregular property boundary that intersects the faces, a variable face length could be used to uncover all the coal. However, once mining is initiated, a significant deviation from the planned face length could require an extensive redesign of the mine.

In simplistic terms, two options exist between the optimum, too long or too short face lengths;

- Long face lengths are not desirable because a large fleet of excavators would be required. However, a high production rate from a single pit may dictate the use of a long face and in such cases, coal and overburden haulage through the middle of the pit may be desirable.

- On the other hand, a short face would dictate a series of narrow panels. This is not desirable because the length of coal haulage roads would be continually fluctuating. Short faces require wide benches and a longer pit to maintain production levels.

When a panel has advanced to the end of the property, the mine must execute a 180° turn into the next new panel to be opened. These turning methods are quite complicated to execute and will require constant pit monitoring. The easiest approach is to turn in two 90° maneuvers. When the mine has advanced to the end of the panel, the shovels are turned 90° and begin excavating the next panel. This short turning panel is advanced to the full width of the new panel while backfilling the void in the first panel; then the excavators are turned 90° again and begin retreating that panel.

When two or more panels are required for the mine property, it is beneficial not to backfill a portion of the pit immediately adjacent to the next panel. This creates an open area, called a slot and it obviates the need to rehandle the previous panel's spoil to expose
the coal. Two-panel property requires one slot whereas four-panel property requires three slots.

**BOX CUT ORIENTATION, PANEL AND WORKING FACE LENGTH**

If the coal seam is lying horizontally or dips slightly (up to 10°), advance can be essentially in any direction as long as the terrain is flat. This condition is preferred in terrace mining because coal production will remain uniform. However, geotechnical or other factors may dictate the direction of advance. In the case of slightly dipping seams, if the coal does not outcrop within the mine property boundary, an area should be chosen with the least amount of box-cut spoil to initiate mining. In cases of rising topography and horizontal coal seams, it is usually preferred to advance perpendicular to the rising terrain if stability of the material permits, because of the reduced stripping ratio in the first panel. If the mine property is rectangular but the seam dips close to the maximum of about 10°, it is generally recommended that the mine advances along the strike. However, the mine can also proceed along the dip. The decision partly depends on how the property lies to the dip, and if it is long and narrow or nearly square.

As seen with the case of strip mining, the economics play a major role in deciding whether to advance down-dip or along the strike. For a mine situation where the length of the mine property occurs along the strike, advancing the first panel along the strike is very economic for the first half of the mine life because of its low stripping ratio. But, there is a great danger that the coal will not be mined on the second panel because of its higher stripping ratio. Figure 3.17 shows the relationship between the stripping ratio and a mine life of 20 years. During the first 10 years the first panel's stripping ratio average remains uniform; then as the mine turns to mine the second panel, the stripping ratio jumps up dramatically to a level to be maintained for the last 10 years. If a slot (of sterilised coal reserves) is not left between the two panels, a portion of the first panel must be rehandled in mining the second panel. This raises the second panel’s average stripping ratio.
Figure 3.17 Change in stripping ratio over the life of a terrace mining operation, with dipping coal seam sub-outcrop on the panel side

If this same property is mined along the dip, four panels will be necessary. Figure 3.17 shows how the stripping ratio will fluctuate over the life of the mine. The first panel will proceed down-dip, the second panel up-dip, the third down-dip, and the fourth up-dip. If slots are not left between adjacent panels, the average stripping ratio will be the same as for the first panel. If slots are left, the last three panels will have a reduced stripping ratio. When mining down-dip, problems result because there is less room to spoil the overburden into the previous cut, but when retreating up-dip, the trucks may encounter haulage problems, the waste is more stable, the ground water is drained from coal and overburden and there is more spoiling volume available.

Generally, a terrace mine can operate on a seam dip of up to 10° either along the strike or along the dip. When the coal seam dips in excess of 10°, the pit floor becomes too steep for truck haulage. Under these circumstances, terrace mining can still proceed by mining neither along the strike nor along the dip, but at an angle oblique to both. This tends to reduce the dip to a somewhat smaller angle.
TERRACE BENCH DESIGN
The basic extraction component in a terrace mine is the bench. Bench nomenclature is shown in Figure 3.18. Each bench has an upper and lower surface separated by a distance \( H \) equal to the bench height. The exposed subvertical surfaces are called the bench faces, described by the toe, the crest and the face angle. The bench face angle can vary considerably with rock characteristics, face orientation and blasting practices. Normally bench faces are mined as steeply as possible. The exposed bench lower surface is called the bench floor. The bench width is the distance between the crest and the toe measured along the upper surface. The bank width is the horizontal projection of the bench face.

There are several types of benches. A working bench is one that is in the process of being mined and the width being extracted from the working bench is called the cut (of width \( C \) in bench width \( W_B \) to leave an access of width \( S_B \)). In the extraction of a cut, the drills operate on the upper bench surface and the loaders and trucks work on the bench floor level.

A number of different factors influence the selection of bench dimensions. In the absence of any other mining, geological or selectivity constraints, a general guideline is that the bench height should be matched to the loading equipment. Table 3.2 shows a general rule of thumb, based on the concept that the bench height should not be greater than that of the sheave wheel.

Table 3.2 Approximate bench height guidelines based solely on loading equipment match

<table>
<thead>
<tr>
<th>Bucket size ( m^3 )</th>
<th>Bench Height (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>(&lt;5.0)</td>
<td>9</td>
</tr>
<tr>
<td>(5.1 – 8.0)</td>
<td>12</td>
</tr>
<tr>
<td>(8.1 – 20.0)</td>
<td>14</td>
</tr>
<tr>
<td>(20.1 – 30.0)</td>
<td>16</td>
</tr>
<tr>
<td>(&gt; 30.0)</td>
<td>18</td>
</tr>
</tbody>
</table>

Operating in benches with heights greater than this sometimes result in overhangs which endanger the loading and other operations. At one time, bench heights were limited by drilling depth, but modern drills have largely removed such restrictions. However, in large terrace mines, at least, it is desirable to drill the holes in one pass. This means that the drill must have a mast height.
sufficient to accommodate the bench height plus the required subdrill.

Figure 3.18 Basic bench nomenclature in terrace mining

Higher and wider benches yield:

- less selectivity (mixing of high and low grade seams or overburden of different types);
- more dilution (mixing of waste and coal);
- fewer working places hence less flexibility;
- flatter working slopes; large machines require significant working space to operate efficiently.

On the other hand, such benches provide:

- fewer equipment setups, thus a lower proportion of fixed set up time;
- improved supervision possibilities;
- higher mining momentum; larger blasts mean that more material can be handled at a given time;
- efficiencies and high productivities associated with larger machines.

The steps which are followed when considering bench geometry are:

1. Coal deposit characteristics (total tonnage, qualities, seams, interburden distribution, etc.) dictate a certain geometrical approach and production strategy in the coal itself.

2. The production strategy yields daily coal and overburden production rates, selective mining and blending requirements, numbers of working places.

3. The production requirements lead to a certain equipment set (fleet type and size).

4. Each piece of equipment in the set has an associated operating geometry.

5. A range of suitable bench geometries results.

6. Consequences regarding stripping ratios, operating vs. capital costs, slope stability aspects, etc. are evaluated.

7. The ‘best’ of the various alternatives is selected.

Today, highly mobile rubber tyred/crawler mounted equipment has reduced the detailed evaluation requirements somewhat.

Terrace heights will inevitably vary in a multiple-seam mine because the interburden, overburden, and coal seam thicknesses change and depending on the various coal qualities encountered through
the full thickness of the seam, it is often preferable to mine each
seam independently and to avoid dilution with interburden. The
slope of the land or the dip of the coal seam causes the terrace
height to change as the mine progresses and significant changes
will require the addition or deletion of terraces and their side-pit
haulroads. Generally, where overburden benches are concerned
and there are no selectivity issues in equipment application, the
bench height should be designed to be as near optimal for the
loading equipment as possible. The geological and physical
characteristics of the various overburden strata encountered may
require different equipment combinations or separate terraces. For
example, unconsolidated weathered overburden and easily
fragmented rock may be most economically excavated by free
digging with hydraulic excavators, while other material might be best
excavated by a drill and blast and rope shovel - or front-end loader –
truck combination.

Any unnecessary bench width will increase haulage distance and
costs. Therefore, all terraces, including the exposed coal seam,
should be kept to a minimum width. The benches should be around
30 to 50 meters wide to permit equipment passing and turnaround,
and to minimize congestion in the loading area. Generally, their
width and haulroad width should be at least three times the
maximum vehicle width, thereby enabling two haulage vehicles to
pass maintenance equipment without interrupting production.
However, due consideration should also be given to the production
rate or the excavator size which may also be a determining factor in
the selection of the final bench width.

COAL AND OVERBURDEN HAULAGE
Two options may be considered;

- If the face is short, a side panel coal haulroad is preferred
  because this layout allows higher average truck speeds and
  lower costs. In this situation the slot between panels can be
  used for coal haulage, and it can also become the coal haulroad
  in the next retreating panel.

- When the face length is long, it is preferable to have mid-panel
  roads where the coal haulroad runs through the middle of the pit
  because overburden and coal haulage distance become
  excessive. In this case, a slot (in addition to the middle-pit
  haulroad) may not be feasible.
Generally, coal haulage through the side slot is preferred because coal can be hauled up a shallow incline to the surface, rather than up steep in-pit haulroads.

Overburden is hauled to one side of the pit only to avoid interference with the coal haulroad; for example, when a side-pit coal haulage slot is used. Overburden haulage on the same side as coal haulage would require steep ramps down into the pit and back up to the spoil terraces. Overburden haulage from one side of the pit, opposite coal haulage, is preferred because level roads can be maintained. If mid-panel coal haulage is used and no slot is left, coal haulage can be along both pit sides and through the middle.

3.5 Overburden and Waste Dumping

In addition to the mine design requirements for excavation below the surface (most often overburden removal to expose the coal), there are design requirements for dumping material thus created on the surface. For the disposal of mine overburden, and to a lesser extent coal preparation waste, there are a number of different types of mine waste dump design and embankment construction required. These may typically include the following categories:

- Embankments surrounding slimes ponds associated with the coal preparation process and eventually the settled slimes themselves.

- Dry rock and coal wastes extracted in the coal preparation process not requiring impoundment.

- Overburden waste produced primarily from operations using predominantly truck and shovel stripping techniques.

In the majority of surface coal mines using draglines or bucket wheel excavators, etc., separate overburden waste dumps are not necessary, other than for top-soil and sub-soil storage. At these properties coal preparation, if performed, produces the major waste disposal problem in the form of coal rejects which is quite often highly acidic. Many companies conveniently dispose of this material by using it to backfill mined out areas. Surfacing materials suitable for revegetation are then overlaid to complete the backfill process to final grade.
3.5.1 Design and Construction of Dumps

Waste dump design involves consideration of the following factors;

- The volume of material to be dumped and the in-pit location of these volumes
- The extent of surface area available for waste dumping
- The strength parameters of both the waste and proposed foundation, or ground surface onto which the waste is tipped
- The drainage of the ground under and around the dump, and the ground surface profile where dumping is to take place.

The proposed dump location will be selected considering both the volume of material and the haul distance involved. Since the major cost penalty in waste transport is related more to load elevation than distance hauled, it is important to locate the dump as close to the excavation as possible and ideally, maximise the lateral extent of the dump. This ideal is not always possible however, since a large flat dump will require a large available surface area and increase the size of a mining property considerably. It is also necessary to determine the optimised design of the dump, in terms of haulage costs and efficiency since, for a large flat dump, truck travel times may eventually exceed those of a shorter dump ramp, albeit at a higher ramp grade.

Therefore, most waste rock dumps are constructed in a series of lifts by end-tipping (tipping the waste rock over the end of the dump face), each successive lift or dump level being placed on the previous layer, the height of the lift being determined by dump extent and width, such that the ramp constructed upwards to the next lift is maintained at an optimum grade of 6-8%. Waste material from the pit may be routed to any of a number of active waste dumps, the principle decision being based on where the waste is loaded and dumped, such that transport distance and load elevation is balanced for each bench level. This implies that waste loaded from benches deeper in the mine should be tipped on a dump level close to the original surface, whereas waste loaded from benches nearer the surface can be tipped on higher dumps.

Some important factors that should be observed during waste dump construction include;

- The removal of soil, for rehabilitation of the dump, to a depth specified by the Mines Inspectorate, together with the additional
necessity to remove weak foundation material below topsoil horizon if so prescribed in the design (moreso when a dump is built on a downslope)

- Major drainage courses should be avoided, or re-routed if possible, as detrimental effects on pore water pressures may result influencing the dump stability. Surface runoff should be routed clear of the dump area

- Stability analysis must be performed to prevent failure both during and after construction. In general, as the height of the dump increases the overall slope must be decreased by the introduction of berms. Special consideration needs to be given to the location of weak, weathered overburden when tipped on a dump as it will cause instabilities.

In the case of a discard dump at a surface coal mine, a number of design requirements additional to those of waste rock dumps apply. The discard dump results from the disposal of carboniferous residues, namely the solid waste products of the beneficiation plant, and/or waste rock from the roof or floor of the mine or from shale bands within or between the coal seams. The finer fractions are usually transported by water, and the resultant slurry is disposed of in containment or settling ponds, impounded by purpose-built embankments.

The shape of a discard dump is influenced by a combination of slopes (gradients and lengths) and, in turn, affects the stability of the dump, the flow of surface water (hydrological and hydraulic considerations), the sub-surface water, and the acid mine drainage.

Closure stipulations generally relate to the control of the stability of dumps and associated pollution-control structures for a minimum of twenty years. This is in addition to a stipulation in regard to the prevention of the excessive scour and erosion caused by storms of a 1:100 year intensity. Although the prediction of, and design for, hydrological effects are fairly straightforward, the problem is more the run-off storm water and the resultant hydraulic effects. Slope gradients should be limited to 1:5 (vertical) and 1:200 (lateral), while slope dimensions of 25 to 30m down the slope and 500m along the contour berms are typical. Depending on the intensity of the storms and erosivity of the soils in the area, a slope of 1:7 or less should obviate the need for run-off control.

Both to reduce the risk of spontaneous combustion and to increase the stability of the dump, the discard material should be effectively
compacted. In contrast to end-tipping of overburden waste rock, the material should be dumped and spread in uniform layers 0.3m thick, and the layer then compacted. After the required number of lifts, slope and surface blankets of inert sealing material should be used to isolate the dump material. The capping material should be well compacted and the dump contours streamlined.

SPONTANEOUS COMBUSTION OF DUMPS

Spontaneous combustion is the burning of coal due to self heating. All coals oxidise to various extents when exposed to the atmosphere or when air is introduced to the coal. Since oxidation is exothermic, heat is released and the continued heat release, if not dissipated, will cause the temperature of the coal mass to increase. Continued oxidation will eventually raise this temperature to a point where the coal combusts, or burns. Oxidation usually proceeds very slowly at ambient temperatures but increases rapidly and progressively as the temperature rises. Neither oxidation, spontaneous combustion, nor burning, will occur in the absence of air. For burning to occur with visible flame it is normally necessary for the material to be combustible, to reach its specific ignition temperature, and for sufficient oxygen to be supplied from the air.

Spontaneous combustion applies equally to coal exposed during the mining process in the pit itself and to coal placed in waste dumps. In addition to the coal-related factors themselves, Table 3.3 lists the risk and contributory factors likely to influence how and under what circumstances spontaneous combustion is likely to arise.

Of the coal factors, five major reasons for the spontaneous combustion potential of coal are;

- **Rank** Generally the lower rank coals are more reactive and hence more susceptible to self heating than higher rank coals. Rank is generally regarded as a fairly reliable indicator of a coal's relative propensity towards weathering and also towards spontaneous combustion. The lower the rank, the more liable the coal will be to oxidation. The maximum inherent moisture content of coals has been used as an indicator of their liability towards spontaneous combustion.
Table 3.3 Contributory factors to spontaneous combustion of coal in dumps or during the mining process (in-pit)

<table>
<thead>
<tr>
<th>Risk factor</th>
<th>Contributory</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Coal factors (in-pit and dumps)</strong></td>
<td></td>
</tr>
<tr>
<td>Reactivity</td>
<td>Calorific value</td>
</tr>
<tr>
<td>Density</td>
<td>Ash content</td>
</tr>
<tr>
<td>Volatile matter</td>
<td>Inherent moisture</td>
</tr>
<tr>
<td>Pyrite Content</td>
<td>Total sulphur</td>
</tr>
<tr>
<td>Rank, fixed carbon</td>
<td>Friability</td>
</tr>
<tr>
<td>Porosity</td>
<td></td>
</tr>
<tr>
<td><strong>Geological factors (in-pit)</strong></td>
<td></td>
</tr>
<tr>
<td>Dykes and sills</td>
<td>Depth</td>
</tr>
<tr>
<td>Floor rolls</td>
<td>Parting</td>
</tr>
<tr>
<td>Overlying strata</td>
<td>Seams in proximity</td>
</tr>
<tr>
<td>Faults</td>
<td></td>
</tr>
<tr>
<td><strong>Environmental factors (dumps)</strong></td>
<td></td>
</tr>
<tr>
<td>Air moisture</td>
<td>Annual temperature and rainfall distribution</td>
</tr>
<tr>
<td>Conditions while dumping</td>
<td>Wind speed and direction</td>
</tr>
<tr>
<td>Pressure fluctuation</td>
<td>Groundwater inflow</td>
</tr>
<tr>
<td><strong>Environmental factors (in-pit)</strong></td>
<td></td>
</tr>
<tr>
<td>Gas emission rate</td>
<td>Source of hot spots, springs, adjoining fires</td>
</tr>
<tr>
<td>Water table</td>
<td>Annual temperature and rainfall distribution</td>
</tr>
<tr>
<td>Subsidence</td>
<td>Shallow abandoned workings</td>
</tr>
<tr>
<td><strong>Mining factors (in-pit)</strong></td>
<td></td>
</tr>
<tr>
<td>Working method</td>
<td>Ratio of mining thickness to seam thickness</td>
</tr>
<tr>
<td>Rock bumps</td>
<td>Accumulation of fines</td>
</tr>
<tr>
<td>Rate of advance</td>
<td>Nature of extraction</td>
</tr>
<tr>
<td>Humidity</td>
<td>Ventilation</td>
</tr>
<tr>
<td><strong>Mining factors (dump)</strong></td>
<td></td>
</tr>
<tr>
<td>Stacking or dumping method</td>
<td>Segregation</td>
</tr>
<tr>
<td>Compaction</td>
<td>Particle size</td>
</tr>
<tr>
<td>Slope angle</td>
<td>Height of dump</td>
</tr>
<tr>
<td>Sealing method</td>
<td></td>
</tr>
</tbody>
</table>
Moisture  The absorption of moisture by coal is an exothermic reaction. The amount of heat liberated is, however, a function of the initial moisture of the coal. Dry coal, for example, will produce more heat when rewetted than coal that already contains some moisture. Coal, once dried, will re-absorb water when it comes into contact with either humid air or a source of water. Rewetting of coal generates heat and this can contribute to the build-up of heat within the coal. Unless such heat is dissipated, it may contribute to increased oxidation and spontaneous combustion of the coal. It has often been observed that coal may have been stored without problems through the dry winter months, only to combust at the first rains.

Temperature  At low temperatures an increase in moisture increases the rate of spontaneous heating. The presence of free moisture is essential for the oxidation of iron pyrite.

Pyrite content  The presence of iron pyrite can also have a significant effect. Pyrite will oxidize at normal temperatures with moist air producing ferrous sulphate, sulphuric acid and heat thus increasing the tendency of spontaneous combustion if present in coal piles. Hydrogen sulphide may also be produced if heat and limited air is present which offers one way of detecting trouble. Pyrites are also believed to promote the breaking-up of coal through the formation of ‘higher volume’ oxidation products.

Void ratio and surface area  Coals differ in their propensity to absorb oxygen and this usually varies with the rank of the coal. Anthracite, on the high end of the rank scale, is not known to heat spontaneously, whereas those coals with a high inherent moisture content, a high oxygen content and a high volatile matter content, i.e. low rank coals, are known to be problematic in this regard. The voids ratio controls the flow of air through the coal dump. If large pieces of coal predominate, air flow through the pile is good and heat is dissipated and does not build up. A small voids ratio allows oxidation to occur but soon stops as air is exhausted because of small or no connections between voids. Intermediate sized voids are the worst condition when oxygen becomes readily available but heat is not dissipated. The rate of oxidation is directly proportional to the specific surface area of coal, that is inversely proportional to the size of the coal particles.
3.5.2 Design and Construction of Embankments

The primary use for embankments at a coal mine is the containment of processing plant rejects in the form of slurried fines in preselected areas and the temporary storage and clarification of process water. The size of the pond so produced depends on the coal production, associated slimes production and the rate of slimes settling in the pond. Another use for embankments at special locations is for the containment of overburden material which cannot be dumped by itself due to an extremely low angle of repose.

Embankment design must first consider the topography of the selected site along with the estimated rate of slimes input. The total storage requirement is found followed by an estimate of the final crest elevation and corresponding ultimate height. Stability analysis must then be carried out taking into account the shear strength parameters and the densities of the materials in both the foundation and the embankment, as well as the pore water pressures within the embankment and the foundations. Seepage pressures can be estimated using flow net analysis.

The embankment cross section will result from the design analysis and may include intermediate berms for the larger banks. For equipment operation, the width of the crest of the embankment should not be less than 4m. Generally the crest width should be 4m plus one-fifth of the maximum height of the embankment.

Embankments are usually designed to retain water and for this reason the site is usually selected where suitable foundation material is present. Thus excavation of weak materials, sealing or grouting the foundation is not usually necessary. Following removal of topsoil as required or stipulated, scarifying and compacting soil foundation surfaces is sometimes required as is special compaction of the initial layers of the fill. Compaction is used to increase material density as placed, increase fill shear strength and decrease fill permeability. The fill itself will also generally be compacted, the procedure depending on the layer thickness, type of compactor and water content.

3.6 Surface Mine Service Infrastructure

Services refer to the provision of infrastructure to supply, generally speaking, consumables and for the removal of the by-products of the mining method. Service infrastructure can cover such items as electricity supply, diesel supply, explosives delivery systems, in-pit equipment consumables (lubricants, ground engaging tools, trailing
cables, etc.) and water reticulation, either as service water for the mining method, or more commonly as groundwater handling. In this section, only electrical reticulation and groundwater handling will be covered in detail.

### 3.6.1 Electrical Reticulation Systems

Most large strip coal mining equipment is usually mobile and self-propelled and most are powered electrically through portable cables. However, this generalisation is not necessarily applicable to all coal mining methods and in fact, under certain circumstances, a decision can be made in the planning stage of a mine to adopt diesel-only driven equipment. This latter option is, however, rarely considered where large (electric) draglines are used, and is suited only to smaller pits where truck and shovel mining is exclusively used, or where the utility company cannot establish supply at a reasonable cost.

Mine power systems generally have two voltage levels, distribution and utilisation. Power supply to the mine is at transmission levels, from which point it is transformed (reduced) at the mine substation down to distribution levels. Power at this voltage is distributed through conductors from the substation to the power center; hence it is termed the distribution or reticulation system. The power center or load center, in actuality a portable substation, transforms the voltage to utilisation levels.

#### POWER TERMINOLOGY

Several power terms are used to describe the operation a power system. These terms are applicable not only in system design and operation, but also in electricity supply billing. If the sum of the electrical ratings is made for all equipment in an electrical operation, the result will provide a total connected load. The measure could be expressed as kilowatts (kW), kilovolt-amperes (kVA), or amperes (A). However, many loads operate intermittently, especially mining production equipment, with varying load conditions (typically dragline or production shovels with full or empty buckets). Accordingly, the demand upon the power source is frequently less than the connected load. This fact is important in the design of any mine power system, as the system should be designed for what the connected load actually consumes rather than the total connected load. Obviously, these considerations have great impact on the capital required to build the power system. The important definitions in respect of reticulation system design are;
- Demand - the electrical load for an entire mine or a single piece of equipment averaged over a specified time interval, generally expressed in kilowatts, kilovolt-amperes, or amperes.

- Peak load - the maximum load consumed by one piece or a group of equipment in a stated time period. It can be the maximum instantaneous load, the maximum average load, or (loosely) the maximum connected load over the time period.

- Maximum demand - the largest demand that has occurred during a specified time period.

- Demand factor - the ratio of the maximum demand to the total connected load.

- Diversity factor - the ratio of the sum of the individual maximum demands for each system part or subdivision to the complete-system maximum demand.

- Load factor - the ratio of the average load to the peak load, both occurring in the same designated time period. This can be implied also to be equal to the ratio of actual power consumed to total connected load in the same time period.

**POWER DISTRIBUTION EQUIPMENT**

The evolution of mine systems has resulted in major items of power apparatus, each serving a specific function;

- Mine power centers. The power or load center is an essential unit for surface mines. Its primary function is to convert the distribution voltage to utilisation voltage for operating equipment throughout the mine.

- Switchhouses. Switchhouses are portable equipment that protect and provide a means for branching of the distribution circuit. As with power centers, the principal function of the disconnect switch is to remove power manually from downstream mine power equipment that is connected to the distribution system. The sectionalising aspect is to provide protective relaying in the distribution system and to allow branching.

- Main substations. A main (primary) substation is required to transform incoming levels down to a primary distribution voltage for the mine. Main substations are usually permanent installations with components mounted on concrete pads, the nature of the mining operation and its power requirements.
dictate how many main substations are required and where they should be placed.

- Portable and unit substations. These units are used for further transformation and branching of the supply, often for specific pieces of equipment.

- Distribution (cabling and connectors). This category of major power equipment is often referred to as the weakest link in mine power systems. It encompasses all the overhead power lines, cables, cable couplers and trolley lines used to carry power and grounding between the power equipment and eventually to the loads. In surface mines, the cables that feed from the switchhouses or unit substations to mobile equipment are trailing cables. Those moved only occasionally that are not connected directly to a machine are portable cables. Stationary cables can be feeder or portable types. All cable, higher voltage distribution systems are becoming more common, compared to the overhead pole-line systems. The higher voltage allows increased line length, is necessary for the large machines and allows cost savings for high loads. The all cable system also offers reduced costs and is more flexible than the older pole-line plus cable distribution system, but often, between switch houses, pole-lines are easier to maintain (less easily damaged than cables), but are more difficult to relocate.

**RETICULATION SYSTEMS**

The primary purpose of any distribution scheme in a surface mine is to provide a flexible, easily moved or modified power source for the highly mobile mining equipment. System designs must also be considered as an integral part of the total mine operation. The distribution system that serves portable equipment is subject to damage from the mining machinery itself, and as a result, the system must be designed with optimum flexibility and consideration for personnel safety.

Three systems most commonly encountered in strip mining are;

1. Simple power-line and -cable radial reticulation. Radial reticulation in its simplest form consists of a single power source and substation supplying all equipment. Radial systems are the least expensive to install as there is no duplication of equipment, and they can be expanded easily by extending the primary feeders. A prime disadvantage is tied to their simplicity; should a primary component fail or need service, the entire system is down. Figure 3.19 illustrates such a system.
2. Secondary-selective power-line and -cable reticulation. In a secondary-selective system, a pair of substation secondaries are connected through a circuit breaker. The arrangement allows greater reliability and flexibility than a radial system. Generally, the reticulation is radial from either substation. If a primary feeder or substation fails, the bad circuit can be removed from service. Maintenance and repair of either primary circuit is possible without creating a power outage by shedding non-essential loads for the period of reduced capacity operation. Figure 3.20 illustrates such a system.

3. Dual baseline power-line and -cable radial reticulation. Similar to a simple radial reticulation system, two baselines are established on the highwall for two distribution voltages. Here, a large unit substation interconnects the two baselines. Figure 3.21 illustrates such a system.

In all configurations, a portion of the primary reticulation is established as a baseline or bus. The baseline is usually located on the highwall, paralleling the pit for the entire length of the cut. Its location is typically maintained about 500m ahead of the pit, and it is moved as the pit advances. Lateral feeder cables are then fed at regular intervals at right angles from the baseline, and where
necessary, over the highwall into the pit itself. Additional feeders from the spoil pile side are sometimes included if for instance a separate dragline is being used in the pull back method. As the machines move along the pit, the baseline connections are changed to other convenient locations.

Figure 3.20  Typical secondary-selective electrical reticulation system

The systems described above are typical of normal production systems, but where long equipment moves are required, power requirements should be carefully planned. This is especially true for moves of several miles between mines. Apart from the careful preparation of the route as regards walking surface, the power supply must be maintained at appropriate points throughout the entire length of the move. Mobile cable carriers are almost mandatory for this type of operation, their ability to pick up and then lay out complete lengths of cable becomes invaluable.
3.6.2 Water Reticulation Systems

This section will deal with drainage, the dewatering of pits and the prevention of water entering pits. Good drainage is essential for all surface mining operations. The main problems which develop through water accumulating in the pit are:

- difficulty in coal handling
- increased explosives cost
- possible waste slope instability
- reduced operating life of machinery, tyre cuts, cable damage, etc.
- nuisance factor
- possible floor heave
- generally unsafe working conditions
Water sources are part of the hydrological cycle and are derived from various sources as shown in Figure 3.22. In the Witbank coalfields of the Mpumulanga region, roughly 90% of the water encountered in operations originates from the surface either as rainfall directly into the mine workings or as surface run-off from surrounding areas. The remaining 10% is made up of groundwater seepage from surrounding aquifers into the mine workings. Table 3.4 lists the average water-recharge characteristics for mines in this region.

![Hydrological cycle in a surface strip coal mine](after Cogho and Hugo, 1996)

**Table 3.4 Average water recharge for surface strip coal mines**

<table>
<thead>
<tr>
<th>Sources contributing water</th>
<th>Average values</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rain onto ramps and voids</td>
<td>70% of annual rainfall</td>
</tr>
<tr>
<td>Rain onto unrehabilitated spoils</td>
<td>60% of annual rainfall</td>
</tr>
<tr>
<td>Rain onto levelled spoils (run-off)</td>
<td>5% of annual rainfall</td>
</tr>
<tr>
<td>Rain into levelled spoils (seepage)</td>
<td>20% of annual rainfall</td>
</tr>
<tr>
<td>Rain onto rehabilitated spoils (run-off)</td>
<td>10% of annual rainfall</td>
</tr>
<tr>
<td>Rain into rehabilitated spoils (seepage)</td>
<td>8% of annual rainfall</td>
</tr>
<tr>
<td>Surface run-off from surrounding areas</td>
<td>6% of total pit water</td>
</tr>
<tr>
<td>Groundwater seepage</td>
<td>10% of total pit water</td>
</tr>
</tbody>
</table>
Detailed initial studies of the hydrological cycle are essential prior to operation and form the basis of a water management plan, as described in a later Module. The main components of which should consider precipitation, evaporation, and drainage (run-off and catchment yield). These studies may even influence the overall mining sequence and will help to plan an efficient drainage operation.

- **Precipitation.** The main source of fresh water in an surface mining is often from precipitation. Extreme rainfall events can cause serious flooding problems in the mine, as well as in the storage facilities, where excess mine water is evaporated and recycled, and may eventually result in flooding or the failure of impoundment structures. Infiltrating precipitation is the main source of fresh-water recharge to the surrounding groundwater regime. This may vary between 1 and 5% of the annual rainfall for typical Witbank stratigraphy, but recharge values as high as 15% of the annual rainfall can occur naturally. Where the sub-stratum has been disturbed by mining operations, this value can increase substantially (up to 30 % of the annual rainfall), depending on the composition of the spoil material.

- **Evaporation.** Precipitation and evaporation needs to be considered when determining drainage requirements. The mean annual evaporation for the Witbank region is about 1700 mm which exceeds the annual precipitation by roughly 1000 mm per annum., although the daily precipitation can exceed the daily evaporation. This has to be allowed for in the management of surplus water on a mine.

- **Drainage and Run-off.** A knowledge of the run-off and drainage of certain areas can be used to recognize those parts of the landscape which are major contributors to either stormwater run-off or surface ponding. Large surface mines have significant impacts on the catchment yield and run-off volume, depending on the mine’s rehabilitation progress. The latter implies that significant volumes of ‘clean’ water have to be diverted around the mining area, making these operations extremely vulnerable to possible flooding, although storm-water-management structures may be in place.

To keep drainage costs to a minimum, the first consideration is the prevention of surface water from entering the pit workings. The main method of doing this, other than by the diversion of streams to new channels, is by the construction of surface ditches and levees. This is not complicated when coal lies above the natural drainage.
When water is collected in these ditches it is usually a straightforward process to direct the water into natural drainage channels. When the coal lies below the natural drainage then, although the collection is similar, levees must be constructed at discharge points to prevent water flowing back into the cut.

The in-pit drainage, whenever possible, should be developed to make use of gravity flow. Ditches should be provided both sides of haulage roads and in the pit itself where useful, often in the void between spoils and coal seam. On occasions it is possible to drain the complete cut out at one or both ends when end access is available. The use of culverts under roads and ditches in the pit, blasted if necessary, can provide the total mine dewatering system if grades etc, are correct. A useful alternative is the use of French drains under the ramps where they run into the cut. Unfortunately, these cases are more the exception than the rule and most properties with water problems must resort to pumping. Drainage sump locations are usually best sited near the bottom of inclines or at the end of the cut. Sumps can utilize existing hollows in the pit but it is usually more convenient to create a hollow at the desired location using a few sweeps of a dozer blade.

Pump selection will depend largely on the heads required and quantities of water to be pumped. The most common pumping equipment employed in sumps are submersible and centrifugal pumps. The latter are installed on floating platforms, or on the side of the sump where suction head is limited to 4 - 6m.

In deep pits the pumping head can be considerable, requiring thickwalled steel pipes and large pumps. To lower the head requirements intermediate stations are sometimes used. The water removed from the pit is delivered to settling ponds, usually via ramp roads or end-of-cut roads, strategically located around the perimeter of the mining property so that no water can escape to natural channels without passing through a pond. These ponds are often also used as a source of mine road water for dust suppression. The overflows from the ponds should be regularly checked for solids and chemical pollution. The number, location, and size of the settling ponds must be carefully considered at the outset of mining, using the water management plan as a basis for estimating volumes. Regular checks should be made to ensure that ponds do not silt up reducing the volume to a critical level. Periodically new ponds are required or old ones should be cleaned out.
## ROTARY DRILLING MACHINES

<table>
<thead>
<tr>
<th>Learning outcomes</th>
<th>Knowledge and understanding of</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>- Selection factors for rotary drills</td>
</tr>
<tr>
<td></td>
<td>- Rock strength and hole diameter relationship for machine selection</td>
</tr>
<tr>
<td></td>
<td>- Main components of the machine and their function</td>
</tr>
<tr>
<td></td>
<td>- Bit selection parameters</td>
</tr>
<tr>
<td></td>
<td>- Empirical approach to production potential of machine</td>
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<td></td>
<td>- Benefits of drill monitoring</td>
</tr>
<tr>
<td></td>
<td>- Total drilled cost per meter</td>
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**Apply, calculate or predict**
- Rock strength and hole diameter to select a machine
- Penetration rate of a drill
- Total drilled cost per meter to compare drill bits
- Number of drilling units required for given mining parameters

**Evaluate or design**
- Conditions for suitable drill selection
- Suitability of a particular machine for a surface mining application
4.1 Mechanical Excavation or Drill and Blast

The first of the mining sub-system unit operations conducted during the exploitation phase of surface coal mining is most often production drilling of the overburden material. However, some materials such as softs (soils), weathered overburden, shales and some grades of coal may not require drilling and blasting, since they may be mined directly using:

- rippers
- scrapers
- bucket wheel excavators
- hydraulic shovels, etc.

Ripping, or free-digging can be very expensive, equipment repair and maintenance costs can be 130% more when used in material that is only marginally rippable – or where the equipment size or breakout forces are unsuitable for the application.

The first question then is not so much to assume that drilling is the first phase of production, but to be able to select a rock breaking technique appropriate to the material type. This will be based not only on rippability, but also the production rates of the excavation fleet used.

The geological features which influence ripping are typically:

1. Rock type - sedimentary and metamorphic rocks are more easily ripped than igneous rocks
2. Rock hardness - the softer the rock, the more easily it is ripped
3. Rock weathering - the more weathered, the easier ripped
4. Rock structure - discontinuities of all kinds are planes of weakness and make ripping easier
5. Rock fabric - coarse-grained rocks rip more easily than fine-grained rocks.

The Rippability Rating chart has been produced from work done on support requirements for tunnels. Using the geotechnics classification system, it is possible to produce a rating for the
assessment of rippability. Table 4.1 shows this chart and when the total rating for the rock is in excess of 75, the material should be regarded as unrippable without pre-blasting.

Table 4.1 Rippability rating chart

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<thead>
<tr>
<th>Rock class</th>
<th>I</th>
<th>II</th>
<th>III</th>
<th>IV</th>
<th>V</th>
</tr>
</thead>
<tbody>
<tr>
<td>Description</td>
<td>Very good rock</td>
<td>Good rock</td>
<td>Fair rock</td>
<td>Poor rock</td>
<td>Very poor rock</td>
</tr>
<tr>
<td>Seismic wave velocity (m/s)</td>
<td>&gt;2150</td>
<td>2150-1850</td>
<td>1850 - 1500</td>
<td>1500 - 1200</td>
<td>1200 - 450</td>
</tr>
<tr>
<td>Rating</td>
<td>26</td>
<td>24</td>
<td>20</td>
<td>12</td>
<td>5</td>
</tr>
<tr>
<td>Rock hardness</td>
<td>Extremely hard rock</td>
<td>Very hard rock</td>
<td>Hard rock</td>
<td>Soft rock</td>
<td>Very soft rock</td>
</tr>
<tr>
<td>Rating</td>
<td>10</td>
<td>5</td>
<td>2</td>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>Rock weathering</td>
<td>Un-weathered</td>
<td>Slightly weathered</td>
<td>Weathered</td>
<td>Highly weathered</td>
<td>Completely weathered</td>
</tr>
<tr>
<td>Rating</td>
<td>9</td>
<td>7</td>
<td>5</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>Joint spacing (mm)</td>
<td>&gt;3000</td>
<td>3000-1000</td>
<td>1000 - 300</td>
<td>300 - 50</td>
<td>&lt;50</td>
</tr>
<tr>
<td>Rating</td>
<td>30</td>
<td>25</td>
<td>20</td>
<td>10</td>
<td>5</td>
</tr>
<tr>
<td>Joint continuity</td>
<td>Non continuous</td>
<td>Slightly continuous</td>
<td>Continuous no gouge</td>
<td>Continuous some gouge</td>
<td>Continuous with gouge</td>
</tr>
<tr>
<td>Rating</td>
<td>5</td>
<td>5</td>
<td>3</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Joint gouge</td>
<td>No separation</td>
<td>Slight separation</td>
<td>Separation &lt;1 mm</td>
<td>Gouge - &lt;5mm</td>
<td>Gouge - &gt;5mm</td>
</tr>
<tr>
<td>Rating</td>
<td>5</td>
<td>5</td>
<td>4</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>Strike and dip orientation</td>
<td>Very unfavourable</td>
<td>Un-favourable</td>
<td>Slightly unfavourable</td>
<td>Favourable</td>
<td>Very favourable</td>
</tr>
<tr>
<td>Rating</td>
<td>15</td>
<td>13</td>
<td>10</td>
<td>5</td>
<td>3</td>
</tr>
<tr>
<td>TOTAL RATING</td>
<td>100-90</td>
<td>90-70</td>
<td>70-50</td>
<td>50-25</td>
<td>&lt;25</td>
</tr>
<tr>
<td>Rippability assessment</td>
<td>Blasting</td>
<td>Extremely hard ripping or easy blasting</td>
<td>Very hard ripping</td>
<td>Hard ripping</td>
<td>Easy ripping</td>
</tr>
<tr>
<td>Kilowatts engine power</td>
<td>&gt;600</td>
<td>&gt;290</td>
<td>&gt;200</td>
<td>&gt;135</td>
<td></td>
</tr>
</tbody>
</table>
A typical assessment for coal seams commonly encountered in the Witbank, Mpumulanga Province of South Africa, is soft rock, unweathered, fractured and horizontally bedded with a tight joint spacing of 200-250mm and a seismic wave velocity (SWV) of 1500-1600m/s (using refraction seismograph). The Rippability Rating chart yields the following rating values:

- **SWV** = 20
- **Hardness** = 1
- **Weathering** = 9
- **Joint spacing** = 10
- **Continuity** = 5
- **Gouge** = 5
- **Strike and dip** = 15
- **Total rating** = 65  
  Analysis is “very hard ripping”

In this case, although ripping or free-digging would be possible, especially from a vertical face, from a horizontal face it would be extremely difficult and unlikely to be cost effective compared to drill and blast. Furthermore, the required production would need to be considered, especially the bucket fill time as part of the overall loading production cycle. Whilst free digging the coal may save on drill and blast costs, these savings may not compensate for the reduction in load and haul fleet productivity.

The bulk of surface mines require breakage of the waste overburden and coal by explosives, and to accomplish this, it is necessary to drill a cavity into the rock. This Module will primarily discuss rotary drilling which is the principle method used in strip mining today. However, in selecting the correct piece of drilling equipment, it is first necessary to know the range of methods available and the conditions under which they would be applied.

### 4.2 Drilling Methods

Drilling methods, herein referring to the actual mechanics of the process, can be broken down into two broad categories, and then further divided into subsections as follows:
- Percussion drilling
  - Drifter (top hammer) drills
    - Air-driven
    - Hydraulic driven
  - Down-the-hole drills (DTH)
- Rotary drilling
  - Drag bit rotary drills
  - Tri-cone rotary drills

4.2.1 Percussion Drilling

Rock is fragmented by repetitive impaction of the drill bit against the rock. The drill engine consists of a piston which gives a series of impacts to the drill rods, and a rotating mechanism to reindex the bit so that a fresh rock surface is presented for each blow. The piston strikes the shank of the drill rod and the impact pulse passes along the rod to the bit, causing it to penetrate and fragment the rock. Rotation can be achieved by either a mechanism contained within the drill machine or an external device. In either case, it is synchronised with the intervals between impacts when the bit has tended to rebound from the rock surface. Finally, to prevent regrind and to ensure a fresh rock-bit contact, a means of removing or flushing the drill cuttings and dust from the hole must be provided, which for these types of drills is commonly compressed air.

4.2.2 Drifter Percussion Drills

In this method of drilling, the drifter, or drill engine (powered either by compressed air or hydraulics), moves along a guide or track on the drill carrier. It contains both the piston and the rotation components. The piston imparts its energy to the bit via a series of drill steels or rods connected by detachable couplings. The drifter and drill string are fed up and down the drill guide by (generally) a motor and chain device, in order to maintain down-pressure on the bit and also to allow the addition and removal of drill steel. Air-operated drifter drills, generally use air at pressures of about 690 kPa to 860 kPa. Hydraulic-operated drifters employ similar principles of operation, except that hydraulic fluid is relatively non-compressible and the
4.2.3 Down-The-Hole Drills (DTH) Percussion Drills

Another approach to percussive drilling is to place the engine in the hole directly above the bit. With this method, straighter holes can be drilled over a longer distance than is usually the case with top-hammer drills. The penetration rate is relatively constant with depth, whereas the penetration rates of top-hammer units tend to decrease as the steel lengths are added, due to blow energy to the bit being attenuated greatly through the absorption by the drill string. At present, all DTHs are compressed air operated at pressures of 1700 kPa to 2400 kPa. The minimum hole size is necessarily limited due to the space requirements and the metallurgy of the components, particularly the piston, to withstand the forces involved. For blasthole purposes, and to preserve the economics of production, this minimum hole diameter would be about 114 to 127 mm.

Percussion drilling, whether by drifter or DTH, is possible in all types of material regardless of hardness, strength or abrasiveness. However, such a range of applications does not imply that percussion drilling is the economically optimum choice, especially in softer rock types.

4.2.4 Rotary Drilling

In rotary drilling, the drill bit attacks the rock with energy supplied to it by a rotating drill stem. The drill stem is rotated while a thrust is applied to it by a pulldown mechanism using up to approximately 65% the weight of the machine, forcing the bit into the rock. The rock is broken either a ploughing-scraping action in soft rock, or by a crushing-chipping action in hard rock, or by a combination of the two.

Compressed air is supplied to the bit via the drill string. It both cools the bit and provides a medium for flushing the cuttings from the hole. Water may be used in addition to the compressed air to suppress the
effects of dust, however, this is normally found to have a detrimental effect on bit wear. The consideration here is more toward providing enough volume to maintain a suitable bailing velocity rather than pressure, actual pressures in rotary drilling run in the range of 340 to 690 kPa only.

Rotary drilling is more of a brute strength operation and requires massive pulldown systems and rotation power. However, with a suitable high-pressure air supply, a rotary drilling machine can be used with downhole drills should conditions dictate. Some manufacturers, such as Reedrill, provide drilling equipment that can be used either as DTH machines or for rotary drilling. This introduces considerable flexibility for the operator since the type of drilling can be matched to the formation encountered.

For blasthole rotary drilling, bits take usually two configurations;

- **Drag bits.** These bits are generally confined to quite soft formations (usually less than 130 MPa compressive strength) and in smaller diameters in the range of 89 to 152 mm.

- **Tri-cone bits.** Tri-cone or rolling-cone bits are the most widely utilised for blasthole drilling. The bit consists of three cones that rotate on a combination of roller and ball bearings. On these cones some kind of cutting teeth are mounted, the design depending on the hardness and compressive strength of the material being drilled. On some of these bits the teeth are cut on the cone body itself for softer formations, whereas for harder rock, the cutters are hardened steel or tungsten carbide inserts or buttons.

When rating rotary drilling rigs, the emphasis is placed on the pulldown power in terms of kilograms of force exerted by the feed system. In reality, most of the power consumption in a rotary drill rig is apportioned off in driving the air compressor (about 60 %) and in rotation power (or torque to maintain rpm under heavy resistance) (about 25%), whereas pulldown consumes less that 5%.

**4.3 Selection Methods**

Before describing typical application ranges for the drilling methods mentioned above, some of the application factors that should be considered when evaluating and selecting a feasible drilling method and associated equipment will be introduced.
4.3.1 Selection of Hole Diameter

Generally, the larger the blast hole diameter the less expensive the drilling cost per unit of production. This is only true if the diameter conforms to the particular operation, especially in terms of the results from blasting (which is the ultimate purpose of a drill hole). The first step in the process is to determine what hole diameter is most suitable for the application, bearing in mind that this could change over time. This is probably the single most important factor since it will determine the size, quantity and type of drill that will be needed.

Some of the major factors are:

- Material characteristics
- Blast design
- Bench height
- Type and size of excavating and hauling equipment
- Terrain.

MATERIAL CHARACTERISTICS
This refers primarily to the characteristics of the rock that lend themselves to drillability and blasting fragmentation.

Hardness or the compressive strength (UCS) of the rock. From the previous section it was noted that percussive drilling is not as radically effected by rock hardness as is rotary drilling. This is because there are certain restraints placed on rotary bit bearing loading, particularly as the diameter decreases. The other effect of rock hardness is its resistance to blast fragmentation, particularly if it is somewhat homogeneous in nature. This can effect the distribution of the explosives in the bench, so that smaller holes, closer together may have to be considered, depending on desired fragmentation.

Rock structure or existence of joints, fractures, bedding planes or faults etc. If the rock, even with a high UCS, is friable, then larger holes farther apart may be used and still deliver optimum fragmentation. On the other hand, if a well defined joint or fracture pattern is present, then smaller holes closer together may be necessary.

Thus it is clear that drill selection based on hole diameter or UCS alone is inappropriate and some cognisance of blasting requirements
is needed. Figure 4.1 illustrates a relationship between hole diameter and rock UCS as an initial guide to selection.

![Drilling Machine Selection (modified after Praillet, 1983)](image)

**BLAST DESIGN**

This is a selection factor based on the most appropriate or "balanced" blast design for a particular combination of bench height, hole diameter and material types, as will be discussed in the following Module. To select a piece of drilling equipment and to then dictate through that the type of blast design, will often result in sub-optimal blasting. Further parameters are related to blasting limits, especially vibration, and although larger holes offer savings in cost per unit of production, the larger the hole, the more weight of explosives per hole, and unless multiple in-hole delays are used, the higher the weight per delay and the greater the vibration that results.

**BENCH LIFT OR HEIGHT**

If there is a fixed existing bench height, then selection of blasthole size has to be made with this in mind. In many cases bench height is pre-determined by stratigraphy, production equipment and production volume requirements. In selecting equipment for a particular bench height, the practicality of drilling to the required depth (often including sub-drill below bench height) should be considered. Percussive drills are limited in the depth to which they can effectively drill, due to losses in percussive energy in a drifter, or air loss in DTH drilling,
together with the necessity to extend the drill string every 3 to 7m. Rotary drills are ideally applied in single-pass drilling, where the drill string (and masthead) is large enough to contain the length of drill string necessary to complete the hole without extensions being added.

Figure 4.2 illustrates the typical hole depth and diameter limitations for the major drill types discussed.

Figure 4.2  Typical hole depth and diameter limitations for drilling in medium-hard abrasive (above) and softer (below) rock types
TYPE AND SIZE OF EXCAVATING AND HAULING EQUIPMENT

The primary concern here is to balance the amount of drilling required to fragment a given volume of rock (m$^3$ drilled per BCM or drilling density measured as holes per m$^2$ bench surface). If large excavators and haul units are employed, the blastholes should be as large as possible (within the limits of good blast design) and the drill pattern spread, since larger sizes of blasted rock can be handled. To a certain extent this is true, but it can be argued that the primary purpose in larger excavating and hauling units is to promote greater production capabilities more economically, not to save money on the drilling and blasting phases. In large tonnage operations, the cost effect of an increased drilling density is more often than not offset by improved rock diggability and production rates. On the other hand, if excavating and hauling equipment is relatively small, then careful consideration of hole size relative to desired fragmentation should be made.

TERRAIN

The larger the hole diameter, generally the larger the drilling platform that is required. Big drills, even those mounted on tracks are more limited in their ability to traverse adverse landscapes than smaller machines.

When developing a new mine from scratch, in the pioneering phase, then gradability and "traversability" of various drill units should definitely be studied. As the operation matures and will grade into better defined series of benches, the constraints will be altered. Therefore, initially it might be wise to choose a smaller hole diameter and a smaller drills until such time as the development proceeds to the point where larger holes and larger drill rigs can be utilised.

When mining matures into a more or less conventional mining method, the other considerations include an evaluation of the importance of the mobility factor versus the possibility of less maintenance and probably more efficient drilling in the sense that while on the blast block, the track-mounted rig should consume less time in moving from hole to hole, thereby reducing the "non-drilling" time of the cycle. The mobility requirements of the unit must also be considered in terms of mounting and power supply, especially if the drill rig is required to drill small bench blasts and these are removed from each other. In this case, drill relocation time is an important consideration and track-mounted electrically driven units are more problematic, requiring trailing cable management systems and /or a low-bed for long distance relocation (for example, bench to bench blast block).

In general, truck-mounted units have an advantage of speed and mobility between sites and drill areas. Its disadvantages are;
• It cannot traverse severely adverse terrain.

• It does not provide as substantial and as heavy a platform for straight rotary drilling as track machines.

• It takes longer to set up on a hole and usually requires an additional person to spot holes.

The track-mounted configuration overcomes most of the disadvantages of the truck mounting except it is less adaptable for rapid deployment between drilling areas.

Both rotary drill mounting types are limited in their ability to drill inclined blastholes. This capability is purchased as an option and is usually limited to 20 to 30° holes in 5° increments and in one plane only. This causes increased set up time because the rig has to approach the hole location in a plane perpendicular to the face rather than along a parallel plane as with vertical holes.

### 4.3.2 Required Production Rates and Cost of Drilling

In the case of drilling, as with much equipment, costs can be analysed using various techniques and measures. In the final analysis, the bottom line is the cost per meter drilled and this value can be made comparable to other drill bit types, and even other types of drills or drill hole diameters. Larger holes are more expensive to drill (in terms of unit volume of material removed), but may work out cheaper in terms of production unit or total bench volume excavated (by virtue of the larger burden and spacing possible with bigger blastholes). When analysing the cost of drilling, the first assessment is between types of drill, and then between drill bit types and hole diameters.

The amount of drilling and blasting depends on the types of strata present in the area. The number of drills needed for drilling can be found by considering the case of overburden drilling to uncover a certain areal exposure of coal seam;

\[
\begin{align*}
\text{COAL}_A &= \text{Square meters of coal to be exposed per month} \\
\text{OB}_T &= \text{Overburden depth (m)} \\
\text{D}_S &= \text{Spacing of the overburden drill holes (m)} \\
\text{D}_B &= \text{Burden of the overburden drill holes (m). (In this case no sub-drill used since blasting to top of coal is assumed)}
\end{align*}
\]
\[ D_R = \text{Gross drilling rate (m/hour)} - \text{the average speed of penetration inclusive of non-drilling time (maneuver, reposition, leveling, extensions, blowing, etc.).} \]

\[ H = \text{Hours available per shift for drilling (availability and utilisation of drilling rig over full shift, including time loss due to breakdown, maintenance, operator breaks, etc.)} \]

\[ S = \text{Shifts per day} \]

\[ D = \text{Drilling days per week} \]

\[ W = \text{Weeks worked per month} \]

\[ N_D = \text{Number of drills required} \]

\[ N_D = \frac{COAL \cdot OB}{D_s \cdot D_R \cdot H \cdot S \cdot D \cdot W \cdot D_R} \]

When evaluating drilling hours per day, it is common to find drill utilisation is only 50-60%, based on;

Available operating time = Shift time – maintenance and mechanical/electrical downtime

Availability % = \[
\frac{\text{Shift time – maintenance and mechanical/electrical downtime}}{\text{Shift time}}
\]

Utilisation % = \[
\frac{\text{Available operating time – non drilling activity time}}{\text{Available operating time}}
\]

This is due in part to the necessity to shut drills down due to blasting, extra capacity in drill fleets to maintain adequate floor stocks, scheduling problems and long drill moves where the drill mast must be lowered. Table 4.2 gives a typical drill availability and utilisation time breakdown.
Table 4.2 Drill availability and utilisation time breakdown (on an annual hours basis)

<table>
<thead>
<tr>
<th>Activity</th>
<th>Hours</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total calendar time</td>
<td>8760</td>
</tr>
<tr>
<td>Less holidays per year</td>
<td>216</td>
</tr>
<tr>
<td><strong>Possible available time</strong></td>
<td><strong>8544</strong></td>
</tr>
<tr>
<td>Less maintenance and repair downtime</td>
<td>1440</td>
</tr>
<tr>
<td><strong>Available operating time (equipment available)</strong></td>
<td><strong>7104</strong></td>
</tr>
<tr>
<td>Less operational restrictions</td>
<td>624</td>
</tr>
<tr>
<td>Less long drill moves and other major interruptions</td>
<td>216</td>
</tr>
<tr>
<td>Less personnel time:</td>
<td></td>
</tr>
<tr>
<td>travel time</td>
<td>432</td>
</tr>
<tr>
<td>lunch</td>
<td>432</td>
</tr>
<tr>
<td>other</td>
<td>72</td>
</tr>
<tr>
<td><strong>Less other non drilling time:</strong></td>
<td></td>
</tr>
<tr>
<td>lubrication &amp; inspection</td>
<td>288</td>
</tr>
<tr>
<td>short moves</td>
<td>72</td>
</tr>
<tr>
<td>running repairs</td>
<td>216</td>
</tr>
<tr>
<td>other</td>
<td>216</td>
</tr>
<tr>
<td><strong>Net drilling time</strong></td>
<td><strong>4536</strong></td>
</tr>
</tbody>
</table>

In the case of the data in Table 4.2, availability is 83%, utilisation 64%, the percentage time spent operating (as a percentage of overall time) is 53% equivalent to 5.1 operating hours per 8 hour shift.

From this basic data it is necessary then to compare fleet size requirements across a range of drill hole diameters (that are feasible for the bench geometry and blast design) to determine the fleet size for a given hole diameter. Table 4.3 gives a typical evaluation based on the data in Table 4.2 for two sizes of drill hole. It is important to consider not only present needs, but also the growth path in
succeeding years, dictated by the overburden depth, coal qualities (recovery per unit coal in-situ which would impact on coal exposure rates), overburden material types and gross penetration rate and how this may effect the future drilling aspect of the operation.

Table 4.3 Drill fleet number evaluation for drilling a 12m bench in three formations with a rock density of 2.56t/m³

<table>
<thead>
<tr>
<th>Drill size (mm)</th>
<th>Annual tonnage requirements</th>
<th>Blast design burden (m)</th>
<th>Blast design spacing (m)</th>
<th>Formation tons per hole</th>
<th>Number of holes drilled</th>
<th>Gross drilling rate (m/shift)</th>
<th>Drill operating shifts</th>
<th>Drills required</th>
</tr>
</thead>
<tbody>
<tr>
<td>229</td>
<td>9688896</td>
<td>8.2</td>
<td>9.1</td>
<td>2292</td>
<td>4227</td>
<td>57905.31</td>
<td>91</td>
<td>636.3</td>
</tr>
<tr>
<td></td>
<td>11303712</td>
<td>7.3</td>
<td>8.2</td>
<td>1839</td>
<td>6147</td>
<td>84213.89</td>
<td>84</td>
<td>1002.5</td>
</tr>
<tr>
<td></td>
<td>11303712</td>
<td>6.4</td>
<td>7.3</td>
<td>1435</td>
<td>7876</td>
<td>107899.05</td>
<td>68</td>
<td>1586.8</td>
</tr>
<tr>
<td>TOT</td>
<td>32296320</td>
<td></td>
<td></td>
<td></td>
<td>250018.2</td>
<td>3225.619</td>
<td>(i.e. 4)</td>
<td></td>
</tr>
<tr>
<td>270</td>
<td>9688896</td>
<td>9.1</td>
<td>10.0</td>
<td>2796</td>
<td>3466</td>
<td>47482.36</td>
<td>137</td>
<td>346.6</td>
</tr>
<tr>
<td></td>
<td>11303712</td>
<td>8.2</td>
<td>9.1</td>
<td>2192</td>
<td>4931</td>
<td>67556.20</td>
<td>122</td>
<td>553.7</td>
</tr>
<tr>
<td></td>
<td>11303712</td>
<td>7.3</td>
<td>8.2</td>
<td>1839</td>
<td>6147</td>
<td>84213.89</td>
<td>103</td>
<td>817.6</td>
</tr>
<tr>
<td>TOT</td>
<td>32296320</td>
<td></td>
<td></td>
<td></td>
<td>199252.4</td>
<td>1717.936</td>
<td>(i.e. 2)</td>
<td></td>
</tr>
</tbody>
</table>

Once fleet size options are known, annual drill operating and maintenance costs can be determined and an economic assessment of the most suitable option completed in terms of capital costs and net present value. The hourly costs to operate the drill machine and related equipment vary considerably with machine and hole size and are quite significant. The total drilled cost per meter (TDC) of hole drilled with rotary blasthole bits can be determined by the equation:

\[
TDC = \frac{C}{M} + \frac{H}{D_R}
\]

Where:

\[
TDC = \text{Total drilled cost per meter}
\]
$C = \text{Consumable costs – made up of the cost of the entire suite of drilling consumables for the particular method selected}$

$D_R = \text{Gross penetration rate (m/hour)}$

$H = \text{Hourly rig operating rate – includes all aspects of the hourly cost (excluding consumables) and would typically cover;}$

- Repair, maintenance and overhaul costs
- Parts costs
- Maintenance and operating labour costs
- Energy costs
- Lubrication costs
- Undercarriage

$M = \text{Meters drilled}$

The factor $D_R$ refers to gross penetration rate or the time spent actually drilling (bottom-hole time) as related to the time spent in mechanical down-time and time spent in the non-drilling functions which include moving, setting up, adding drill pipe or steel, pulling rod or steel from the hole, time "stuck" in the hole etc. From the previous sections it should be clear that in choosing a drill rig only on the basis of its ability to "punch" down holes is therefore not enough without consideration of these other parameters.

### 4.4 Rotary Blasthole Drilling

A typical rotary drilling machine, shown in Figure 4.3, is capable of drilling holes from 102 to 445mm diameter to a depth of 50m or more.

Most rotary drills are track-mounted and totally self-propelled. They are driven by a diesel engine (up to 27t pulldown on drill bit) or equipped with an electric drive system (above 27t pulldown or 230mm diameter). Hydraulic power is used for tramming, leveling, hoisting, rotation and the application of pulldown.
Drilling takes place through the application of pulldown and rotation on the tricone drill bit via the drill steel. When the compressive strength of the rock is overcome, penetration takes place. The resulting cuttings are removed out of the hole during drilling by air pressure (bailing air).

4.4.1 Tricone Rotary Drill Main Components

The main components of a tricone drill are:

- Machine room - Contains the engine (diesel-generator or electric motors, compressor and pumps for hydraulic system).
- Cabin - Must give operator a view over the work area and blasthole collar.
- Tracks - Width depends on the ground conditions and the allowable bearing capacity of the material under the machine. Tracks also provide better stability during the drilling process.
- Mast - It is very important to choose the mast according to the depth of the blasthole that is required in single-pass drilling. If an inclined borehole is required, support is necessary to position the mast correctly. If for example a hole of 30m depth must be drilled, a mast of 25m length will require that an extra drill steel must be added to complete the hole. This requires extra handling equipment in the mast itself (costly) and reduces the drill’s productivity.
- Drillhead - Contains the drill steel rotation motor which is subjected to heavy torque and loadings. Rotation speed is generally between 60 and 120 rpm., depending on the rock type.
- Pulldown system - For loading the drill bit, mostly by hydraulic cylinders and/or a chain system that applies the load to the drill bit. The pulldown can vary from 0t to 68t (depending on the drill bit diameter and weight of the rig itself). This system is also used to pull the drill steel out of the blasthole, or when the hole is very deep, to reduce the drill bit loading (due to the weight of the long drill steel).
- Leveling cylinders - For the alignment of the machine in the x-and y-plane in order to achieve and maintain correct blasthole direction.

**4.4.2 Rotary Drill Penetration Rate**

One of the most important factors in drilling is how fast can a drillhole be produced while the machine is actually drilling. This factor almost entirely influences productivity and has a strong influence on unit costs as seen earlier. It will, therefore, be used as a starting point for this discussion of some of the important aspects of blasthole rotary drilling.

From studies conducted in iron-ore operations it was found that penetration rate is linear with the pulldown weight and rotary speed. The penetration rate is estimated following Bauer and Calder as;
\[ P = \left( R_f - 28 \log_{10} \frac{W \omega}{17614d} \right) \]

where

- \( P \) = penetration rate, m/h
- \( R_f \) = rock factor (Table 4.4)
- \( UCS \) = uniaxial compressive strength, MPa
- \( \omega \) = rotational speed, r/min
- \( W \) = pulldown weight, kg
- \( d \) = blasthole diameter, mm

In weaker formations, the above formula predicts penetration rates much lower than those achieved in practice due the changing nature of the rock breakage under the indentors as the rock strength decreases. This action varies from a crushing and chipping action in hard rock to a ploughing and scraping action in soft formations. Medium strength rocks are likely to fail under the indentors by a combination of the two processes.

Regardless of the formation, the penetration rate remains linear with the pulldown and rotational speed. This linearity requires that the pulldown be not so great as to cause the penetration per revolution to be greater than the height of the cutting elements.

Table 4.4 provides recommended values of the empirical rock factor for different rock strengths. These values provide reasonable predictions, but variations may occur in specific geological environments. For example, the pulldown or rotational speed may be lower than expected in a soft rock that is highly fractured, because of the vibration of the drill.

For typical overburden found in the Mpumulanga coalfields, UCS values of 60-170MPa are typical.
Table 4.4  Recommended values of the rock factor for different rock strengths

<table>
<thead>
<tr>
<th>Rock</th>
<th>Uniaxial compressive strength, (UCS) MPa</th>
<th>Penetration rock factor, R_f</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very hard</td>
<td>&gt;207</td>
<td>84.5</td>
</tr>
<tr>
<td>Hard</td>
<td>103—207</td>
<td>104.0</td>
</tr>
<tr>
<td>Moderate</td>
<td>69—103</td>
<td>123.5</td>
</tr>
<tr>
<td>Soft</td>
<td>34—69</td>
<td>158.0</td>
</tr>
<tr>
<td>Very soft</td>
<td>7—34</td>
<td>224.0</td>
</tr>
<tr>
<td>Extremely weak</td>
<td>&lt;7</td>
<td>323.5</td>
</tr>
</tbody>
</table>

An alternative formula was proposed by Praillet, specifically for softer formations, in which penetration is given by:

\[
P = \frac{63.9W \omega}{UCS^2 d^{0.9}}
\]

Figure 4.4 compares the application of these prediction equations using 70rpm, a pulldown of 55t on a 311mm diameter blasthole in rock with a UCS of 60-170MPa and rock factors as given in Table 4.4. It is evident that for the most reliable value and where at all possible, test drilling should be undertaken to confirm the actual penetration rate achieved under varying conditions of pulldown, rotation speeds, etc.

To achieve maximum penetration tempo in various rock types, considerable skill is required to match rpm with pulldown to optimise penetration rate. The relationship between these parameters lend themselves to automatic control and monitoring.

To appreciate the factors which go into this, consider that the rotary drilling penetration rate equation (Bauer and Calder) states that, for a given rock strength, the penetration rate is proportional to the weight per mm of bit diameter and the rotary speed. The weight per mm of bit diameter is a function of bit size; for reasonable bit life, it varies from 100kg per mm of diameter at 228mm diameter to 135kg per mm at 312mm diameter.

For a given diameter of hole, one approach would be to keep the pulldown weight and rpm constant and at the desired values. The
The approach being used is to determine the drilling speed from testing at about the normal pulldown weight. The drill pipe feed rate is now selected and dialed into the machine. The limit of pulldown weight or weight per mm bit diameter is also put in as a control. Desired rpm is set into the operation along with desired depth of hole and air pressure sensors are interlocked into the system that, if vibration increases beyond a certain preset level, the rpm and pipe feed rate are reduced in increments. If the air pressure goes beyond the preset limit, then the feed rate is slowed down.

When the foregoing limits are not exceeded, rpm and feed rate are stepped up incrementally in a set time sequence providing the limiting value is not again exceeded. Indications are that an appreciable increase in meterage drilled and an increase in bit life can be gained from using integrated drill monitoring systems.

![Figure 4.4](image)

**Figure 4.4** Comparison of penetration rate formulae for rotary tri-cone drilling

The rotary drive motor turns the drill tool string thus turning the drill bit at the bottom of the hole. This action brings the successive lines of teeth or buttons into contact with the base of the hole. As the rotary speed increases so the number of contacts increases and the penetration rate increases. The limit to rotary speed is hot bearings in the bit or stripping of the heel row teeth. Current rotary speeds vary from 60 to 90 RPM for hard materials with greater speed for softer rocks. The limitation on penetration rate at many mining properties is
the rotary power available. The power requirement \( KW_R \) can be estimated using the empirical equation:

\[
KW_R = k \omega d \left( \frac{W}{1000} \right)^{1.5}
\]

Where;

\( KW_R \) = Rotary power requirements (kW), increased with 25 to 30% where drill string stabilisers are used.

\( k \) = Constant, typically 13.5\( \times 10^{-6} \) for soft rock, to 3.7\( \times 10^{-6} \) for hard rock.

The other important aspect as regards the rotary power requirement is the method of stabilisation. The type of stabiliser, or for larger holes, whether stabilisation is used at all, will greatly effect the rotary power requirement and thus the penetration rate in most materials.

A portion of the machine weight is applied by the pulldown motor via the pulldown chains, rotary head and drill stems to the drill bit. As the drill bit size increases, the bearing size increases thus allowing an increase in the tolerable load. Over loading the bit results in severe loss of bit life. The maximum pulldown \( W \) (kg) is typically:

\[
W = 0.57d^2
\]

### 4.4.3 Tricone Rotary Drill Bits Selection and Air Requirements

The specific choice of the button (or tooth) drill bit design depends basically on the compressive strength of the rock. Table 4.5 gives further basic information, but reference should be made to individual suppliers for detailed selection criteria.

Typically the problem on a mine is to assess different bit designs in the same drilling scenario (often as a result of encountering various rock types in the same hole). For example, when three bits of Type A were run alternately with three bits of Type B, Type A bit averaged 141.9m in 27.7 hours; Type B bit averaged 157.7m in 33.0 hours. Which bit made the better run?
Table 4.5 Basic choice parameters for drill-bit types

<table>
<thead>
<tr>
<th>Description of Rock</th>
<th>Compressive Strength (UCS)(MPa)</th>
<th>Drill Bit Type</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very Soft</td>
<td>&lt;34</td>
<td>Long steel teeth</td>
</tr>
<tr>
<td>Medium soft</td>
<td>35-69</td>
<td>Short steel teeth</td>
</tr>
<tr>
<td>Medium hard</td>
<td>70-103</td>
<td>Medium to short tungsten carbide teeth</td>
</tr>
<tr>
<td>Hard</td>
<td>103-207</td>
<td>Buttons (med long and narrow)</td>
</tr>
<tr>
<td>Very Hard</td>
<td>&gt;207</td>
<td>Buttons (short and broad for the hardest rock types)</td>
</tr>
</tbody>
</table>

A comparison of the average meterage will sometimes show this, but in this case where Type B bit drilled somewhat more hole at a lower penetration rate, it is not obvious which bit drilled at a lower "cost per meter."

Using R45 per hour drill operating cost and R5000 bit cost, the total average cost per meter may be calculated, resulting in the following answers:

Type A bit cost per meter = R43.79
Type B bit cost per meter = R41.12

Now the bits can be compared and Type B bit is the better due to the lower cost per meter.

Air is used to bail the drill cuttings from the hole as well as cool the bit bearings and, when used, roller stabiliser bearings. Approximately 20% of the air is forced through the roller cones for cooling purposes by adjusting the air pressure across the bit using the bit nozzles.

There are two factors important to the successful bailing of cuttings from the hole;

- the pressure drop across the bit
- the velocity of the bailing air.
The pressure drop governs the split of air sent to the bit bearings and the air into the bottom of the hole, which sweeps the cuttings away from the bottom of the hole. Once a chip clears the bottom of the hole, the upward velocity must be sustained to clear the cuttings from the hole. The cuttings must be flushed rapidly enough to avoid clogging of the annular spacing between the pipe and the borehole wall. If the chips do not clear the bottom of the hole quickly, or are not propelled up the hole fast enough, the penetration rate decreases. The bailing velocity is given by:

\[ U_m = 126.2 \sqrt{\rho \sqrt{d}} \]

Where

- \( U_m \) = balancing air velocity, m/min
- \( \rho \) = relative density of cuttings
- \( d \) = diameter of cuttings, mm.

A bailing velocity of 1800 m/min is usually adequate to remove cuttings of 13 mm diameter. Velocities above 3000 m/min are generally not advised because of excessive damage to the drill rods and related equipment. In the drilling of soft formations such as coal overburden, the chips are often inconsistent in size and shape. They tend to bind together to form larger agglomerations of cuttings, and can stick and plug the area between the borehole wall and the drill steel. Thus, when such formations are being drilled, it is advisable to allow more annular area. A larger compressor is then required to maintain the same bailing velocity.
# SURFACE STRIP COAL MINE BLAST DESIGN

## Learning outcomes

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<td>Role of blasting in cyclic excavation systems</td>
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<td>Blast design terminology</td>
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<td>Factors to consider in blast design</td>
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<tr>
<td>The relationship between cost and fragmentation</td>
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<td>Guidelines for successful design</td>
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<td>Factors for initial design</td>
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<td>Requirements of blasthole choice</td>
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<td>Empirical design rules and balance of design concepts</td>
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<td>Results of blast to determine its efficacy</td>
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<td>The concepts of hole layout and coverage for overburden and coal blasting</td>
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<td>Muck pile considerations – shape of blasted overburden with normal, buffer or cast blast</td>
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<td>Initiation sequence and how hole layout effects blast results for various types of coal strip mine overburden blasts</td>
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<td>Airblast, ground vibration and flyrock problems, their limits and avoidance strategies</td>
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<td>Cast blasting design parameters</td>
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</table>

## Apply, calculate or predict

| Controllable and uncontrollable variables in strip coal mine blast design |
| Powder factor, RBI and burden, spacing, stemming and specific charge for a surface blast layout |
| The effect of changes in fragmentation throughout the production process |
| Empirical rule for blasthole diameter and bench height limits |
| Burden, spicing, stemming, bench height, charge or modifications to design values |
| How to improve a blast design based on results achieved |
- Limits to a blast design
- Control measures, limiting charge per delay amounts for cautious blast design
- Appropriate applications of square and staggered hole layouts in strip coal mine blasting
- Chevron initiation with square or staggered hole layout

**Evaluate or design**
- The balance and suitability of a concept blast design, layout and charges based on powder factor or RBI
- Apply value judgments to a proposed initiation sequence for a surface mine blast
- Evaluate the fragmentation achieved and the cost implications throughout the production process
- Make value judgments concerning proposed blast designs
5.1 Introduction to Explosives and Blasting

Where large reserves of coal are situated near the surface of the earth, surface strip or terrace mining is often the most convenient and cost effective method of mining. In surface strip coal mining, large amounts of explosives are used for primary blasting of the waste or overburden overlying the coal seam. Primary blasting breaks the rock out from solid, whilst secondary blasting reduces the large (primary) blocks to a smaller size if necessary. Secondary blasting is expensive and time-consuming reduces the productivity of the mine. In most cases primary blasts are design in such a way as to reduce secondary blasting to the minimum (the ideal is for example that primary blasting produces the necessary tonnage of rock at the required fragmentation “one-shot”). The concept applies equally whether overburden is being blasted, or the coal seam itself. All that differs is the outcome of the blast, most significantly fragmentation.

The safety and economic aspects of a blast design are most important: The requirements of a Code of Practice include specifications and blast design method statements, both for production and wall control blasts.

Maximum profitability depends largely on how effective is the excavation system - either mechanical or drill and blast, selected typically according to rippability and volumes or production rates of rock to be removed. Drilling and blasting are often amongst the first unit operations performed in any surface mine and the results of these operations will affect more down line activities, such as loading, hauling, crushing and processing, than any other unit operation. The total effect of a change in the blast design must be continuously reassessed throughout the production process, not only in conjunction with blasting cost changes but also the changes it generates in other activities allied to the further transport and processing of the product. The relationship between cost and fragmentation is generally similar to that shown in Figure 5.1, finer fragmentation reducing loading, transport and crushing (plant) costs, while drill and blast costs increase. Note that these relationships may not hold true for coal blasts – finer fragmentation may result in an increase in cost per ton product – due to loss of fines in the coal process.

Through the combination of the individual costs of each activity, a total cost profile for the mine can be generated, where cost is at a minimum for a certain degree of fragmentation. It is thus important to analyse the effect of any new or optimised design in terms of the effect on total production costs, since coarse fragmentation will definitely reduce drill and blast costs, but the other cost elements may
not necessary be reduced. A good blast design will always ensure minimum costs for the whole operation, both coal and overburden mining. Figure 5.2 illustrates the systems approach to this type of analysis, note how a large degree of "prediction" is initially assumed, at least until the explosive-rock interaction at a particular mine is better understood.

![Figure 5.1 Fragmentation effect on total costs](image)

5.1.1 Blasting Theory

The rock adjacent to a blasthole is affected by a detonating explosive charge in three principal stages:

In the first stage, starting from the initiation point of the explosive, the blasthole expands as a result of crushing of the blasthole walls caused by the high pressure generated by the detonating explosive. The blasthole expands to double its original volume and will remain at this volume for relatively long time (0.1 to 0.4 ms) before radial cracks start to open.

In the second stage, compressive stress waves emanate in all directions from the blasthole with a velocity equal to the sonic wave velocity in the rock. This generates radial cracks. When these compressive stress waves reflect against a rock free-face, they cause tensile stresses to develop in the rock mass between the blasthole...
and the free face. If the tensile strength of the rock is exceeded, the rock breaks in the burden area (the area or distance from the blasthole to the free-face) and burden relief begins, which is the case in a correctly designed blast. Besides the natural cracks in the rock, new cracks are formed mainly by interaction between the stress field around the blasthole and tensile stresses formed by reflection of the outgoing shock wave at the free face. Reaction products expand from the blasthole (which volume now is quadrupled) into the cracks. Fragmentation starts.

Figure 5.2 Mine productivity model related to blast performance

In the third stage, the gas volumes generated upon detonation are driven into the second stage crack formation under high pressure, expanding the cracks. If the distance between the blasthole and the free-face is correctly calculated, the rock mass between the blasthole
and the free face will yield and be thrown forward. Throw of the fragmented rock mass begins as gas loading is converted into kinetic energy.

The explosives reaction in the blasthole is very fast and the effective work of the explosive is considered completed when the blasthole volume has expanded to 10 times its original volume, which takes approximately 5 ms.

5.1.2 Explosives Properties and Types

Explosives used in mining are classified as commercial explosives and sub-classified according to their function;

- Initiating explosives – used as a charge in detonators or detonating cord.

- High explosives – characterised by a high velocity of detonation (VOD), high pressure shock wave, high density and by being cap-sensitive. A cap sensitive explosive can be initiated by a detonator of a certain charge size (eg. 6D or 8D).

- Blasting agents – mixtures of a fuel and oxidiser system, with additional sensitising agents, where none of the ingredients themselves are classified as explosives and are usually not cap-sensitive – typically represented by ANFO (ammonium nitrate fuel oil).

Explosives have undergone considerable development from Alfred Nobel’s invention of dynamite in 1866, being based on nitro-glycerin. The development covers the following generic product types;

- 1st generation: Dynamites and blasting gelatins, sensitised by nitro-glycerin

- 2nd generation: Watergels, or slurry explosives, with an ammonium nitrate (AN) base and sensitised by tri-nitro-toluene (TNT) with other organic compounds

- 3rd generation: Emulsion explosives with an ammonium nitrate (AN) base and sensitised by micro spheres

- 4th generation: Emulsion explosives with an AN base and sensitised by gassing agents
**EMULSIONS**

The formulation of an emulsion contains an emulsifying agent. This water in oil emulsifying agent suspends minute droplets of the ammonium nitrate (or a combination of AN with either calcium nitrate or sodium nitrate) oxidiser within the fuel. This yields a very intimate oxidizer and fuel mix that leads to high detonation velocities.

Emulsions are sensitised to introduce sufficient air into the mix and control the density. Bulk emulsions are used in larger diameter holes and may be mechanically (micro balloons) or chemically (gassing agent) sensitised, with chemical sensitisation being less costly. Bulk loaded product fully fills the cross sectional area of the hole and delivers maximum energy to the surrounding rock. This is a characteristic of all bulk loaded products unless they are intentionally decoupled as is often the case in presplitting. Packaged emulsion will usually result in some decoupling with an attendant reduction in blasthole pressure generated.

**ANFO**

ANFO remains one of the most commonly used products in mining. It is a combination of ammonium nitrate (oxidiser) and fuel oil (fuel). Blasting grade AN prills are typically small and porous and together with their uniformity in size results in a poor packing density, with considerable interstitial voids present. Hence a product which typically bulk loads in a density range of 0.80 to 0.85 kg/m³. ANFO has virtually no water resistance and degradation of the product is immediate, if but slow to completion. Even if holes will be detonated 2 or 3 hours after loading, performance will have been seriously affected. Therefore, whenever ANFO is to be loaded into wet holes, the blastholes should first be pumped and a plastic liner placed in the hole.

One way to increase the energy output in ANFO is to add aluminum. The reaction of ammonium nitrate with aluminum releases more energy per unit of weight. The aluminum must be of a suitable size to be reactive, but not so fine as to constitute an explosion hazard. The upper limit on aluminum addition is usually about 15 percent. As more Al is added to the mix increasing percentages of the energy are trapped in a solid product of detonation. Beyond 15 percent Al by weight there is little additional energy output for the aluminum added.

**HEAVY ANFO**

Another way to increase the energy output of ANFO is to add emulsion to it. The emulsion fills the voids between the prills, the density increases and there is more energy output per unit of
blasthole volume. This class of explosives are known as Heavy ANFO. They provide a cost effective way to increase the energy output of ANFO.

Heavy ANFO may be produced solely for the purpose of increasing the energy output. However, at higher emulsion percentages by weight (typically at 50 percent emulsion addition – but for pumping reliably it is common to use a waterproof Heavy ANFO containing 60 to 70 percent emulsion) these products become waterproof. Such formulations can be bulk loaded into wet holes. The performance of Heavy ANFO becomes sluggish as more emulsion is added unless the emulsion has been sensitised by gassing or microballoons. In softer formations up to 30% of unsensitised product can usually be employed because suitable fragmentation of the rock depends to a greater degree on heave energy. The degree of non-ideal detonation introduced by the lack of sensitisation means that a greater degree of the total energy is released as heave energy.

INITIATION SYSTEMS
All commercial mining explosives require some form of initiation system – usually in the form of a small charge of initiating or priming high explosive – to cause the detonation of the main explosive charge in the blast hole. Some of the explosive types described above are cap sensitive, which means the product can be efficiently detonated by a blasting cap or delay detonator of adequate strength, or by compatible detonating cord. Small diameter emulsions and slurries are typically cap sensitive.

Bulk loaded explosives used in hole diameters greater than 125mm almost always require heavier priming than a detonator alone can provide. Primers can take many forms, usually comprising a cap-sensitive explosive which itself transfers enough energy on detonation to initiate the non-cap-sensitive bulk charge. The higher the primer detonation pressure, the more efficient is the bulk explosive initiation. Thus formulations with high velocity of detonation (VOD) generally give the best results. The primer should also have a sufficient diameter to act on an adequate cross sectional area of the bulk charge, thereby insuring efficient initiation. It must be long enough to allow the VOD in the primer to build up, providing maximum pressure off the end of the primer. Therefore, there is a trade off between length and diameter to provide effective initiation with a primer of reasonable dimensions and cost.

Figure 5.3 illustrates the concept of chain of initiation – how a small energy source (flame, spark, etc.) is progressively magnified (from deflagration – burning, to detonation – exploding) by the chain of
initiation, to eventually detonate a non-cap sensitive bulk explosive charge.

Initiation methods may be classified as;

- Non-electric
  - Fuse and plain detonator
  - Detonating cord
  - Shock tube
- Electric detonators
- Electronic detonators

Figure 5.3 Chain of initiation design options
DETONATORS
As seen in Figure 5.3, detonators are used to initiate the blast. These may be electric or non-electric, instantaneous or delay. For modern day blasting delay detonators are virtually always used. Delay detonators are available for use in the hole, and also for connecting into the surface tie-in.

A delay detonator is similar to an instantaneous cap except that a delay element is included between the initiation charge that is activated by the incoming energy, and the base charge. The delay compound burns at an accurately known rate and provides the desired delay time. The main reasons for using delay detonators will be analysed in detail later in the Module, but can be summarised as;

- Improved fragmentation due to the greater freedom for the material to swell and throw
- Greater flexibility in firing sequences and burden to spacing relationships due to the ability to orient the blast through the tie-in.
- Greater ability to control blast vibration, airblast and more predictable throw.
- Reduced backbreak behind the last row of holes.
- Minimized cut-offs.

Down-the-hole delays are used alone to provide the proper firing pattern or in combination with surface delays. In the former case different delay times are used in the appropriate blastholes to provide the desired sequence of detonating holes. When used together with surface delays a constant down-the-hole delay time is often used. The in-hole delay is of sufficient duration to allow several rows of surface connections and downlines to be activated in advance of blasthole detonations. This approach avoids cut-offs and misfires that reduce blast performance and introduce subsequent safety concerns. When down-the-hole delays are used it is often possible to use longer surface delays without fear of cut-offs.

Fewer blasts in strip mines are initiated totally with electric or fuse-based systems today than once was the case. However, electric detonators are widely used as the lead-in to a shock-tube or other main initiation system.

Construction of electrical caps and delays is similar to non-electric components, except that the energy to ignite the ignition compound is provided electrically. This does have the advantage of minimizing
noise on surface, but has the disadvantage of being more susceptible to stray radio frequency and currents, lightening, etc.

With electronic detonators, the delay element is replaced with a pre-programmable micro-chip, each with a unique identity. The electronic system is very flexible in that up to 1000 individual delay times between 1ms and 16000ms can be programmed. Each detonator can be identified and programmed with a delay, irrespective of its position in the tie-up. On initiation, all holes are activated immediately.

SHOCK TUBE SYSTEMS

The shock tube system is a plastic tube with several thin explosive coatings on the inside of the tube. Upon detonation this material continuously detonates at a low velocity of approximately 2 000m/s. Thus, the plastic tubes are not consumed and the noise level is low. It is, therefore, good to use to connect holes together when used as part of a long lead surface delay system. It is used in the blasthole as a long lead down-the-hole delay system to replace detonating cord downlines (often with top and bottom priming to ensure initiation). Shock tube systems, unlike some detonating cords, will not set off a primer and must always be used with a in-hole initiator and compatible primer. A relatively new delay system combines a surface delay unit and a down-the-hole delay as one unit connected together by a shock tube. A variety of down-hole and surface combinations are available, providing reasonable flexibility for tie-in design.

DETONATING CORD

Detonating cord contains a core load of high explosive (usually PETN) which detonates at about 7 000m/s. Detonating cord is made with various weights of PETN per meter of cord (from 3g/m to 80g/m) and can, in certain circumstances, be used to initiate cap-sensitive explosives. It is itself initiated by a 6D detonator. Detonating cord is used (or required, depending on local regulations) as downlines in the blasthole to transfer initiation energy to primers and down-the-hole delays. It is also used for surface trunklines to connect blastholes together. It is easy to connect up, but has the disadvantage of generating substantial airblast. Therefore, it is usually used on surface when operating in remote locations. Shock tube systems are more commonly used when operating in proximity to built up areas.
5.1.3 Explosive Strength and Performance

The strength of an explosive is a measure of its capacity to break hard materials. Because rock breaking is increasingly being recognised as the single most critical area in many mining operations, and also as a result of the increasing range of explosive types, interest in comparative strengths and performances of explosives has grown.

Owing to the violence of explosives reactions, it is not yet possible to measure directly such basic parameters as explosion pressure or temperature. Initially, the only way of assessing explosives was by comparing their effects, and this approach led to such standardized tests as the Trauzl Lead Block, Ballistic Mortar, and the Dautriche method for measuring detonation velocity using detonating cord. The Ballistic Mortar was the most important means of comparing explosives strengths when Nitroglycerine (NG) based explosives dominated the scene. The ballistic mortar is not suitable for testing ANFO or slurry type explosives (due to small charges used, critical diameter limitations and non-ideal detonation) and thus alternative measures of strength are used nowadays.

The Dautriche method for testing the Velocity of Detonation (VOD) is often thought to indicate the strength of an explosive, but this is not the case. NG explosives illustrate this point well, as they exhibit two quite different characteristic VOD’s. With large, well confined charges the higher detonation velocity (+5 000 m/s) is encountered, whereas small, unconfined charges normally detonate at less than 2 500 m/s. In very general terms, a high VOD gives higher shock energy (cracking) and a low VOD more gas energy (heave or throw).

VOD is of real significance in assessing the condition of an explosive, which is confirmed as being good if a normal figure is achieved. Sub-normal figures indicate that for whatever reason, detonation is not progressing within specification.

There are a number of alternative approaches to determining the energy released by an explosive, all being based on useful energy release following chemical reaction and thermodynamic principles. To simplify this analysis, comparative energy is more often used, in which energy of an explosive is compared to ANFO at 800kg/m$^3$ density.
RELATIVE WEIGHT STRENGTH (RWS)
RWS is the energy per unit mass relative to ANFO, which is taken as 100. The significance of Relative Weight Strength is that it expresses the energy per unit mass being purchased.

\[ RWS = \frac{ASV_{\text{(explosive)}}}{ASV_{\text{(ANFO)}}} \times 100 \]

where ASV = the heat energy of an explosive \((\Delta H_d)\) expressed in MJ/gram.

As an example, where the amount of explosive required to break a given volume of rock is known (referred to as powder factor or PDF – kg explosive/m\(^3\) rock or kg explosive /t rock), it is possible to use RWS to check which explosive would deliver more energy.

If a certain heavy ANFO has a RWS of 84 and ANFO (at reference density of 800kg/m\(^3\)) a RWS of 100, then the energy factor \((E_f)\) can be determined as;

\[ E_f = PDF \times RWS \]

Using a PDF of 0.5kg/m\(^3\), then the energy factor for ANFO is 50, whilst that of heavy ANFO is 42. This implies that the heavy ANFO will have about 16% less energy then ANFO in this application.

RELATIVE BULK STRENGTH (RBS)
RBS is the energy per unit volume relative to ANFO, at a density of 800kg/m\(^3\), which is taken as 100.

\[ RBS = RWS \times \frac{\rho_{\text{(explosive)}}}{\rho_{\text{(ANFO)}}} \]

Where;

\[ \rho = \text{density of the explosive (kg/m}^3\text{)} \]

Relative Bulk Strength is thus dependant on the densities of both the explosive in question and the ANFO with which it is rated. It reflects the energy actually available per unit charged volume of blasthole.
Following from the previous example, assuming the heavy ANFO has a density of 1200kg/m³, the RBS is then 100,8 compared to ANFO at RBS of 100.

The RBS would appear to be the most meaningful of the strength indices, but in practice it can be misleading. Two explosives of the same RBS will not necessarily exhibit the same effectiveness, because:

- a high density, low weight strength explosive will have a greater mass (and use more cartridges) of explosive per hole than a low density, high weight strength explosive, and whether this is advantageous or not will depend on such factors as rock type, cost of drilling and explosive cost.

- a vital characteristic not included in strength calculations is the softness of the composition. Soft explosives like slurries tend to burst their sleeving when dropped down deep holes, more or less filling the blasthole and increasing their effective bulk strength compared to the more rigid NC-based explosives. This refers to the concept of coupling factor in which explosive density is reduced by the ratio of hole and explosive diameters.

- the assumed density of the explosive may be significantly different from actual density. This should be detected in the field, either from the cartridge count or the column rise.

**RELATIVE ENERGY (R.E.)**

This is the concept that relates the energy per meter of charge length to that of ANFO, pneumatically loaded to a density of 1000kg/m³ in a 30mm diameter blasthole.

In order to calculate the RE, the charge mass per meter, \( M_c \) is required, which is given by:

\[
M_c = \frac{\pi d^2 \cdot \rho_w \cdot c}{4}
\]

Where:

\( \rho_w \) = explosive density (kg/m³)

\( c \) = coupling factor (1 for poured or pumped explosives)

For cartridge and/or packed slurry explosives with a cartridge diameter of \( d_c \) in a blasthole of diameter \( d \),
Thus for ANFO at 1000kg/m\(^3\) in a 30mm blasthole, \(M_c = 0.707\) kg/m

Thus;

\[
RE = \frac{(M_c \times RWS_{\text{explosive}})}{0.707}
\]

Following from the previous example, for the heavy ANFO at a density of 1200kg/m\(^3\), loaded in 50mm diameter holes, \(M_c\) is 2,356 kg/m, and R.E. is 280. Thus, provided the RWS and density of an explosive are known, its energy rating either in terms of unit volume or unit charge length can be readily calculated relative to other explosives.

As will be seen in the following sections dealing with blast design, the chief means of determining drilling patterns is by using a PDF. This approach is still useful, but where strength per unit mass can vary, the use of powder factors can be misleading. The inclusion of energy factors in explosive calculations enables corrections to be made to a drilling pattern in proportion to the root power of any increase in explosive relative energy, i.e.:

\[
B_n = B_i \left[\frac{RE_n}{RE_i}\right]^{\frac{1}{2}}
\]

Where:

\[
B_n \text{ and } B_i = \text{ New and current burden (m)}
\]

\[
RE_n \text{ and } RE_i = \text{ New and current relative energy of explosives}
\]
5.2 The Blast Design Process

Blast design is a term used to describe the full range of drill and blast activities at a mine, and the relationship of each within the total mining system. The main objective of the blast design process within the mine is the provision of a framework within which steps towards blast improvement may be implemented, in response to varying geological conditions and monitored impacts of blasting on downstream operations. The blast design process comprises the following main components:

- Blast objectives, relating to achievement of production, grade and cost targets, as well as provision of a safe working environment through cautious blasting
- Characterisation of the rock mass condition, in terms of its blastability
- Formulation of appropriate blast design criteria to achieve the objectives of blasting
- The operational and information systems in use at the mine, and the flow of information between departments and databases, with respect to the requirements of drill and blast
- Implementation of blast designs, and the control measures adopted
- Identification of relevant blast performance indices, and their measurement
- Provision of feedback loops between performance and design objectives, to enable blast optimisation.

Figure 5.4 presents a flowchart of the blast design process, showing the role of the mine operating system in determining the basic design data, and the drill and blast activities within the blast management system. Four distinct processes relating to blast design are identified.

1. The core mining process has the rock mass as the initial input, which is drilled and blasted to produce suitably fragmented material to enable subsequent mining operations of loading and/or crushing.
2. Mining activity is managed by the 'mine operating system', which performs the role of technical information management. This system fulfills mine geology, mine design, and production planning functions. Much of this information is also essential for blast design, notably geology and bench survey. Integration of drill and blast functions with the mine operating system is critical to achievement of effective blasting.

3. Blast objectives, including development of design to achieve those objectives, implementation of designs and measurement of blast performance. Blast results may be measured directly, in terms of fragment size distribution and muckpile shape, or indirectly through the impact of the blasted product on mining operations, notably digging, transporting, crushing, coal recovery and disruptions to mining caused by oversize fragments (as evident from secondary breakage and crusher delays).

4. The final component is the feedback loop between blast performance and design, to enable the process of optimisation. Blast improvement may only be achieved through measurement of the impact of blasting on subsequent mining and processing operations, and provision of that information as feedback to blast design.
5.2.1 The Influence of Blasting Practices on Production

It is very difficult to get good accurate and meaningful figures directly showing the actual influence of the drilling and blasting practices on production. What is the effect of good blasting as compared with not so good and even poor? To obtain this type of data, detailed records in conjunction with accurate blasting control must be available for relatively lengthy periods. This should be for a single property. Comparisons between different properties contain so many other variables that the effects of the blasting operations alone are difficult to isolate.

The first statement that can be made is that blasts that cannot be dug and require reblasting, have extreme effects on both production and costs. The cost of reblasting an area is high as well as difficult. The initial blasting operation tends to open up cracks and fissures in the rock that provide ready exit routes for the gaseous products formed by later explosive charges. The partially blasted material is difficult to drill and a good portion of the explosives are lost down cracks in the rock. The result is that even higher powder factors must be used. The resulting blast will almost certainly be poor even if it can be dug.

Each property that requires drilling and blasting will have a powder factor for different areas of the pit, different rock formations, etc. This blasting will allow a certain excavation system production rate (normally a cyclic system – dragline or truck and shovel). The questions now become, 'Is the production rate satisfactory and at the optimum rate'? In other words, is the drilling and blasting operation good enough?

Some of the questions that should be investigated in order to evaluate the prospect of introducing a test blasting program are based invariably on the follow-on process from blasting – the excavating system as discussed in the following module;

- How long does it take to load the excavator bucket?
- What is the variation in fill time?
- Does the bucket fail to penetrate the muck-pile?
- Does it obviously have to break rock in order to be filled?
- How many boulders require secondary breakage?
- What is the cost per ton for wear items, teeth, bucket liners, cables, cylinders, etc?
- What is the cost per ton for maintenance?
These types of questions will help point out if drilling and blasting practices could be improved. Also a significant jump in explosives energy consumption should be made, say of the order of 25%. The test should run for a long enough period to reduce the influence of other parameters that may affect production, a minimum period of 3 to 4 weeks should be sufficient.

In this manner, the quantity of explosives, or more precisely, explosives energy units, for a given operation can be optimised to give maximum production and minimum operating costs. This is not to say other aspects of the drilling and blasting operation are not equally as important, but they can often be optimised as part of the blast design process in isolation of the production process. Mining operations that have tried this approach to blast test work have found additional benefits also materialise including:

- Less crusher downtime due to oversize
- Reduced secondary breakage costs
- Easier bench grade control, less dozer work
- Reduce truck maintenance, especially with respect to suspensions
- Marked increase in excavation system consumables life, especially for ropes, teeth, etc.
- Reduced shovel maintenance

5.3 Conceptual Blast Design

The conceptual blast design process is shown in Figure 5.5. The critical role played by the geological model in blast design is highlighted. In developing a conceptual design, the starting points are:

- Blast objectives
- Characterisation of the rock mass blastability
- Specific blast design criteria to achieve the objectives
It is impossible to establish a conceptual blast design mechanistically – certain empirical rules must be used to enable the blast to be numerically analysed. Also, as shown in Figure 5.6, it is necessary to distinguish between controlled and uncontrolled variables, geology being the most significant.

The blast objectives must be defined to enable the correct empirical relationships to be adopted to ensure these objectives are met. These include such considerations as:

- If the blasting is for fragmentation or excavation purposes (coarser fragmentation is allowable for the larger types of loaders, draglines etc. and also for coal loading). For excavation alone, since there is no secondary processing, there are no cost savings to be generated by size reduction through blasting.

- Cleaning and loading equipment requirements, especially maximum rock block sizes, rock pile profile (front-end-loader needs a flatter profile than a rope shovel)
Figure 5.6  Blasting design parameters

- How critical is overbreak in comparison with problems of a unbroken toe, large rocks etc.

- Blast-hole sizes available (on drill) and their inclination

- Bench geometry and pit production

- Slope stability, pre-split, and/or buffer blasting to protect the highwall

- The cost effectiveness of the blast in comparison to the productivity of the whole mining system
Guidelines for a successful conceptual blasting are as follows:

i. The design is based on empirical rules and explosive energy distribution in the rock (measured in kg explosives per BCM or kg explosives per ton rock – the so called powder factor PDF).

ii. Proposed conceptual design from (i) for rock type that occurs in block – note that to accommodate different strata in one hole, different loading densities or decking may be required. At this stage consider a single continuous rock type throughout the full length of the hole.

iii. Observation of performance – especially fragmentation, size distribution, total costs etc.

iv. Modification or optimisation of design (ii) to improve productivity R/m$^3$ broken rock and to reduce total costs.

Throughout the process, be aware of changeable ground conditions that can cause blasting results to change. Hence the importance of blasthole logging to determine the rock types to be broken and the conceptual design parameters. Reconciliation occurs between blast performance and design objectives taking account any unforeseen variation in geological conditions, enabling iterative design modifications to be made based on experience.

Reconciliation between blast design and performance is to a large extent dependent on the reconciliation of the blast geology, i.e. were the conditions encountered in the bench the same as those expected at the time of blast design?

For the initial design, take the following factors into account:

- Rock mass characteristics
- Blasthole size (diameter)
- Bench geometry
- Explosive type.

The determination of the specific blast conceptual design criteria can be divided into two activities:

FIRSTLY: Identifying a suitable powder factor (PDF) – the amount of explosives (kg/BCM) necessary to break a specific type of rock.
SECONDLY: Calculate the distribution of the explosive within the rock. The explosive type and blasthole size determines the weight of the explosive per meter length of blasthole (charge density) and therefore the burden and spacing (which determine the BCM’s per hole), to coincide with the powder factor. Take note that certain explosives are sensitive to a minimum hole diameter. Detonation will not occur (or energy release will be sub-optimal) when the blasthole is smaller than the minimum or critical diameter of the explosive.

The design process follows an iterative approach since the previous variables are inter-related; powder factor is a function of the blasthole charged volume which is itself a function of bench height which is a function of the blast design (burden, spacing etc.) which is related to the powder factor. How many iterations are required depends on the “balance” of the design – i.e. to what extent the design meets the requirements of a good design. These requirements are determined mostly from empirical rules that relate blasting parameters.

In any blasting design the starting point will be to establish how much explosives is necessary to break one cubic meter or BCM of rock. Figure 5.7 illustrates typical blast design criteria adopted throughout this Module. In this case, a BCM is defined as the product of Burden (B), Spacing (S) and bench height (H).

Figure 5.7 Blast design criteria
5.3.1 Powder Factor (PDF)

The term powder factor may be used for the energy that is required to break the rock and it is dependent on rock types to be fragmented. There are several approaches to determining PDF.

Published values are an easy, but generally unreliable starting point. In weak rock 0.2 kg/BCM is sufficient, while in hard rock, as much as 1.6 kg/BCM may be needed. A good rule of thumb will be approximately 0.3 – 0.4 kg/BCM if no data is available, but this approach does not really account for the variability found in a rockmass. Other typical values are given in Table 5.1.

Where more detailed rock mass information is available, the Rock Blastability Index (RBI) can be used. The RBI is a number used to describe the blastability of the rock based on the following properties:

- Rock uniaxial compressive strength (UCS)
- Rock density ($\rho$)
- Joint or bedding spacing ($J_s$)
- Joint or bedding orientation ($J_o$)
- Rock mass descriptor (R)

Table 5.1 Estimated PDF for various types of rock

<table>
<thead>
<tr>
<th>ROCK</th>
<th>PDF (kg/BCM)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Soft laminated strata (shales, mudstones, etc.)</td>
<td>0.1 - 0.25</td>
</tr>
<tr>
<td>Med-hard sedimentary strata (sandstones)</td>
<td>0.3 - 0.45</td>
</tr>
<tr>
<td>Hard jointed rock (limestone, dolomite)</td>
<td>0.4 +</td>
</tr>
<tr>
<td>Quartzite, granite, etc.</td>
<td>0.65 +</td>
</tr>
<tr>
<td>Massive dolerite, etc.</td>
<td>0.9 - 1.2</td>
</tr>
</tbody>
</table>

The rock blastability index has a range between 0 and 100 with 100 being very difficult to blast. The equation used to define the rock blastability index is based on the values in Table 5.2. RBI is determined from;
\[ RBI = 0.5 \left( \frac{UCS + 23.7}{47.6} \right) + \rho + J_o + J_s + R \]

The RBI can also be used as a parameter in the fragmentation analysis models discussed later. For conceptual blast design, the values given in Table 5.3 can be used to determine the scaled burden \((B_s)\), without direct reference to a PDF. Scaled burden is defined as;

\[ B_s = \frac{B}{\sqrt{M_c}} \]

Where;

- \( B \) = Burden (m)
- \( M_c \) = Charge mass per meter blasthole length (kg)

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS</td>
<td>Uniaxial compressive strength (MPa)</td>
<td>Actual</td>
</tr>
<tr>
<td>( \rho )</td>
<td>Density (kg/m(^3))</td>
<td>Actual</td>
</tr>
<tr>
<td>( J_o )</td>
<td>Horizontal</td>
<td>10</td>
</tr>
<tr>
<td></td>
<td>Dips out of face to vertical</td>
<td>20</td>
</tr>
<tr>
<td></td>
<td>Strikes normal to face</td>
<td>30</td>
</tr>
<tr>
<td></td>
<td>Dips into face</td>
<td>40</td>
</tr>
<tr>
<td>( J_s )</td>
<td>Planes or joints spaced closer than 100mm</td>
<td>10</td>
</tr>
<tr>
<td></td>
<td>Planes or joints spaced 100-1000mm</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>Planes or joints spaced more than 1000mm apart</td>
<td>50</td>
</tr>
<tr>
<td>( R )</td>
<td>Friable</td>
<td>10</td>
</tr>
<tr>
<td></td>
<td>Blocky</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>Massive</td>
<td>50</td>
</tr>
</tbody>
</table>
Table 5.3 Scaled burden values for RBI

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Bₜ for blasting with limited throw</th>
<th>Bₜ for blasting with maximised throw</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>20-40</td>
<td>40-60</td>
</tr>
<tr>
<td>RBI</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Shales and mudstones</td>
<td>1.3</td>
<td>1.2</td>
</tr>
<tr>
<td>Sandstone</td>
<td>1.2</td>
<td>1.1</td>
</tr>
<tr>
<td>Limestones or dolomite</td>
<td>1.2</td>
<td>1</td>
</tr>
<tr>
<td>Granite</td>
<td>1.2</td>
<td>1</td>
</tr>
<tr>
<td>Dolerite</td>
<td>1.1</td>
<td>1</td>
</tr>
</tbody>
</table>

Note – guidelines only, some field optimisation and adjustment will be required.

With yet more rock mass data, the following formula can also be used to determine the PDF for surface mining blasting with parallel and vertical blastholes, and assuming a “good” fragmentation result:

\[
PDF = 94.6 \log\left[\frac{3.3 \rho \cdot \tan(\phi + i) \cdot \frac{\sqrt{\frac{d}{100}}}{\sigma_c}}{115 - RQD}\right] + 540
\]

Where:

- PDF = Powder factor (g/BCM)
- \( \rho \) = Rock density (t/m³)
- \( \phi \) = Friction angle of rock joint (°)
- \( i \) = Angle of roughness of rock joint (°)
- \( d \) = Blasthole diameter (mm)
- \( \sigma_c \) = Compressive strength of rock UCS (MPa)
- RQD = Rock quality designation (%)
5.3.2 Size of Blasthole

The choice of the blasthole diameter must satisfy two opposing requirements:

- Drill and blasting cost efficiency (larger diameter holes are cheaper per unit volume (BCM) drilled).
- Fragmentation - smaller diameter holes give finer fragmentation – important if what you blast is what you sell, i.e. aggregates, or for ore blasting where fragmentation can replace the use of primary crushers. In the case of waste or overburden blasts – size to suit excavation equipment used.

Longer holes can be cheaper in terms of costs per unit volume (BCM), but may give insignificant improvements to fragmentation and can cause an increase in overbreak, flyrock and secondary blasting, etc. Larger diameter blastholes require a larger bench height. But if the hole is too long, the direction control will be poor, while if it is too short, just a small (lower) part of the hole gets loaded (because of stemming length requirements) and the fragmentation (especially in the uncharged portion of the blasthole) is then poor.

A general empirical rule is based on the bench height (H) to the blasthole diameter (d) ratio:

\[ 200 > \frac{H}{d} > 60 \]

From an economical point of view:

- Unit drilling costs per BCM reduce as the diameter and length of the blasthole increase
- Initiating costs increase with reducing hole diameter, as also the loading costs thereof
- Sensitivity of the explosives in smaller holes may result in the explosive energy not being fully utilised because of the reduction in the speed of detonation
- Secondary breaking is very expensive, but the savings on the drilling costs is in general much larger when larger diameter holes are drilled – this is particularly relevant where waste blasting occurs – there is no need to generate a very fine product size
- Larger holes can result in an unfavourable layout in comparison to the jointing of the block. That gives poor final wall profiles and fragmentation.
5.3.3 Parameter Empirical Relationships

The bench geometry of a blast design is defined in Figure 5.7. Initially, blast design estimates for these measurements can be described as multiples of the drilled burden \( B \) in terms of the blasthole diameter \( d \):

**BURDEN**
From 20\( d \) to 45\( d \). Approximately 20\( d \) in difficult conditions, to 45\( d \) where the rock can be broken easier. In relation to the calculated powder factor then:

\[
\begin{align*}
B & \approx 20d, \text{PDF}>0,9\text{kg/BCM} \\
B & \approx 21d \text{ to } 30d, \text{PDF}>0,6\text{kg/BCM} \\
B & \approx 45d, \text{PDF}>0,3\text{kg/BCM}
\end{align*}
\]

Note that the above approximate range of values of \( B \) are estimates only used TO TEST THE BALANCE OF THE DESIGN. They should NOT be used in determining \( d \) or \( B \) alone.

The following empirical rules CAN be used in the design, both to design the round and TO TEST THE BALANCE OF THE DESIGN:

**SPACING**
\( S \approx 1,2B \) To guarantee that the charge breaks out in the required direction. \( B<S<1,5B \) will work with certain initiating sequences (where the spacing and burden during initiation \( S' \) and \( B' \)) can differ significantly from \( S \) and \( B \)). Using this rule, the results will be satisfactory irrespective of the initiation \( S' \) and \( B' \) values.

**STEMMING LENGTH**
\( T \approx B \) As an estimated starting point to avoid the charge breaking through to the surface and causing fly rock. \( T \approx 1,1B \). To avoid fly rock make \( T \approx 1,5B \) (for blasting in built-up areas or near mine equipment). Alternatively, a stemming length of between 15-30\( d \) can be used, with 20\( d \) being typical for competent rock.

**CHARGE LENGTH**
\( L \geq 3B \) The more rock that is in contact with the explosive column (charging length) the better it is for fragmentation. In longer blastholes the charge can be decked charges - next to hard layers or where the bench is very high.

**SUB-DRILLING**
\( U \approx 0,5B \) Excavating tempo is affected by bad toe conditions caused by a combination of contaminated explosive in the bottom of the blasthole and more difficult breaking. Sub-drill is critical in a solid
rockmass and vertical blastholes – but NOT used in overburden blasts above coal seams. In inclined holes the toe effects are reduced and sub-drill can be reduced. Since sub-drill does not contribute to the broken volume (BxSxH), but does contain explosives, it will have a direct influence on PDF (referred to as the “actual” PDF) and cost per BCM broken. The charge mass (\(M_u\), kg) in the sub-drill can be found from;

\[M_u = 0.39 \overline{M}_c B\]

where;

\[\overline{M}_c = \text{Average charge mass per linear meter blasthole (slightly higher than } M_c \text{ when hydrostatic pressure increases density of pumped explosives in the toe region) (kg/m)}\]

**BENCH HEIGHT**  
\(H \approx T + L + U.\) In most strip mines the bench height is dependent on the depth of the coal seam and the digging depth of the dragline (on overburden benches). In coal, the bench height is usually limited by seam thickness. For terrace operations, bench height is selected either according to machine productivity limitations and safety (in waste) or depth of coal between waste. Note also the limits imposed on \(H\) by blasthole diameter \(d\).

### 5.3.4 Determining Design Burden

The empirical design rules are thus established, now the explosives must be distributed in the rock according to the powder factor (PDF kg/BCM). For vertical holes;

\[PDF = \frac{M_h}{BSH} \quad \text{or} \quad PDF = \frac{M_h}{BS(H+U)}\]

Either form can be used, dependant on whether the “technical” PDF (sub-drill ignored) or “actual” PDF (with sub-drill) is required. For blastholes inclined at an angle \(\beta^\circ\) to the horizontal (sometimes used to give safer bench faces and/or to produce the final bench slope and to reduce the toe and back break) a longer blasthole therefore a larger weight of explosives in the hole is needed. In this case:

\[PDF = \frac{M_h}{BSH \sin \beta} \quad \text{or} \quad PDF = \frac{M_h}{BS(H+U) \sin \beta}\]
Where:

\[ M_h = \text{Explosive loading per blasthole (kg)} \]

Other variables as defined previously.

Now if:

\[ A = \text{Burden to spacing ratio (B/S)} \]
\[ Y = \text{Stemming to burden ratio (T/B)} \]
\[ M_c = \text{Explosive charging per meter length of hole (kg)} \]

Then to solve initially for B:

\[
B = -\left( A \cdot M_c \cdot Y \right) + \sqrt{\left( A \cdot M_c \cdot Y \right)^2 + 4 \left( H^2 \cdot M_c \cdot A \cdot PDF \right)}
\]
\[
2 (H \cdot PDF)
\]

Using the appropriate expression for H depending on whether sub-drill is included in the PDF and/or inclined holes are used.

Once burden is determined, spacing can be found from the empirical relationships introduced earlier. At this point it is good practice to refer back to the remaining empirical relationships and substitute for B. The conceptual design is acceptable if all the empirical relationships are satisfied. Other more sophisticated design methods can be used, but the dominant effect of the rock mass characteristics should always be taken into account, irrespective of the method finally used to design the blast.

### 5.4 Blasthole Layout and Initiation

The blasthole layout and initiation timing used in a blast is often variable and dependant on the blast objectives defined earlier, and other considerations including:

- length and shape of the free face
- position of the ore on the block
- existence of a final wall along one or more sides of a block
- desired shape of the final muckpile
- need to control vibration and air blast
- required fragmentation.
Usually square or rectangular blasthole layout patterns are drilled because it is easier to mark and drill, especially on smaller operations. The staggered pattern however, delivers a better distribution of explosives energy within the rock mass and gives thus better fragmentation, because of the fact that the bench surface is better covered. But unless surveyors or GPS are used to mark out each blast round it is easy to position holes incorrectly and one such mistake will be the cause of poor results. Therefore, the staggered pattern is mainly used on the larger coal strip-mines, often in conjunction with mine operations planning software. Figure 5.8 compares the two basic layouts.

Seen in plan on the surface of the bench, the fractured areas around the blastholes can be represented as circles. It is logical to assume that every point on the surface must fall within at least one of these fracture circles for effective fragmentation to occur. Figure 5.8 contrasts the arrangement of square drilling patterns with that of staggered drilling patterns, for a Spacing/Burden ($S_b$) ratio of 1.25:1.

The staggered pattern produces a more uniform distribution of fracture circles and thus more even fragmentation in the muckpile for the same powder factor. For optimal fragmentation a total coverage is necessary, where there is no (or little) uncovered areas (out of a cost point of view no overlapping of fragmented areas).

The greatest potential for good breaking with the most extended drilling pattern lies in using a staggered pattern having an S/B ratio of between 1 and 1.5. The square layouts all give significantly less coverage. Table 5.4 summarises the percent bench cover associated with the various B:S options.

Real as the benefits of staggered patterns may be, they are less evident in highly fractured ground where the fracture planes seriously hamper the development of radial fractures. Also, with small diameter blastholes in high benches, drilling inaccuracy and hole deviation can result in the pattern at the toe being unrelated to the laid out pattern on the bench. In these conditions, therefore, the merits of staggered patterns may be outweighed by the convenience of drilling square patterns.

A further factor which can affect the results of the blast design, especially degree of fracturing, is the simultaneous firing of neighbouring holes. In the technique of presplitting this interaction is advantageous, as shown in Figure 5.9. Cracks developing parallel to the line between closely spaced simultaneously detonating holes are promoted, while those at right angles to it are suppressed.
This phenomenon is obviously detrimental to fragmentation, as crack formation in the burden is inhibited, and large unbroken beams of rock are thus projected onto the muckpile. Optimum fracturing is therefore obtained by ensuring that each blasthole fires independently, and this can be achieved in two ways, namely:
Table 5.4 Percent cover square versus staggered blasthole layouts

<table>
<thead>
<tr>
<th>SPACING TO BURDEN RATIO (S:B)</th>
<th>PERCENTAGE COVER (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Fragmentation radius (optimal for staggered pattern)</td>
</tr>
<tr>
<td>1</td>
<td>0.49</td>
</tr>
<tr>
<td>1.15 (Equatorial)</td>
<td>0.56</td>
</tr>
<tr>
<td>1.25</td>
<td>0.61</td>
</tr>
<tr>
<td>1.50</td>
<td>0.71</td>
</tr>
<tr>
<td>2.00</td>
<td>0.88</td>
</tr>
</tbody>
</table>

Figure 5.9. Interaction of stresses from closely spaced blastholes fired simultaneously.

1. Initiate each blasthole in sequence
2. Arrange that simultaneously initiated blastholes are far enough apart to prevent mutual interaction between their stress fields. This can for example be achieved by "chevron" firing or by the selection of surface and down-hole delays that result in unique firing times.

It is usually considered poor practice to initiate blastholes individually in sequence, although this technique finds application where blasting vibration considerations impose a restriction on the charge mass detonated per delay. Normally, the best solution is to resort to chevron firing, which is simply achieved and in most cases gives excellent results.
The initiating order that is chosen depends on several layout principles, the three most important ones being:

- Free-face position during the blast and the inter row (delay) timing
- Maximising of the free-face area during the explosion
- The shape of the muck pile required and it’s footprint.

5.4.1 Free-face position during blast and inter row (delay) timing

Holes are blasted in the order that is established through the position of the free-face as it exists in the different stages of the blast. This concept is referred to as burden relief – where each row has only the design burden to break to the free-face position. Where burden relief is poor, coarser fragmentation, an uneven floor, more fly rock, a taller muckpile and more final wall damage is seen.

Burden relief is not shorter delay between holes or rows. Shorter delays can be used to achieve finer fragmentation or to make a muckpile higher for more efficient loading with a face shovel. There is a point for each rock type and blast geometry, however, where a delay can become too short and the burden is not relieved before the next hole fires.

Different rocks respond differently to the energy released from the detonation of a charge in a hole. Softer rocks will absorb more energy by plastic deformation and the burden will be released more slowly. Harder rocks will absorb some elastic energy, but will begin to fail more rapidly and the burden will therefore be released more quickly. This response can be determined using laser profilers to measure the front row burdens and high-speed cameras to time and track the burden after hole detonation. Table 5.5 summarises typical burden response times.

The only exception to the concept of burden relief is with buffer blasting which is sometimes used at coal strip-mines, where the blast gets ‘buffered’ by the previous pile of blasted burden. Buffer blasting prevents excessive loss of bench height (in comparison with a ‘normal blast’), when a high working bench is necessary for the dragline and also gives an improved highwall condition in poor ground conditions.
Table 5.5 Typical burden response times

<table>
<thead>
<tr>
<th>Rock</th>
<th>Minimum response time per meter of burden</th>
<th>Shortest delay per meter of burden for a higher muckpile</th>
<th>Shortest delay per meter of burden for a flatter muckpile</th>
</tr>
</thead>
<tbody>
<tr>
<td>Soft sandstone</td>
<td>8 ms/m</td>
<td>16 ms/m</td>
<td>24 ms/m</td>
</tr>
<tr>
<td>Hard Sandstone</td>
<td>6 ms/m</td>
<td>12 ms/m</td>
<td>18 ms/m</td>
</tr>
<tr>
<td>Granites</td>
<td>5 ms/m</td>
<td>10 ms/m</td>
<td>15 ms/m</td>
</tr>
<tr>
<td>Massive dolerite</td>
<td>4 ms/m</td>
<td>8 ms/m</td>
<td>12 ms/m</td>
</tr>
</tbody>
</table>

The initiation delay is the product of design burden and the delay per meter.

In cast-blasting, which is the direct opposite of buffer blasting, the maximum amount of the burden is thrown over the pit to the waste side by the explosion. In this case a small burden is used and the timing of the round is of most importance. It is important to avoid the effect of ‘choke’ in which the ejected (moving burden) of one row accelerates into the moving burden of the previous row (which is decelerating). If this happens, the throw of the rock gets hampered. In this type of blasting the drop in height of the bench is high since most of the rock is thrown far.

5.4.2 Maximizing of free face area during explosion

Holes without a large free face area must be delayed to achieve the maximum free face area (or to let develop) during blasting. A good example of this, is with the box-cut (the first excavation at a new strip mine where a cut is develop and into which the following strips’ waste will be moved as soon as the coal is loaded out). The box-cut has typically only one free face – the ground surface. Once established, the box-cut vertical face provides the ‘conventional’ bench vertical free face for subsequent blasts.

5.4.3 The shape of the muck pile

The shape of the fragmented muck pile can be controlled by the initiating sequence, to achieve certain types of results that suit the
types of loaders, bottom bench (footprint) area available and other production limitations. Chevron tie-up sequences are typically used for both square and staggered layouts of drill holes.

A closed chevron gives a very steep muck pile that suits rope- or hydraulic loaders very well and which can dig high faces easily. The steep pile forms because of collisions of ejected burdens from both sides of the chevron that leads to a loss in momentum in the moving rock and displacement is thus less. Since the front end loader can’t dig high rock piles safely, open chevrons are used which promote the throw and displacement of rocks over a wider area (and are easier to tie-up and give less toe problems). Note that there must be sufficient free face and bench footprint area available to use an open chevron. Alternatively, it is possible to create an open chevron from a single free-face, but the results are often poor. Figure 5.10 illustrates these chevron types and muck pile results.

![Figure 5.10](image)

Figure 5.10 Chevron initiation tie-up options (a) closed, (b) open

A variety of chevron tie-up patterns can be used depending on the true excavation conditions, geology and blasting layout as shown in Figure 5.11. Take note that as soon as the coupling pattern deviates from the straight line (in other words row on row V0) then the burden and spacing change.

Considering a square pattern of holes as depicted in Figure 5.10(a), it is evident that several different chevrons can be drawn through the pattern. From a single front row hole, the chevron which intersects the nearest hole in the next row, i.e. the hole immediately behind, defines the "V0" chevron. If the angle of the chevron is flattened so that it extends through the next nearest hole, this defines the "V1" chevron, and so on. Chevrons through staggered hole patterns are similarly defined but look rather flatter, as in each case the sideways distance is increased by half the spacing. This is also shown in Figure 5.10(b).
The effectiveness of these chevrons are summarised as:

**SQUARE V0:** Not a chevron (normal row upon row blasting).

**SQUARE V1:** Good fragmentation, tight (solid) muck pile, good for shovel productivity, if the bottom bench area is limited, this pattern is a good option.

**SQUARE V2:** Very good, specially when the spacing to burden ratio is between 1.3 and 1.5. Suits high face (>20m) where rope or hydraulic front shovel excavation height is less then 20m (because of the further lateral displacement of the rocks).

**STAGGERED V0:** Poor fragmentation and displacement, round inclined to choke and usually causes a lot of damage to the back wall because of the reflection of the shockwave.

**STAGGERED V1:** Very good, tends to promote fragmentation and drop height, good for
front-end-loader. Minimal damage to the back wall.

**STAGGERED V2:** Less effective than V1 because of the larger real burden (in comparison with calculated value), but is often used to join the last two or three rows because it controls excessive over breaking on the last row of holes, due to the small initiation burden. (But the V2 rows are inclined to displace the rock far and can thus cause problems where the benches below are not wide.)

Higher order chevrons (3, 4) are not as advantageous and show usually poorer blasting results because of the decrease in effective burdens and increase in spacing, as compared with the conceptual design values. Despite higher initiation spacing and burden ratio’s, the resulting convex bi-planar free-face actually makes V0-V2 chevrons more effective blasting due to the interaction of the explosion shock front and the face shape produced as each row initiates (see Figure 5.12). The bi-convex face shape makes radial crack formation more efficient (since rock strength is less in tension than in compression and the compressive shock front reflects as tensile wave).

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**Figure 5.12** Convex free-face formed by change from *designed* (B, S) to *initiation* (B’ S’) burden and spacing by adopting higher order (V2) chevron on a square pattern layout.
5.5 Optimising a Conceptual Blast Design

The main requirement of blast design process in mining is the provision of a framework within which blast improvement may take place. This highlights the important distinction between blast design and blast optimisation. Blast design is concerned with the design of blasts according to clearly defined objectives and criteria.

In order for blast improvement to take place, it is essential that some form of performance measurement is carried out. Because blasting impacts in some way on most of the downstream operations in mining and processing, it is not possible to define a single performance index which completely defines blast results. A number of indices are required to fully gauge performance.

Mining operations are specifically structured towards the achievement of high production tonnages with a minimum of disruption either at the bench face or at the primary crusher or waste tip. In the absence of quantitative blast measurement, mine management relies on drill and blast cost control and production targets measured in terms of tons/drill, tons/shovel and tons/truck, on a shift or period basis, to assess mine performance. Whilst these measures are necessary for evaluation of machine performance, they may not be the most appropriate measure of mine performance. Ideally, an approach of total mining cost to the optimisation of blasting should be used, but the problem remains however of obtaining reliable cost measures for each of the mining components shown in Figure 5.1.

The following alternative factors provide a basis for assessment of blast performance in surface mining:

1. Occurrence of oversize in the blast muckpile causes a major disruption to material flow and excavator productivity, as reflected in records of secondary breakage in the pit.

2. Fragment size distribution, is important in terms of mean fragment size (which may be expected to influence ease of handling and rate of material flow within the mine) and the weight percentage of oversize material requiring isolation from the mainstream material flow because of potential blockages at the tip. The amount of fines generated by blasting will also be important in coaling operations.

3. Excavator productivity is an important economic index and a potentially valid indication of blast performance since digging efficiency is influenced by both fragment size and muckpile condition (looseness and shape). Fragmentation in this context infers both mean fragment size as well as the amount of oversize.
requiring sorting. The degree of difficulty experienced in digging at grade level is also incorporated in this index.

4. Crusher performance, measured in terms of material flow (delays caused by blockages), maintenance, and monitoring of power consumption.

5. Coal recovery, measured in terms of the ROM feed and percentage of (reject) fine material.

6. Initial observations of the blast, followed by regular inspections during the course of digging, together with liaison with operator’s provide useful information concerning blast results and should be documented. Figure 5.13 illustrates typical parameter observations.

Consider the following parameters;

BROKEN ROCK
Too compact, difficult to load, poor production from the loader. Too much fly rock and too much clearing up must be done before the high production area of the muck pile is loaded. The rock pile can also be too flat for the particular type of loader used, or too spread-out (rock falls on benches below blasted bench).
TOE OF BENCH
Hard because of insufficient sub-drilling or under-charging in toe region. Poor floor because of too much sub-drilling and explosive sub-grade. Low tyre life, difficult to load from the floor and hard to control floor level – ubiquitous in surface mining.

HIGHWALL
Over-charging or excessive burden - badly broken and dangerous. Dangerous to load under and possible failures. Under-charging - saw-tooth blast effect. Difficult to load, possible failures and loader operator often scrapes face to clean it.

FRAGMENTATION
Large rocks need secondary blasting, disrupting the production cycle and is costly. Poor fragmentation (big rocks) result in too much wear and tear of the loading machine, ground engaging tools, poor loading productivity and damages the truck body.

FLY ROCK
Craters are seen in the muck pile and beyond, because of the fall of rock ejected from the collar region. It is dangerous and will necessitate lot of cleaning work. Poorly designed blast, stemming too short or burdens excessive.

5.5.1 Fragmentation Analysis and Optimisation
For blast design and optimisation purposes it is necessary to predict and measure fragmentation, especially where there are limits on the size of broken rock produced – typically loading or hauling equipment limits on productivity, if the blast is to produce a final saleable product or where fines are problematic (coal blasting). There are three parameters that are especially important namely:

- The average size ($x_{avg}$), or the grizzly (sieve) size at which 50% according to weight of the broken product will pass through

- The percentage of rock that is larger than the maximum size allowed and that will have to be secondary broken.
- The percentage of product that is smaller than the minimum required size allowed, cannot be processed or sold and will be treated as waste.

Figure 5.14 illustrates these concepts, based on a typical fragmentation curve. Very often, fragmentation is quoted only as a mean fragment size (or $d_{50}$) and a uniformity index. As the uniformity index decreases, the range of fragment sizes increase (and the curve in Figure 5.14 flattens). In other words, a larger proportion of fines and coarser material occurs. As the uniformity index increases (and the curve in Figure 5.14 steepens), the range of fragment sizes decreases.

![Fragmentation Curve](image)

Figure 5.14 Typical fragmentation curve for blasted rock

These measures assume that fragmentation will fit a curve that is similar to the one in Figure 5.14 and is the fundamental assumption of the KuzRam model used to predict fragmentation. In practice, this has generally been found to be the approximate case where fragmentation has been measured – although as will be discussed later, for softer rock types this is not strictly correct.

Several empirical formulas are designed to meet these requirements and if correct constants are used, relatively accurate predictions can be made. The Kuznetsov equation has been used with success in South Africa. From this equation the average size ($d_{50}$) can be established if the rock factor is known;
\[ d_{50} = 10 \left( A_f \left( \frac{E_f}{100} \right)^{-0.8} M_c^{0.167} \left( \frac{115}{RWS} \right)^{0.633} \right) \]

Where:

- \( d_{50} \) = Average fragment size (mm)
- \( A_f \) = Rock factor
- Other variables as previously defined

The uniformity index \( I_u \) can be determined from:

\[
I_u = \left( 2,2 - 1,4 \frac{B}{d_e} \right) \left( \frac{1 + A^{-1}}{2} \right)^{0.5} \left( \frac{L + T + U}{L} \right) \left( \frac{L}{d_e} + 0.1 \right)^{0.1}
\]

Where:

- \( d_e \) = Explosive diameter (mm)
- \( L \) = Charge length above toe level (m)
- Other variables as previously defined

Rock blastability index (RBI) can be used as a quantitative measure of the rock factor \( A_f \) and can be used as input into the rock fragmentation model. The conversion factor between RBI and \( A_f \) will be site specific. Experience indicates the following conversion to be generally effective in hard rock:

\[ A_f = 4 + 0,1RBI \]

This correlation will differ from site to site and it is best, on large-scale operations, to determine the site constants by carrying out fragmentation measurements from a number of blasts.

Alternatively, the drilling index can be used to obtain a value for \( A_f \) in the Kuz-Ram equation. Drilling is an effective way to assess rock blastability because a number of drilling variables are measurable on drilling rigs with real-time drilling monitors. An empirical relationship has been obtained for the drilling index such that:

\[ A_f = 66.9 \left( \frac{P_s d^2}{W_\omega} \right)^{0.4852} \]
Where;

\[ P_s = \text{penetration rate (m/hour)} \]
\[ W = \text{pulldown pressure on the bit (N/m}^2\text{)} \]
\[ \omega = \text{rotation speed of the bit (rpm)} \]
\[ d = \text{diameter of the blasthole (m).} \]

As with the RBI, the relationship constants need to be established for each blasting operation.

The average block size and uniformity index is not all that is of importance because if the material has also a large size distribution (between largest and smallest blocks), then there may be significant secondary blasting required even though the average size may be within limits. The Rossin-Rammler equation can be used to predict this size distribution;

\[
R = \exp\left(\frac{x}{x_c}\right)^n \quad \text{or} \quad x = -x_c \ln(R)^{\frac{1}{n}}
\]

Where:

\[ R = \text{Mass fraction of fragmentation larger than } x \ (\text{cm}) \]
\[ x = \text{Fragment size in (cm)} \]
\[ x_c = \text{Characteristic size (cm)} \]
\[ n = \text{Rossin-Rammler exponent} \]

\[
n = \left(2.2 - 14.2 \frac{B}{d_c}\right) \left(1 - \frac{\omega}{B}\right) \left(1 + \left(\frac{I - I_l}{2}\right)\right) \frac{I}{H}
\]

Where:

\[ \omega = \text{Standard deviation of blasthole spacing (m)} \]

Other variables as previously defined

and

\[
x_c = \frac{d_{50}}{\left(0.693\right)^n}
\]
As allude to earlier, the Kuz-Ram fragmentation distribution curve is likely to under-estimate the percentage of fines in the blasted product — especially where coal or other soft materials are concerned. The form of the Kuz-Ram model is governed by the distribution of pre-existing fractures and discontinuities in the rock mass. In the immediate vicinity of the blast hole the stress field is compressive. Further away from the hole the stress becomes tensile, causing radial fracture extension. This model, in its classical form, does not incorporate the mechanism of fragments created by the compressive/shear failure of the rock and thus it generally underestimates the proportion of fines (fragment sizes less than 10-20mm). For relatively hard rock this introduced error is insignificant. However, in the case of softer rock, where the size of crushing round the blasthole is greater, it is necessary to exercise caution in interpretation of the results of fragmentation prediction, and to apply alternative models for fines prediction.

Such an approach is the Two Component Model (TCM), based on the Kuz-Ram model, but splitting the total product mass into two fractions ($F_c$) representing the percentage of product generated by tensile failure and compressive failure. Thus the fragment size distribution becomes:

$$R = \left[ 1 - \left( 1 - F_c \right) e^{-0.693 \left( \frac{x}{A} \right)^B} - F_c e^{-0.693 \left( \frac{x}{C} \right)^D} \right]$$

Where;

- $A, C = \text{mean fragment size in the tensile and compressive failure regions (respectively) (cm)}$
- $B, D = \text{uniformity coefficients in the tensile and compressive failure regions (respectively)}$

Other variables as previously defined

The easiest approach to determining the values for mean fragment sizes and uniformity indices, is to fit the measured fragment size distribution to the equation given above. To determine $F_c$, blast chamber testing is required, or observation of the volumetric extent of crushing (generated from compressive and shear failure) around a hole in the last row of a blast, expressed as a fraction of the blasted volume (generated by tensile failure).
5.5.2 Measuring Fragmentation

Actual measurements of fragmentation are often done today using the edge technique to analyze photographs or video of a blast to determine the actual size distribution. While these techniques may not always be totally accurate for the fines fraction they do provide a good overall assessment of the blast fragmentation. When photographs or video are taken at different times during the digging of the blasted rock a good idea of the degree of fragmentation throughout the blast can be obtained.

A number of powerful imaging codes have become available with which to measure fragmentation from digital photographs. An example is the WipFrag System and the Split System that both use digital images of fragments to measure fragment size distribution.

The image quality of the photographs is very important because the code delineates fragments by detecting dark and light areas. Therefore light source and light angle must be consistent for each image taken. The following points need to be considered when photographing a muckpile:

- Sampling strategy needs to be developed so that the results statistically represent the product as a whole. Very often muckpiles have coarser material on top and finer material below – poor results can be expected if just photographing the muckpile after a blast due to throw segregation effects.

- Most image processing systems like WipFrag or Split allow the results from several images to be merged and averaged into a single sample. It is advisable to take several shots on a muckpile or at tipping points.

- No single fragment should occupy more than about 20% of the width of the image. Fines can be more accurately resolved by zooming some images and merging them with other images.

- Each image must have a scale in it for reference.

Digital imaging systems have been shown to be quick and very accurate under the correct photographing conditions. This makes the routine measurement of blast fragmentation a practical solution to controlling blast results and to calibrate the $A_i$ value in the Kuz-Ram or similar prediction models.
5.5.3 Blast Monitoring

Most large surface mines employ experienced blasting personnel who have a well developed sense of the ground reaction to blasting on a local scale, but are not always able to predict with total confidence the results of a blast or why a particular blast fails to produce the required results under similar field conditions.

One of the main problems is analysing what takes place over the second or two that the blast occurs. Any number of factors could come into play which effect the final result, the majority of which are purely transient in nature. The only parameters that are more permanent (for analysis) are the final product parameters such as fragmentation, blast profile and back break together with the initial conditions such as hole depth, inclination, charge and position relative to other holes in the blast.

High speed photography has become a powerful diagnostic tool in analysing these transient effects and illustrating the likely source of blasting deficiencies. Data from such analyses is classified into two groups;

- Qualitative
  - First rock movement
  - Firing sequence of holes
  - Confinement or blowout of stemming
  - Shape of primary burden movement
- Quantitative
  - Detonator firing time and delay timing
  - Duration of escaping gasses
  - Acceleration, direction and velocity of rock
  - Ground swell velocity
  - Confinement time of gasses
  - Throw of fragmented material

A comprehensive collection of this data, together with the initial and final conditions can be used to recognise;
• Cause of misfires
• Poor loading of explosive
• Poor timing between holes or rows
• Inadequate forward burden relief
• Minimum rock response time
• Massive ground movements
• Source of flyrock or oversize
• Top movement characteristics (cut-off potential)

A further illustration of blast monitoring is in the analysis of blast induced damage to coal seams. Damage to coal is defined as any blasting technique which causes excessive dilution, cracking, pulverisation or mass movement of the target coal seam, that can reduce the recovery of the coal. The main objective of strip mine operators is to strive for maximum recovery at the lowest possible mining cost. Monitoring instrumentation can be extended to incorporate, in addition to high speed cameras;

• Multi-channel continuous VOD recorders
• Strain gauges, accelerometers and pressure transducers
• Laser survey equipment (face profilers) and blasthole survey systems
• Seismographs for vibration/airblast analysis
• Synchronous video cameras for time studies
• PC based video imaging systems for fragmentation analysis

The information extracted from a single instrument can only be used in a limited way and incorrect conclusions can easily be drawn. Typical examples would be;

• Explosive performance/poor mix
• Water effects
- Poor hole alignment, etc
- Detonator scatter
- Interburden relief timing

Using a combination of the equipment listed above, several factors contributing to poor blast results, or coal damage, as the case may be, may be recognised, including:

- Too large a burden
- Drilling into coal or insufficient cover left above seam
- Insufficient stemming
- Decking with small charges
- Primer location
- Waterlogged conditions around coal seam and hole
- Poor timing

5.6 Cautious Blasting

A blast design can be optimised according to minimal total mining costs, muck pile shape and throw, excavation difficulty (diggability), fragmentation, etc. Just as important is the effect of the blast design on the mine, private property and the local environment. Three factors that must be taken into account with blast design are:

- Fly rock
- Airblast
- Ground vibration

The effect of each of the above blast-induced problems is primarily related to distance from the blast, amount of explosive initiated per delay and the type of structure concerned. The response of structures to a blast depends on their design (and users). A house cannot be placed in the same response category as say a warehouse, similarly powerlines, silos, bridges and the pit slopes themselves will all react differently, and the consequences of that reaction is what dictates the acceptability or limiting criteria applied in blast design.
5.6.1 Control of Flyrock

Fly rock can be generated from two main sources:

1. From a face that is under burdened.

2. From the collar of a blasthole when a hole is overcharged, the stemming is ineffective or the holes fire out of sequence.

Assuming that the conceptual blast design is suitable and well balanced and that the burden and stemming length relationships (T, B and Y) are correctly applied, some modification of the design will be required to reduce fly rock from either of these sources.

INSUFFICIENT BURDEN

An important and often neglected source of fly rock is from the free faces of a blast. Fly rock occurs from this source when the face burden is too low and/or when the face has been badly damaged from a previous blast.

To avoid fly rock from the free face, the scaled burden \(B_s\) at any point should not be less than \(0.7\ m^{36/32}\ kg^{-1/2}\). This can be determined from laser profiling of the face. Where the scaled burden is less than the recommended value, stemming should be increased in that region of the hole.

INSUFFICIENT STEMMING

Stemming length and stemming performance are the critical factors here. Two approaches are possible, one based on hole diameters and one based on scaled depth of burial to determine stemming length.

A good general rule for designing stemming length is to use at least a length equivalent to 10 hole diameters \((10d)\). This can be modified according to conditions as given in Table 5.6.

An alternative approach to determining the stemming length \(T\) is to use the scaled depth of burial \(D_s\), based on hole diameter and explosive energy. The equation for the scaled depth of burial has been established empirically from crater tests;

\[
T = D_s \left[8M_e d_e\right]^{0.33} - 4d_e
\]

Where the variables are as previously defined and where the scaled depth of burial is determined as in Table 5.7.
Table 5.6  Minimum recommended stemming lengths

<table>
<thead>
<tr>
<th>Condition</th>
<th>Minimum stemming length (in terms of hole diameter d)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hard capping layer</td>
<td>&gt;15d</td>
</tr>
<tr>
<td>Normal blasting</td>
<td>25&gt;d&gt;20</td>
</tr>
<tr>
<td>Much reduced fly rock from collar</td>
<td>30&gt;d&gt;25</td>
</tr>
<tr>
<td>Almost no flyrock from collar</td>
<td>&gt;30d</td>
</tr>
</tbody>
</table>

Note

As stemming length increases the fragmentation in the collar area of a blast will become coarser whilst the fragmentation in the toe area will tend to become finer. Stemming lengths of less than 10 hole diameters can result in poor fragmentation and heave results because of energy loss through the hole collar.

Table 5.7  Scaled depth of burial required

<table>
<thead>
<tr>
<th>Scaled Depth of Burial ($D_s$)</th>
<th>0.7</th>
<th>1</th>
<th>1.2</th>
<th>1.5</th>
<th>1.7</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hard rock</td>
<td>much fly</td>
<td>normal fly</td>
<td>little fly</td>
<td>no fly</td>
<td>no surface expression</td>
</tr>
<tr>
<td>Soft rock</td>
<td>much fly</td>
<td>much fly</td>
<td>normal fly</td>
<td>little fly</td>
<td>no fly</td>
</tr>
</tbody>
</table>

Only one overcharged or short-stemmed hole in a blast can create enough fly rock to cause unwanted damage or injury. The design stemming length should be checked in the field – with bulk explosive systems, this is straightforward, but with hand loading, overcharging can occur more readily. Additional causes and remedial measures include:

- Too short stemming length ($T$), jointed or fractured/broken rock around the collar from previous sub-drill. Correct stemming length or use deck loading in the stemming area if poor fragmentation in this region is a problem.

- Use longer inter-row delay times – short delay time between rows gives too much vertical displacement of rock (rock flies upwards
into each other). This condition is typical of cast blasting however, larger loading is used with which to cast the material to the other side of the strip (refer later section)

- Stemming material. Use good quality stemming material. Stemming material needs a high shear strength and density. This is best provided by a crushed screened aggregate. The fragment sizes should be about 0.1d.

- In strip mining overburden blasting, soil is removed to a depth specified by the Regional Director, but in-situ softs are nevertheless left behind. Leaving it in place acts as a stemming blanket, but it does NOT form part of the stemming length calculation. Soil and weathered material have very little strength and good-quality stemming in soil will not remain in place.

Take note that a poor blast design (unbalanced according to empirical design rules) will also cause a severe air blast problem – especially where blasthole diameters are very large in comparison to bench heights.

5.6.2 Airblast

Air blasts is the cause of most complaints regarding blasting and to increase the extent of the problem, the public is apt to confuse air blast and ground vibration (in most instances ground vibrations get blamed when air blast is the real problem).

There are three basic causes of airblast:

1. The detonation of uncovered and/or initiating systems (e.g. Cordtex) and lay-on charges for secondary blasting.

2. Blow-outs because of no or too little stemming, bad timing or not enough front row burdens

3. The movement of rocks during blasting and the vertical component of the ground shock waves (both are very small effects in comparison with the previous two).

Again, a well-balanced conceptual design will not generate significant airblast. However, coal blasts in strip mines are usually problematic
since the blasthole diameter is often large compared to seam thickness and stemming lengths consequently less than required.

The following formula can be used to calculate the maximum safe charge per delay or distances, based on the USBM limits of 128dB;

$$\frac{D}{\sqrt[3]{W}} \geq 600 \quad (\text{unconfined}) \quad \text{or} \quad \frac{D}{\sqrt[3]{W}} \geq 100 \quad (\text{confined})$$

Where confined refers to the mass of explosive (W, kg) outside the blasthole and confined refers to a normal (well designed) blast with explosives in the blasthole, and D the distance in meters from the blast.

Methods of control include;

- Cover all detonating cord or better still use noiseless shocktube or electric trunk lines
- Limit explosive per delay (use intra-row delays on long rows or heavy charges)
- No blasting early in the morning because of temperature inversion (especially in winter) – this causes the airblast to be reflected back to the ground.
- No blasting if the wind is very strong (specially if the wind is blowing in the direction of residential areas).
- Blast ideally at peak noise time (11h00-13h00).
- Avoid short collars and fill the blastholes with enough stemming.

### 5.6.3 Ground Vibrations

When a charge is buried and detonated, most of the energy is used to break the rock, but rock is generally a competent transmitter of energy and vibrations move outwards from the source in the form of a seismic wave.

Three parameters are important in understanding and reducing vibration from blasts;

- The speed at which seismic waves travel through the rock or along the surface. This speed can vary between about 1000 –6000m/s depending on the rock type and the type of wave.
- Vibrations generated by blasting are complex and have variable amplitudes with a peak amplitude at some point in time. Normally, measurements are quoted in terms of velocity with the peak particle velocity or PPV being equivalent to the peak. In blasting, the peak amplitude is directly related to the charge fired per unit time and is inversely related to the distance from the blast.

- Blasting commonly generates frequencies between about 3 - 100 Hz. Frequency has an important influence on damage risk. Firstly, lower frequencies carry more energy at a fixed amplitude than higher frequencies. Secondly, certain frequency ranges can induce resonance effects in buildings that can amplify the vibration. In large overburden blasts, a frequency range of 4-30Hz is typical.

The ground vibration problems develop because of the peak particle velocity - PPV (mm/s) which is necessary to break rock. PPV’s can be classified in the following broad bands:

- <250mm/s - no fracture of rock
- 250-625mm/s - tensile stress failure of rock
- 525-2500mm/s - tensile stress failure and radial cracks form
- >2500mm/s - fragmentation of rock

In comparison with a PPV that would generate cosmetic damage to buildings which is generally about 50mm/s and more.

The following can be used as a general design rule:

\[
\frac{D}{\sqrt{W}} \geq 23 \quad \text{or} \quad PPV = 1143 \left( \frac{D}{\sqrt{W}} \right)^{-1.65}
\]

But this limit can cause still generate problems where conditions vary from ‘normal’ (e.g. old buildings (PPV_{max} 10-15mm/s), very dense rocks (hematite etc.), plaster and drywalls (PPV_{max} 15-20mm/s) or young cement (> 2 day’s old) and much higher frequency waves. Additionally, humans are more intolerable of ground vibrations than buildings, thus it is good practice to keep blasting PPV’s below perceptible levels.

Methods of control are based on sound blast design (charges and initiation) and the following factors should be considered;
• Small amount of explosive charge per delay (note, some delay combinations will still result in some holes going off together).

• Delay’s between the rows must not strengthen the shockwave (destructive interference – determine from site tests)

• With a chevron initiating pattern minimise the amount of holes with a zero time delay

• Blast parallel to the main joint set (it absorbs some of the shockwave energy)

• Use a pre-split or other highwall control drilling method to isolate the main blast-block from the rest of the rock mass

5.6.4 Highwall control

A safe highwall is of critical importance in terms of controlling rock fall hazards, especially important in coal strip mines where high vertical highwalls are used. Traditionally, pre-splitting has been the method of choice to achieve the safest highwall conditions. However, pre-splitting may not be appropriate to all geotechnical conditions and good quality post-splitting techniques may produce better results. In cases where very poor highwall geotechnical conditions exist, buffer blasting is considered as the most practical option.

Additionally, in strip coal mining, pre-splits are often used also for dewatering a block, to ensure a constant front row burden which assists in reducing coal seam damage and to achieve optimum results in cast blasting.

Some general rules apply to the pre-split designs used in large surface coal mines, specifically;

• Pre-split holes are drilled vertically, through the coal seam to the floor elevation and backfilled to the roof coal seam contact. They are typically unstemmed, except where hard homogenous rock is encountered and higher borehole pressures are required, when T<11d.

• Pre-split hole diameters range from 165 mm to 310 mm. With smaller hole diameters, more deviation is likely on higher benches whilst with larger diameter holes, poor results are typical in softer or heavily jointed rocks. Large diameter holes yield the lowest quality pre-split. Most often, pre-split holes are drilled at the same diameter as the main production holes.
- Pre-split holes are normally spaced at between 10d and 15d, the smaller value typical of strong, highly jointed rock, the larger for weak homogenous rocks.

- Pre-split powder factors of between 0.5 to 1.1 kg/m² face area are typical, with toe charges, however, better practice is to use a single charge in the hole, suspended alongside the hardest rock layer.

- Overcharging is problematic, in addition to damage in the new highwall, it leads to block movement (due to the coal contact acting as a shear plane), leading to opening of joints and difficulty charging the main blast.

In theoretical pre-split design, a fracture needs to be generated between boreholes, as shown in Figure 5.9. This is achieved by generating a borehole pressure sufficient to overcome the tensile strength of the rock, but not so high as to cause crushing of the rock – especially in large diameter holes. This is most often achieved by decoupling the charge from the rock mass. The approach to practical pre-split design is based on three inter-related parameters;

1. Hole diameter
2. Hole spacing
3. Charge per hole

Hole diameter is most often that of the production blast in coal strip mine overburden blasts. It can have some drawbacks in poorer quality rock as mentioned above, but it has the significant advantage of enabling the production drills to also drill the pre-split without equipment change-out.

**HOLE SPACING**

Ideally, the pre-split hole pattern spacing should not exceed twice the mean joint spacing. In terms of rock and explosives used, the hole spacing can be determined from;

\[
S = \frac{d(P_b + \sigma_t)}{\sigma_t}
\]

Where;

\[
S = \text{Pre-split spacing (m)}
\]
\[ d = \text{Blasthole diameter (m)} \]

\[ P_b = \text{Blasthole pressure generated by explosive (MPa)} \]

\[ \sigma_t = \text{Rock uniaxial tensile strength (MPa), typically 10-20\% of the rock compressive strength.} \]

Table 5.7 summarises the typical compressive strengths of strata encountered in the Witbank coalfields area.

Table 5.7

<table>
<thead>
<tr>
<th>Material</th>
<th>Compressive strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Mean</td>
</tr>
<tr>
<td>Siltstone</td>
<td>86</td>
</tr>
<tr>
<td>Layered sandstone</td>
<td>94</td>
</tr>
<tr>
<td>Massive sandstone</td>
<td>168</td>
</tr>
<tr>
<td>Micaceous shales</td>
<td>80</td>
</tr>
</tbody>
</table>

**CHARGE PER HOLE**

This is based on determining the blasthole pressure generated by the detonating explosive, followed by the effect of coupling to give the continuous charge length and number of cartridges (n) required (hence charge per hole, depending on the mass per cartridge);

\[
P_b = 1.255 \rho_e VOD^2 \left[ \frac{L_e}{H} \left( \frac{d_e}{d} \right) \right]^{2.4}
\]

and

\[
n = \frac{H}{L_e} \left[ \frac{d}{d_e} \right]^2 \times \left( \frac{7968P_b}{\rho_e VOD^2} \right)^{0.84}
\]

Where;

\[ d_e = \text{explosive cartridge diameter, about 0.2-0.6d} \]

\[ L_e = \text{Length of explosive cartridge (m)} \]

\[ VOD = \text{Explosive velocity of detonation (m/s)} \]
and other variables as previously defined

From a practical point of view the need to use pre-splitting should be critically assessed where the following poor geotechnical overburden and interburden conditions exist:

- Where the RBI is less than 40
- The Laubscher MRMR value is lower than 25
- Where the Bieniawski RMR value is lower than 55 and where joint spacing is very close (less than 0.2-0.5m)
- Where joints which daylight in the face have trace lengths greater than 5 m
- Where prominent joint planes are expected which cross the pre-split line at 5° to 15°.

Under problematic conditions, buffer blasting is often used - also in conjunction with presplit blasting. Hole diameter and depth are generally identical to holes in the main production blast. Where buffer holes are positioned directly above or near the crest of an underlying berm, they should not have any subgrade drilled, to reduce crest fracturing.

Spacing and burden tend be ½ to ¾ of the spacing and burden used in the main production blast. A buffer hole should be bottom loaded to conform to;

\[
d_c = 1.22 \sqrt[3]{\frac{2}{2} W}
\]

where;

\[
d_c = \text{depth of burial of the charge measured from the surface to the centre of gravity of the first 6 hole diameters of explosive, regardless of total explosive column length (m)}
\]

\[
W = \text{weight of explosive (kg)}
\]

This criterion, for most rocks, will assure minimal cratering at the collar of the hole. In very competent materials, the load should be designed to;
\[ d_c = 1.4^{\frac{3}{2}}2.2W \]

and in less competent material to:

\[ d_c = 1.1^{\frac{3}{2}}2.2W \]

As long as the design depth of burial is less than or approximately equal to the actual depth of burial, cratering will be minimal or nonexistent at the pre-split line.

Another form of buffer blasting, which is unique to strip coal mining, is where a block is blasted adjacent to a block of broken or previously blasted material. In this case, it has been found that the highwall cut is more stable, or does not suffer from overbreak, than when cut against unblasted but damaged strata. This will only work where the rock mass is weak and incompetent and needs only moderate loosening to be diggable with a dragline.

5.7 Cast Blasting

Many surface strip coal mining operations investigate from time to time the advantages of explosive overburden casting as a means of reducing their stripping cost. Explosive overburden casting may be defined as the application of explosive energy to remove overburden from above the mined horizon (normally coal) and cast (or throw) it onto the spoil pile. The percentage cast is defined as the fraction of overburden that is cast (thrown) into its final position on the waste (spoil) side, without the need for further handling.

In this case, the conventional blast design parameters previously discussed to not apply directly, but require some modification to allow for increased throw and a very different shape of muckpile. Blast design considerations must take into account three main areas;

- Adequate energy
- Burden velocity
- Mass-energy distribution.

If any one of these factors is ignored, then poor results can be expected. Specific to cast blasting, the most successful designs can be broadly related to the following guidelines;
ADEQUATE ENERGY
Whilst the ultimate aim of cast blasting is to use explosive energy to move overburden to the spoil side, this energy is over and above that required for conventional blasting. Powder factors as high as 0.5 - 0.7 kg/m$^3$ are required for a cast of 50 - 60%, and where the geometry of the highwall is unfavourable (15 - 20m high) the powder factor may be in excess of 0.9kg/m$^3$. Since cast is a function of highwall height, inclination and initial mass burden velocities, higher highwalls often assist in more efficient casting, assuming the same amount of energy per volume material is used.

BURDEN VELOCITY
Economical overburden cast blast designs require overburden velocities of between 10 and 25m/s. The higher the face burden velocity, the greater is the cast, up to a practical limit of about 27m/s. Over and above this figure, energy losses at joints, etc., become excessive and no benefit is seen.

The greater the number of holes fired per row, the greater is the efficiency of the blast in terms of BCM per unit explosive energy. Inter-hole delays should not exceed 15ms otherwise kinetic energy losses occur between holes and subsequent rows.

Timing between rows should aim to maximise the ejection of row associated burdens, ie., increase forward relief times. Burden response times (Table 5.5) are site specific, depending on material characteristics, but as an example, a 7 row blast may use 50 - 75ms delays between the 1st and 2nd rows and 125 - 175ms delays between the 6th and 7th.

MASS-ENERGY DISTRIBUTION
Once the appropriate energy requirements have been defined, the energy distribution in the rock mass must be found. This is no simple task and various schools of thought exist in this respect. It is imperative, however to maintain the appropriate powder factor in the front row of holes, regardless of how irregular the burdens may appear. The control of the front row burden breakage and displacement is critical to the efficiency of the whole blast, any sub-optimal results in the ejection of the first row burden are magnified to the last row, where results are usually very poor.

As can be seen from the simple guidelines above, the various parameters of the blast design and how they relate to a particular site
need to be assessed on an individual basis before the optimal design is found. Numerous authors have studied cast blasting and their design recommendations remain in the whole empirical. This means that whilst no single method is right or wrong for design purposes, there are a number of contributory design elements that can be used as a starting point in the design and economical analysis of such a design.

Seven different variables have been found to have a pronounced effect on the results of explosive overburden casting. These are:

- drill selection and blasthole inclination
- burden and spacing
- number of rows in the blast
- borehole stemming and decking
- delay timing
- explosives selection
- length of blast

**DRILL SELECTION AND BLASTHOLE INCLINATION**

When looking into the use of explosive overburden casting, the possibility of buying new drills should be examined for potential cost savings. Increasing drill hole diameter significantly increases the volume and thus weight of explosive that can be placed in the holes. An expanded blast pattern can then be used resulting in increased productivity. For example, for a site with a 30m overburden bench using 300 mm drills, a 375 mm drill rig may easily be employed, potentially reducing the number of holes drilled by 36%. Typically, a borehole diameter corresponding to 10.5-15.0mm for every 1m of overburden may be used successfully. Twelve meters approximately is the minimum highwall height that can be cast economically. For shorter highwalls, the cost of drilling the larger number of smaller diameter holes needed to maintain the powder factor probably outweighs any cost savings.

Drilling angle selection is critical in formulating the percent cast. Angled holes add a vertical component to the blast which gives a greater throw, with a theoretical maximum at 45 degrees. However, this may not be practical for various reasons, particularly when dragline reach is already limited (refer to a later Module).
BURDEN AND SPACING
Most disagreement in explosive overburden casting design is over the selection of spacing and burden dimensions. Opinions on optimum spacing to burden ratio vary widely. Two camps have evolved. The first recommends a ratio in the order of $S = 1.4B$, as there is less rock to move in front of the borehole, therefore high velocities may be achieved. The second and opposing camp recommends a ratio of less than one, i.e. a greater burden than spacing.

BOREHOLE INERT-STEMMING, DECKING AND BACKFILLING
The calculation of stemming is crucial in explosive overburden casting, as any premature venting of the explosion gases through the borehole collar will significantly reduce the work potential of the gases and result in a reduced cast. Stemming length should be in the range of 0.7 - 1.3B, and ideally still allow for the maximum possible loading of the borehole.

Decking is used when the total amount of explosive loaded in a hole is above the permitted weight allowed on a blast vibration basis. Inert material (usually drill cuttings) is alternately loaded with the explosive into the hole and used to separate the explosive into separate decks. Decking is generally detrimental to blast casting, as less explosive can be loaded in each hole and therefore the drill pattern has to be contracted to compensate. Up to four decks can be successfully used without significantly increasing the powder factor.

Backfilling is necessary when explosive overburden casting from above coal in order to protect the coal and prevent undue breakage and loss. Backfilling generally has to be increased when explosive overburden casting due to the increased powder factor and closer pattern. To prevent undue breakage and loss of coal, a backfill of 10-15d above the coal top contact is usual.

DELAY TIMING
An important aspect of delay timing is the type of delay system used. Since blast casting requires strict control and attention to detail, it is important that a reliable and accurate initiation system is chosen. Three different delays may be employed in a casting pattern;

- inter-hole delay
- inter-row delay
- delay between decks.

The first two will always be used in a pattern that consists of multiple rows, while decking is less commonly used except where vibration levels dictate. For superior casting the inter-hole delay should be minimized and ideally all holes in a row should be initiated simultaneously to produce the maximum cast and the most uniform muck pile.

The inter-row delay is critical if correct reinforcement between rows is to be obtained. Too short an inter-row delay results in the choking of successive rows, whereas an excessive delay results in the loss of any reinforcing effect between rows. An inter-row delay of approximately 10-15ms per meter of burden gives good casting results for overburden consisting of mudstones, interbedded shales and weak sandstones.

When decking is used, the bottom deck should be fired first, with the firing order progressing upwards, to prevent premature toppling of the face as it moves out, substantially reducing the percentage cast.

EXPLOSIVE SELECTION
Explosive overburden casting relies on the gas pressure produced by the explosive to propel the fragmented overburden towards the spoil pile, thus the gas producing properties of the explosive are the most important to consider. The economics of the selection must be considered in comparison with the anticipated savings derived from overburden handling.

LENGTH OF BLAST
In the middle of the front row of the blast, the confining effect of neighbouring blast holes results in the concentration of energy release and thereby rock movement away from the face, there being only one direction for the energy to dissipate, i.e. forwards. On the ends of a blast, where the end hole has only one neighbouring hole, the percent cast is much lower. Thus, for long blasts, the high percent cast area in the middle will constitute a larger proportion of the blast than the low percent cast areas on the ends. Therefore long blasts will provide a better casting action than short blasts.

ENVIRONMENTAL CONSIDERATIONS
Due to the higher powder factors and shorter delay periods utilised, explosive overburden casting is a more violent use of explosives
compared to more conventional blasting. Environmental considerations when adopting explosive overburden casting at a mine include increased vibration levels, flyrock and airblast.

To summarise, correct cast blast design is required to ensure a sufficiently high percent cast that will create savings in handling that more than offset the increased cost of explosives and drilling. Selectively changing a blast design for a specific site will yield varying percent casts. Although the highest cast possible may appear desirable, it may not necessarily be the best design in term of economics, especially when fixed costs are considered. The percent cast and costs incurred from each blast design should be quantified and analysed in the economic setting of the individual pit to determine the best blast design and target percent cast.
## LOADING AND HAULING SYSTEMS

### Learning outcomes

<table>
<thead>
<tr>
<th>Knowledge and understanding of</th>
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<tbody>
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<td>▪ Various types of loading equipment and their application in strip coal mining (overburden or coal handling)</td>
</tr>
<tr>
<td>▪ Haul length, production tempo and elevation data to select equipment</td>
</tr>
<tr>
<td>▪ Factors which dictate choice of loader</td>
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<td>▪ Loader application and working method</td>
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<td>▪ Current trends in large rope shovel applications</td>
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<td>▪ Single, double, backup methods of loading</td>
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<td>▪ Loading method selection factors</td>
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<td>▪ Truck selection factors for surface strip coal mining applications</td>
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<td>▪ Truck functional specifications</td>
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<td>▪ Tyre selection parameters</td>
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<td>▪ Specifications for haul road design</td>
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<tr>
<td>▪ Factors effecting loader productivity</td>
</tr>
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<td>▪ Cyclic and continuous excavation and transport systems</td>
</tr>
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<td>▪ Ancillary activities associated with the use of large BWE’s</td>
</tr>
<tr>
<td>▪ BWE excavating and dumping machine design options</td>
</tr>
<tr>
<td>▪ C-frame and hydraulic machine options</td>
</tr>
<tr>
<td>▪ Selection parameters and limitations</td>
</tr>
<tr>
<td>▪ Factors that influence BWE productivity</td>
</tr>
<tr>
<td>▪ Continuous and cyclic excavation and transport systems</td>
</tr>
<tr>
<td>▪ Operational advantages of conveying over truck-based haulage</td>
</tr>
<tr>
<td>▪ Material sizing limitation in belt conveying</td>
</tr>
<tr>
<td>▪ LCC and TCC system layouts</td>
</tr>
<tr>
<td>▪ Cost advantages of belt conveying over truck systems</td>
</tr>
</tbody>
</table>

### Apply, calculate or predict

- Predict the type of equipment for a specific combination of haul distance, production and elevation in a surface strip coal mining application
- Suitability of loader to specific loading operations
- TKPH limits on tyre selection
- Rolling resistance for haul road
- Haul road design specifications
- Shovel productivity for full and integer loads
- Suitability of method to given circumstances and situational analysis
- Suitability of conditions for use of BWE in a surface strip mine

**Evaluate or design**
- A suitable combination of cyclic and continuous loading and hauling systems for a given set of operational parameters
- Apply situational analysis in the selection of wheel, hydraulic or rope shovel selection
- Select and appropriate loading method for a given set of conditions
- Assess the advantages and disadvantages of each method in the context of strip coal mining
- Critically assess conveyor system advantages for specific applications in surface strip coal mining
6.1 Introduction and Equipment Options

In this module, loading and hauling equipment and sub-systems are introduced, reviewed and a selection methodology defined for the various machine and material types normally requiring excavation and transport in strip coal mining. The selection methodology is then developed to allow for selection decisions between similar equipment on the basis of continuity, etc. Finally, an analysis is required of what factors effect loading and hauling systems productivity.

The walking dragline is the predominant machine used for overburden removal in coal strip mining in South Africa. Whilst the dragline is a loader, it is however, used exclusively for overburden loading and transport. Since the factors governing its application and selection are different from other loading and hauling equipment described here, it will be more fully described in a later Module.

Under certain circumstances, other equipment combinations and/or mining methods may be more economically attractive. In contrast to overburden removal and transport, coal haulage is accomplished predominantly by truck and shovel methods, although the exact choice of truck type and shovel type varies widely. In the case of terrace or open-pit mining, truck and shovel systems predominate, but as depth or length of haul increases, continuous conveyor and crusher systems may be more economical at a point in the life of the mine.

The load and haul systems mentioned above are broadly referred to as either;

- Discontinuous or cyclic systems; single bucket machines (crawler mounted, wheeled or pivotal machines) with haulage system
- Continuous systems; multi-bucket machines (usually crawler mounted) with haulage system

Figure 6.1 shows the classification system for excavation equipment in terms of the various activities comprising material excavation, namely;

- ground preparation
- excavation
- transport
- dumping.
Some of the equipment options shown in Figure 6.1 can be used without a transport option, but as will be shown later in this Module, material transport distance is a significant cost and efficiency factor in excavation of material and the concept of an excavation system usually implies an excavating (loading) and transport (hauling) combination.

Figure 6.1 Classification of mining excavation and transport systems.

6.2 Loading Equipment - Cyclic

6.2.1 Transport Distance and Production Volume Effects

A large range of loading equipment types are available and their selection depends primarily upon the type of material to be excavated, the distance (including elevation) over which it has to be moved and the mining rate (LCM or m$^3$ per hour) required (LCM – loose cubic meter – the swelled BCM volume in-situ after it has been blasted).

For example, if 1 LCM of material is to be moved over a distance of 1m, then a spade is probably the best tool to use. If the distance increases to 10m, then a spade and wheel-barrow would be a better
option. When the LCM rate increases, larger shovels are used, and where the distance also increases, larger shovels are used in combination with larger mine haul trucks.

From an equipment selection point-of-view, the starting point is low volume push-type equipment, typically the bulldozer, wheel-dozer and motor-grader equipment in which material transport distance is short (less than 100m) and the material can be pushed by a blade. The dozer has a large blade capacity and is designed specifically for bulk material excavation, whereas the grader is primarily designed for “finishing” work – moving only small amounts of material relatively quickly over short distances. All three machines do not have a load elevation capacity – i.e. They cannot ‘lift’ the material. For this type of equipment, production rate (LCM/hr) is a function of blade size and ease of excavation, loose unconsolidated material is ideal, and both machines experience a loss of productivity if the material is difficult to excavate – solid rock cannot be excavated without blasting or ripping first. Most large dozers are equipped with ripping equipment, whereas motor graders only have a limited (depth and material consolidation) capacity to rip. Wheel-dozers are not usually equipped for ripping. Figure 6.2 shows the range of activities normally associated with wheeled or track dozers and gives an indication of the applicability of various models for the task.

Equipment productivity and unit cost of material moved is dependant on the application, speed of operation, depth of cut (or weight of load) and material type and thus difficult to generalise, but Figures 6.3 and 6.4 show typical productivity- and unit cost- transport distance relationships, for loading and transporting using a dozer (tracked) or a front-end loader (wheeled). From Figures 6.3 and 6.4, the concept of economies of scale can be seen; how larger equipment has lower unit costs and higher productivities. Whilst productivity gains are common-place with larger equipment, the cost benefit may not always be realised, especially when large capital expenditures are undertaken for relatively small volumes of material to be moved.
<table>
<thead>
<tr>
<th>Job</th>
<th>Ground conditions</th>
<th>Wheel dozer (rubber)</th>
<th>Track dozer</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Clean-up around small shovel</td>
<td>Friable, free-digging</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>No or light ripping</td>
</tr>
<tr>
<td></td>
<td>Hard friable</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>Light-med ripping</td>
</tr>
<tr>
<td></td>
<td>Hard, med rocks</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>Hard ripping</td>
</tr>
<tr>
<td>Clean-up around large shovel</td>
<td>Friable, free-digging</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>No or light ripping</td>
</tr>
<tr>
<td></td>
<td>Hard friable</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>Light-med ripping</td>
</tr>
<tr>
<td></td>
<td>Hard, med rocks</td>
<td>Small: Blue; Medium: Yellow; Large: Orange</td>
<td>Small: Red; Medium: Blue; Large: Orange</td>
<td>Hard ripping</td>
</tr>
<tr>
<td>Waste dump dozing capability</td>
<td>Med rocks</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks</td>
</tr>
<tr>
<td></td>
<td>Large rocks</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Medium trucks</td>
</tr>
<tr>
<td></td>
<td>Wet, muddy, rocky</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Large trucks</td>
</tr>
<tr>
<td>Spoil dozing capability</td>
<td>Med rocks</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Easy terrain</td>
</tr>
<tr>
<td></td>
<td>Large rocks</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Difficult terrain</td>
</tr>
<tr>
<td></td>
<td>Wet, muddy, rocky</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Small trucks: Green; Medium trucks: Yellow; Large trucks: Orange</td>
<td>Easy terrain</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Difficult terrain</td>
</tr>
</tbody>
</table>

**KEY TO COLOURS**

- **Should consider**
- **May consider**
- **May consider under certain circumstances**
- **May be considered under special situation**
- **Not an option for this equipment**

Figure 6.2 Selection of dozing equipment according to job conditions
When transport distance increases beyond the 200-300m mark, it is often appropriate to consider a combination of load and haul equipment. Where the volumes to be moved are relatively small, scrapers can be used, as long as the material is soft enough to be excavated or 'scraped' into the bowl of the scraper (or in some circumstances, where the material is friable and can be loaded after blasting. Under these circumstances, large dozers are often used to push-load a scraper, since the latter is designed more for hauling than loading). This type of equipment is typically used for top- and sub-soil removal where no blasting is required. They have no load elevation facility (except internally for loading into the bucket) but have a larger storage capacity for the material to be transported.

![Short transport distance systems: Production ranges (ton/hour)](image)

**Figure 6.3**  Productivity – transport distance profile for dozers and front-end loaders in load and carry.

Where the material to be loaded is blocky, or limited in extent (typically a blasted bench muck-pile) then a loader may be used in combination with small haul trucks. For longer haul distances, wheel-type loaders are used (referred to as FEL’s – front-end-loaders) which also have a load elevation capacity (i.e. Can lift the bucket to dump in truck or on a stockpile) – hence this equipment can ALSO be used as a loading shovel (in conjunction with or without trucks).

The haul truck options are typically articulated dump trucks (ADT’s) or small rear dump trucks (RDT’s). These units will be discussed in more detail later in the Module. **Figure 6.5 and 6.6** show the typical
system productivity and relative unit costs for these load and haul systems – note that for the combined wheel loader and truck (ADT or RDT), specific costs and productivities are quite variable, depending on the match of truck to loader, haul distance and number of units operating. Again, this concept will be expanded upon later.

Figure 6.4 Relative cost – transport distance profile for dozers and front-end loaders in load and carry.

Figure 6.5 Productivity – transport distance profile for scraper-dozers and front-end loaders with ADT or RDT.
For much longer haul distances and higher production volumes, large loading and hauling units are used. Again, the relationships in Figures 6.7 and 6.8 are generalised and would vary according to the specific conditions of the job, length and grade (resistance) of haul, size and type of RDT, number of loading units, etc.

**Figure 6.6**  Relative cost – transport distance profile for scraper-dozers and front-end loaders with ADT or RDT.

**Figure 6.7**  Productivity – transport distance profile for loaders with RDT or BDT.
Figure 6.8 Relative cost – transport distance profile for loaders with RDT or BDT.

Table 1 summarises the transport distance and production volume effects for the range of cyclic non-elevating and elevating loading equipment introduced. Before paying specific attention to typical loading systems used in strip coal mining, Figures 6.9 - 12 summarise the application criteria and indicate specific loading equipment suitability to a number of tasks associated with surface strip coal mining.
<table>
<thead>
<tr>
<th>Equipment</th>
<th>Lift/Not lift</th>
<th>Optimum transport distance</th>
<th>Production rate</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Bulldozer</strong></td>
<td>Not lift - push</td>
<td>&lt; 100m</td>
<td>Function of:</td>
<td>Loose, unconsolidated or fragmented rock</td>
</tr>
<tr>
<td></td>
<td></td>
<td>&lt; 200m</td>
<td>▪ blade size</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ engine power</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ ease of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>excavation</td>
<td></td>
</tr>
<tr>
<td><strong>Wheel-type dozers</strong></td>
<td>Not lift - push</td>
<td></td>
<td>Function of:</td>
<td>Loose, unconsolidated or lightly consolidated</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Application</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Blade width</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Speed of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>operation</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Depth of cut</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Material type</td>
<td></td>
</tr>
<tr>
<td><strong>Motor-grader</strong></td>
<td>Not lift - push</td>
<td>&lt; 100m</td>
<td>Function of:</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Depth of cut</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Speed of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>operation</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Rate of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>unloading</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Material type</td>
<td></td>
</tr>
<tr>
<td><strong>Bowl- or elevating-scraper</strong></td>
<td>No load</td>
<td>&lt; 2000m</td>
<td>Function of:</td>
<td>Soft enough to be excavated or scraped into</td>
</tr>
<tr>
<td></td>
<td>elevation,</td>
<td></td>
<td>▪ Depth of cut</td>
<td>the bowl</td>
</tr>
<tr>
<td></td>
<td>larger storage</td>
<td></td>
<td>▪ Speed of</td>
<td></td>
</tr>
<tr>
<td></td>
<td>capacity for</td>
<td></td>
<td>operation</td>
<td></td>
</tr>
<tr>
<td></td>
<td>material to</td>
<td></td>
<td>▪ Rate of</td>
<td></td>
</tr>
<tr>
<td></td>
<td>be transported</td>
<td></td>
<td>unloading</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Material type</td>
<td></td>
</tr>
<tr>
<td><strong>FEL (wheel loader)</strong></td>
<td>Lift</td>
<td>&lt;200m (=&lt;400m round trip)</td>
<td>Function of:</td>
<td>Loose fragmented rock or unconsolidated</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Ease of digging</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Road grade</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Speed of travel</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Ease of dumping</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Truck</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>productivity</td>
<td></td>
</tr>
<tr>
<td><strong>Shovels (also including FEL)</strong></td>
<td>Used only for digging</td>
<td>&lt; 20m to truck</td>
<td>Function of:</td>
<td>Digging conditions difficult – hard, large</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Ease of digging</td>
<td>fragmented rock</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Bucket swing</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>angle</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Ease of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>dumping</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Truck</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>productivity</td>
<td></td>
</tr>
<tr>
<td><strong>Additional truck transport</strong></td>
<td></td>
<td>&gt; 300-400m</td>
<td>Function of:</td>
<td>Any</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Shovel</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>productivity</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Road grade</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Speed of travel</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>▪ Ease of</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>dumping</td>
<td></td>
</tr>
</tbody>
</table>
### Excavating equipment

<table>
<thead>
<tr>
<th>Equipment rating for topsoil</th>
<th>Load only</th>
<th>Load and haul scrapers</th>
<th>Load and haul with transport</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dozers</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Front end loader (FEL)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dragline</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Elevating</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Push-pull</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>With push tractor</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Shovel and truck</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Bucket wheel excav</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>FEL and truck</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hyd front shovel and truck</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Key to Colours

- **Orange**: Should consider
- **Yellow**: May consider
- **Green**: May be considered under special circumstances
- **Blue**: Not an option for this equipment
- **White**: Not an option for this equipment

<table>
<thead>
<tr>
<th>Depth of soil (m)</th>
<th>0 - 0.6m</th>
<th>0.6 - 1.5m</th>
<th>0 - 100m</th>
<th>100 - 150m</th>
<th>150 - 300m</th>
<th>300 - 500m</th>
<th>&gt;500m</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.6 - 1.5m</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0 - 100m</td>
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<td></td>
</tr>
<tr>
<td>100 - 150m</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Haul distance (m)</th>
<th>Good</th>
<th>Fair</th>
<th>Poor</th>
</tr>
</thead>
<tbody>
<tr>
<td>150 - 300m</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>300 - 500m</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>&gt;500m</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Figure 6.9** Loading equipment selection – topsoil removal
### Equipment rating for overburden

<table>
<thead>
<tr>
<th>Excavating equipment</th>
<th>Load only</th>
<th>Load and haul scrapers</th>
<th>Load and haul with transport</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Loaders</td>
<td>Dredge</td>
<td>Shovel (rope)</td>
</tr>
<tr>
<td></td>
<td>Dragline</td>
<td></td>
<td>Shovel and truck</td>
</tr>
<tr>
<td></td>
<td>Shovel and truck</td>
<td></td>
<td>Bagger and excav</td>
</tr>
<tr>
<td></td>
<td>Elevating</td>
<td></td>
<td>Push tractor</td>
</tr>
<tr>
<td></td>
<td>With push tractor</td>
<td></td>
<td>Shovel and truck</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Hyd front shovel and truck</td>
</tr>
</tbody>
</table>

#### Key to Colours
- Should consider
- May consider
- May consider under certain circumstances
- May be considered under special circumstances
- Not an option for this equipment

<table>
<thead>
<tr>
<th>Depth of soil (m)</th>
<th>1 -10m</th>
<th>10 -20m</th>
<th>20 - 30m</th>
<th>&gt;30m</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fragmentation</td>
<td>Poor</td>
<td>Blocky</td>
<td>Fine</td>
<td>Unconsolidated</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Haul distance (m)</th>
<th>15 -50m</th>
<th>50 - 100m</th>
<th>100 - 150m</th>
<th>150 - 300m</th>
<th>&gt;300m</th>
</tr>
</thead>
<tbody>
<tr>
<td>Floor support</td>
<td>Good (Hard)</td>
<td>Fair</td>
<td>Poor (soft)</td>
<td>Good (Hard)</td>
<td>Fair</td>
</tr>
<tr>
<td>Flexibility to conditions</td>
<td>Should consider</td>
<td>May consider</td>
<td>May consider under certain circumstances</td>
<td>May be considered under special circumstances</td>
<td>Not an option for this equipment</td>
</tr>
<tr>
<td>Mobility</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Segregation capacity</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Production potential</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Figure 6.10** Equipment selection – overburden removal
<table>
<thead>
<tr>
<th>Equipment rating for coal loading</th>
<th>Excavating equipment</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Load, with truck transport</td>
</tr>
<tr>
<td></td>
<td>Front end loader (FEL)</td>
</tr>
<tr>
<td>Coal seam thickness (m)</td>
<td></td>
</tr>
<tr>
<td>0.3 - 1.0m</td>
<td>High</td>
</tr>
<tr>
<td>1.0 - 1.5m</td>
<td>Medium</td>
</tr>
<tr>
<td>1.5 - 3.0m</td>
<td>Poor</td>
</tr>
<tr>
<td>3.0 - 7.5m</td>
<td>Good (Hard)</td>
</tr>
<tr>
<td>&gt;7.5m</td>
<td>Poor (soft)</td>
</tr>
<tr>
<td>Fragmentation</td>
<td></td>
</tr>
<tr>
<td>Poor</td>
<td>Should consider</td>
</tr>
<tr>
<td>Blocky</td>
<td>May consider</td>
</tr>
<tr>
<td>Fine</td>
<td>May consider under certain circumstances</td>
</tr>
<tr>
<td>Floor support</td>
<td></td>
</tr>
<tr>
<td>Good (Hard)</td>
<td>Should consider</td>
</tr>
<tr>
<td>Fair</td>
<td>May consider</td>
</tr>
<tr>
<td>Poor (soft)</td>
<td>May consider under certain circumstances</td>
</tr>
<tr>
<td>Mobility</td>
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</tr>
<tr>
<td>Good (Hard)</td>
<td>Should consider</td>
</tr>
<tr>
<td>Fair</td>
<td>May consider</td>
</tr>
<tr>
<td>Poor (soft)</td>
<td>May consider under certain circumstances</td>
</tr>
<tr>
<td>Flexibility to conditions</td>
<td></td>
</tr>
<tr>
<td>Good (Hard)</td>
<td>Should consider</td>
</tr>
<tr>
<td>Fair</td>
<td>May consider</td>
</tr>
<tr>
<td>Poor (soft)</td>
<td>May consider under certain circumstances</td>
</tr>
<tr>
<td>Production potential</td>
<td></td>
</tr>
<tr>
<td>High</td>
<td>Should consider</td>
</tr>
<tr>
<td>Medium</td>
<td>May consider</td>
</tr>
<tr>
<td>Low</td>
<td>May consider under certain circumstances</td>
</tr>
</tbody>
</table>

Figure 6.11 Equipment selection – coal loading
6.2.2 Selection Criteria – Cyclic Loaders

Large loaders combined with haulage trucks are the most widely used excavation system for rock haulage in surface mining, primarily as a result of their flexibility of use. The trucks are loaded by various equipment, with rope shovels dominating large volume rock moving because of their robustness and cost-effectiveness. Other equipment, such as front end loaders and hydraulic shovels, are challenging the rope shovel especially where mobility and excavation complications exist. In the following analysis of selection factors, only the three main classes of cyclic elevating loaders are considered, namely;

- Front end loaders (wheeled)
- Hydraulic shovels (crawler mounted, pivotal)
- Rope shovel (crawler mounted, pivotal)
The front-end-loader combines the excavation and transport function (over a short distance of <200m or to dump in a truck), whilst the rope and hydraulic shovels excavate and transport over a distance of <20m to dump the rock in a truck (whilst the shovel remains in the same position). In the following discussion, it is assumed that the FEL is operated primarily as a loading, as opposed to load and carry, machine.

The overall size (ROM tons coal or BCM mined and stripping ratio) of a surface strip coalmine has considerable effect on the selection of an excavator. As the mine size reduces, so does the amount of material to be moved and, with it, often the size of the equipment used (the physical size of the mine places a limit on the size of a blast and thus large equipment will not be so effective due to their higher production rates). The size of equipment is thus a function of production rates and economic justification: For smaller mines, less capital is available for purchase of equipment, the pay-back period is often short (shorter mine life) and planning is shorter term.

The worldwide shovel population is reviewed and published in the Parker Bay Report and although the data does not allow for a separate analysis for strip or terrace coal mining, the trends in selection and application are useful to review since in supplying shovels to the market, the manufacturers respond to market demand, which is shaped by the selection process.

Analysing the Parker Bay report on the worldwide shovel/excavator population, it is clear that the trend towards bigger loading equipment still continues. Collectively there are nearly 3,500 large (10m$^3$ and larger bucket capacity) loaders operating at the 713 mines identified in this census: an average of nearly five per mine. Cumulative capacity is over 60,000 m$^3$ yielding an average bucket size of 17.7m$^3$.

A summary of the distribution of shovels by type and size is presented in Table 6.2 and the relative change in product type supplied in Table 6.3. It can be concluded that although there are movements in the market share between the different shovel types, the momentum is generated by a drive to increase the size of the shovels. Any increase in market share is at the expense of the next bigger category of shovels and a decrease in units is offset by a further increase in capacity. This tendency might be due to an effort to reduce costs in order to utilise lower grade deposits or reserves at a higher stripping ratio.
Table 6.2 Loading equipment population by product type and payload, 2001

<table>
<thead>
<tr>
<th>Loader</th>
<th>Units</th>
<th>Total capacity (m$^3$)</th>
<th>Percent of total capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rope</td>
<td>1199</td>
<td>26979</td>
<td>44%</td>
</tr>
<tr>
<td>Hydraulic</td>
<td>1045</td>
<td>16340</td>
<td>26%</td>
</tr>
<tr>
<td>FEL</td>
<td>1243</td>
<td>18257</td>
<td>30%</td>
</tr>
</tbody>
</table>

Payload

<table>
<thead>
<tr>
<th>Payload</th>
<th>Units</th>
<th>Total capacity (m$^3$)</th>
<th>Percent of total capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 15 m$^3$</td>
<td>1910</td>
<td>22821</td>
<td>37%</td>
</tr>
<tr>
<td>15 – 24 m$^3$</td>
<td>1045</td>
<td>19961</td>
<td>32%</td>
</tr>
<tr>
<td>&gt;25 m$^3$</td>
<td>522</td>
<td>18794</td>
<td>31%</td>
</tr>
</tbody>
</table>

Table 6.3 Loading equipment population change (2001 against 1998) by product type and payload.

<table>
<thead>
<tr>
<th>Loader</th>
<th>Percent change in units</th>
<th>Percent change in total m$^3$ capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rope</td>
<td>-10%</td>
<td>-4%</td>
</tr>
<tr>
<td>Hydraulic</td>
<td>+ 9%</td>
<td>+9%</td>
</tr>
<tr>
<td>FEL</td>
<td>+20%</td>
<td>+20%</td>
</tr>
</tbody>
</table>

Payload

<table>
<thead>
<tr>
<th>Payload</th>
<th>Percent change in units</th>
<th>Percent change in total m$^3$ capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 15 m$^3$</td>
<td>+13%</td>
<td>+12%</td>
</tr>
<tr>
<td>15 - 24 m$^3$</td>
<td>-12%</td>
<td>-12%</td>
</tr>
<tr>
<td>&gt;25 m$^3$</td>
<td>+16%</td>
<td>+22%</td>
</tr>
</tbody>
</table>

In very general terms, the typical characteristics of a successful truck and shovel application are described as:

- **Mine life.** The lower initial capital investment in such a system can be repaid over a shorter period and the impact on mine economics (in terms of capital expenditures) is not that great, as, for instance, would be the purchase of a dragline for
overburden handling. However, operating costs are also an important consideration too – especially with high volume operations.

- **Product value.** A discontinuous truck and shovel excavating system has a high operating cost when compared to a continuous system or a dragline based system. It thus follows that the material moved must then have a high value, or at least expose a high value material when moved.

- **Deposit depth.** System productivity is very sensitive to load elevation (gain in height from loading to dumping point). However, the incremental penalty (in terms of reduced productivity) is not as great as with some other systems. In relatively shallow strip mining, in the absence of excessively steep seam dips, the load elevation effect, whilst a significant contributor to cost per ton excavated and hauled, does not increase markedly once the floor of the seam is reached.

- **Deposit variability.** If the deposit is variable in terms of coal quality, working faces, dumping areas, etc. then a truck based system is most suitable. When deposit characteristics are uniform and production is generated from fixed locations for long periods of time, some other system will prove to be better. In strip coal mining, the rate of advance of the mining face and the turnover of strips precludes the use of any other system that is onerous to advance with the mining face.

- **Working conditions.** Limited pit room, poor ground conditions, unavailability of skilled labour and poor product after service do not impact on the productivity of the system as would be the case for larger single units (eg. draglines).

- **Risk.** It is not difficult to make such a system work, even when some critical assumptions are not realised when the mining begins. It may not be the most efficient solution but its inherent flexibility allows it to accommodate most planning errors. If it does not work, the units can be easily sold.

When considering the selection of a truck and shovel system in strip coal mining, the most appropriate (but not the only) application is in coal hauling, for the following reasons;

- The economics of the mine are generally not sensitive to coal hauling costs – the less so as overburden depth increases.

- The flexibility of trucks places less constraints on the waste removal operations and allows scope for its optimisation.
The deposit geometry and mining method are suited to a truck and shovel system by virtue of the rate of advance of the mining face and its shape.

Other applications include that of overburden removal (as the main method or as a pre-strip prior to the use of draglines on the main overburden bench). For the pre-stripping of overburden, a truck and shovel combination is widely used primarily as a result of their cost (in view of the possibly short duration of the pre-strip requirement), risk and manageability. This does not mean however, that it is the only option and in the final analysis the choice of equipment in this type of application is very much site specific.

The decision to reject other methods in favour of truck and shovel systems is not always clear cut, even given the reasons above. For main waste removal in large scale terrace mining, a decision to select a cyclic loader and trucks as the primary excavation system is not clear-cut. The trade-off between loader and trucks and other methods is very complex, especially for high volume, low value, marginally rippable overburden stripping. Therefore, the loading shovels here are discussed in terms of their general and application characteristics. Although in surface strip coal mining, these loading shovels are used mostly to excavate blasted coal – for smaller operations (or alternative mining methods) where no dragline can be used, also overburden, there is also a certain amount (distance and elevation) of transport also done by the shovel. Thus the first general selection factor is the amount of material to be excavated and the distance over which it is moved, as was shown in Figures 6.2, 4 and 6.

Loading shovels are characterised by several parameters, which enable the most appropriate shovel to be chosen for the conditions. The four main parameters are;

1. Bucket capacity (m$^3$), machine weight (t) and drive power (kW)
2. Material to be excavated
3. Geometry of the mine
4. Economic considerations

The starting point is normally to evaluate the first three parameters in terms of the extent to which they would produce an ‘obvious’ candidate. Where various types and sizes of loader remain
candidates, economic considerations would then play the deciding role.

**BUCKET CAPACITY, MACHINE WEIGHT AND DRIVE POWER**

Bucket capacity (m³), machine weight (t) and drive power (not a reflection of digging power but generally related to size of bucket) (kW) are typically in the range given in Table 6.4.

Table 6.4 Cyclic loader general specifications

<table>
<thead>
<tr>
<th>LOADER TYPE</th>
<th>BUCKET CAPACITY (m³)</th>
<th>WEIGHT (t)</th>
<th>POWER (kW)</th>
</tr>
</thead>
<tbody>
<tr>
<td>FEL</td>
<td>2.5 – 23</td>
<td>13 - 190</td>
<td>126 - 890</td>
</tr>
<tr>
<td>HYDRAULIC</td>
<td>3.0 – 35</td>
<td>50 - 450</td>
<td>210 - 1850</td>
</tr>
<tr>
<td>ROPE</td>
<td>4.5 – 61</td>
<td>220 – 740</td>
<td>160 - 445</td>
</tr>
</tbody>
</table>

**Bucket capacity (m³):** Actual volume of material that can be carried in an equipment bucket, specified as heaped or struck.

**Heaped:** Bucket capacity rating. Volume in the bucket under the strike-off plane plus the volume of the heaped material above the strike-off plane, having an angle of repose of 1:1., without any consideration for any material supported or carried by the spillgate or bucket teeth.

**Struck:** Bucket capacity rating. The volume actually enclosed inside the outline of the sideplates and rear and front bucket enclosures without any consideration for any material supported or carried by the spillgate or bucket teeth.

Bucket selection will play an integral part in the efficiency of a loader and is influenced by the material type and truck to be loaded, production rates required, cost etc. Bucket loadability is a function of width, height, depth, curvature, material thickness, cutting edge shape (straight versus V-type) and ground engagement tools options. The optimum combination of these attributes allows a bucket to penetrate the material, fill the bucket and dump the material. The ability to penetrate the material is determined by the cutting edge thickness, shape and width. A thin edge penetrates better, but does not wear as well as a thicker edge and a V-type edge penetrates better than a straight edge. The bucket width is determined either by the tyre coverage required (for a FEL), geometric and mass limitations as well as the dump target.
The ease of material flow in and out of a bucket depends on the depth, height and curvature of the bucket. Different material requires different height-to-depth-to-width-to-curvature relationships. Material that is easy to penetrate will allow a wider and deeper bucket. Material that is difficult to penetrate requires a narrow, shallower bucket. This bucket penetrates less surface area and has more breakout force. Sticky material will require a more open curvature and a shallow bucket.

The selection of a bucket is influenced by many factors, thus the bucket seldom will dictate by itself which loader should be chosen, only the performance of the loader in a given set of digging and dumping conditions.

**THE MATERIAL TO BE EXCAVATED**

Following from bucket selection, consider the following basic groups:

- Soils (unconsolidated, no blasting required)
- Overburden (hard materials, blasting required)
- Parting between coal seams (hard materials, blasted)
- Coal (hard material, blasted).

These materials differ in terms of;

- The degree of difficulty with which it can be excavated (diggability) (mostly a function of the degree of consolidation or fragmentation from blasting). Each type of shovel has a diggability rating which is related to the type of shovel, bucket, bucket loading action and applied bucket forces.

- Unit weight \( \gamma \) (N/m\(^3\)), the product of material density \( \rho \) and acceleration due to gravity \( g \):

\[
\gamma = \rho g
\]

- Swell factor (both after blasting and more importantly – as it is excavated to give LCM from BCM)
- Cohesiveness (especially with clays)
- Abrasiveness (determines the life of bucket and ground engaging tools)
- Average fragment size (blasted or non-blasted material – determines bucket fill factor and diggability).

All of the above need to be considered for the basic shovel selection, especially bucket shape.

**THE GEOMETRY OF THE LOADING AREA**
This is related to the mining method chosen. Consider;

- Maximum shovel digging height compared to the height of the blasted rock pile. Bench height is probably the single most vital parameter in the equipment selection process; higher bench heights will normally result in lower operating costs and optimum productivity. A relationship between bench height and bucket size is given in Table 6.5, from the perspective of optimum productivity, not necessarily technical feasibility or safety.

- The impact of bench height over dig cycle time is also important and is related to the method whereby the loader bucket is filled. Two different approaches are proposed in determining the bench height:

  1. Vertical distribution of ore and waste. In the case where an orebody is irregular, both vertically and horizontally, there could be various bench heights that would optimize the ore recovery and dig cycle time. In the case of coal seams, this is often dictated by seam thickness, interburden thickness, etc. and there is little scope for optimisation.

  2. Required production rates: The planned production requirements will determine the size and/or quantity of equipment used in the mine. The desired bench height will be determined within certain limits by the size and type of equipment. In general, savings can be realized if the bench height can be made equal to the maximum vertical working height of the loading and drilling equipment. This would be applicable to overburden as opposed to coal mining.

- Maximum shovel dumping height (free-dump or truck-body – the latter often a limitation with FEL’s and trucks with higher sides or coal bodies fitted)
Table 6.5  Bench height relationship to loader bucket size

<table>
<thead>
<tr>
<th>Bucket size m³</th>
<th>Bench Height (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;5.0</td>
<td>9</td>
</tr>
<tr>
<td>5.1 – 8.0</td>
<td>12</td>
</tr>
<tr>
<td>8.1 – 20.0</td>
<td>14</td>
</tr>
<tr>
<td>20.1 – 30.0</td>
<td>16</td>
</tr>
<tr>
<td>&gt; 30.0</td>
<td>18</td>
</tr>
</tbody>
</table>

- Slope of floor (rope- and hydraulic shovels rotate on a tub and require fairly level floors, the FEL since it runs on tyres can work on almost any floor, but tends to be unstable on dipping floors when the bucket is raised and full)
- Floor conditions (wet, rough, loose rocks etc). Water and loose rocks easily cut FEL tyres and loose rock hinders traction during loading
- Digging face profile – blending on the face, loading from bottom only (possible stability problems), selective mining (hydraulic shovels), etc.
- These factors can be shown in form of a diagram (Figure 6.13) as excavation limitations. The specific influence of each of these geometric factors will be discussed under the descriptions of the loaders themselves and the loading method options.

ECONOMIC CONSIDERATIONS

Where the previous selection factors alone do not bring to light one particular candidate for choice of loader, it is necessary to evaluate the economics of the various options one with the other. With mining equipment accounting for as much as 75% of the initial capital cost of a surface mining operation, and some major equipment having a potential service life of 15 to 20 years, it is obvious that any selection of equipment will have a profound effect on the long term viability of an operation. The following general factors should be considered:
Figure 6.13 Loading shovel limitations of a typical surface mine

- Expected life of machine in comparison to that of the mine
- Availability of the machine over its working life and the likely change with age
- Maintenance costs – especially in difficult ground conditions
- Capital cost and life of mine (repayment period)
- Productivity of the unit in conjunction with existing truck fleet and mining geometry (utilised digging time especially important)
- Operating cost (R/BCM or t excavated) – especially in difficult ground conditions.

To summarize, the above-mentioned selection factors can be used to determine which is the best shovel for a certain application. There is usually no optimum choice, rather a choice that optimises performance in a particularly critical area. It is often useful to compare equipment using a pre-defined analytical framework to determine, for instance, life cycle cost of each. Capital and operating cost estimation and comparison is covered in more detail in a later Module.
6.2.3 Front-end Loaders

The front-end loader is a diesel powered or diesel-hydraulic wheel mounted tractor with a bucket at the front. The wheel loader differs from the rope shovel and hydraulic shovel in that the loading action is not carried out from a stand still. A wheel loader is designed to carry 18-21% of its operating weight as a bucket payload and thus have a very favorable payload-to-operating weight ratio, which translates into very high mobility. It is widely used in surface mining and, depending on its size (larger units are dedicated loaders, smaller units fulfil also a general mining utility function) to:

- Excavate
- Load
- Transport (up to 200m distance at 12% maximum grade)
- General cleaning, support, construction and material transport.

Four-by-four driven wheels on an articulated frame give the FEL excellent maneuverability in good ground conditions. Loading trucks requires the machine to move back and forth 10-20m per bucket load, whilst direct dumping requires longer transport distances. A maximum of 200m round trip is the transport distance limit, beyond that a truck combination should be used due to the FEL’s limited (laden) speed, hill climbing ability and ton-kilometer per hour (TKPH) tyre rating limitations on the bigger machines.

A front-end loader can work on the thinnest material, although if a production operation were designed around very thin bands, output would be reduced substantially (but the advantage of using a front-end loader on these lower benches is that the mobility suits the high lateral advance rate of the face). From the operator’s cab, the operator can see very little of the material flowing into the bucket. When cleaning up the top of a coal seam for instance, the front-end arrangement is easily capable of the necessary selectivity, but it is equally easy to lose coal because the operator cannot see. A smaller unit would be more appropriate for cleaning, or possibly even a wheel dozer, leaving the larger unit for a primary production role.

Front-end loaders are best suited to well blasted and/or soft digging, and are not normally considered for the hardest digging conditions. They can operate efficiently on any face height up to 10-15m and although there is no real upper limit to face height, provided material is free flowing or can be dozed in, the only issue is the safety of working near the final wall which is beyond the lift of the bucket (since digging the face effectively steepens the slope angle of the blasted...
rock). Good working practice is to limit muck-pile height to not exceed the hinge pin height at full lift. Figure 6.14 shows the typical digging profile and Figure 6.15 the bucket mobility.

Figure 6.14 Excavation profile for front-end-loaders

Figure 6.15 Bucket mobility of front-end-loaders

Front-end loaders generally support the bucket on a one-piece arm pivoted on the front of the loader. The arm is raised or lowered by hydraulic cylinders, and the tilt of the bucket is controlled by a second set of hydraulics and link mechanisms, also ultimately supported on the front of the loader. Two alternative front-end arrangements are
commonly offered - a standard arrangement and a high-lift arrangement. High lift arrangements are equipped with smaller buckets but permit easier loading of larger trucks.

The machine digs through a combination of crowd (drive into the rock pile) and hoisting of the bucket. Excessive tyre wear is typical due to abrasive rock – especially on the front tyres (which run into the rock pile). The machine must obviously cut a path into the rock that is as wide as the machine itself and this forms the basis of machine power to bucket size.

Loading is cyclic and exists of:

- Acceleration up into the rock pile and load the bucket (bucket on the floor of the bench), then wrist bucket upwards
- Reverse, lift the bucket and make a “y” turn
- Approach the truck and dump, or
- Lower the bucket (maximizing stability) drive to the dump area
- Drive back to the muck pile.

as shown in Figure 6.16.

Of the three types of loaders, the front-end loader has the second lowest breakout force in consolidated ground, even though some bucket actions such as roll back improves the digging capabilities (but does a lot of damage to the machine). Considering the digging profile of the machine, the primary applications of the machine are:

- Loading and transportation of top soil
- Loading and transportation of the coal seam from the blasted face or stockpile
- Loading and transportation of blasted or soft over-burden

Front-end loaders are not greatly sensitive to block width and can work in less than 25 metres pit working room (this also depends on truck type and loading method used).
Figure 6.16 Front-end loader conventional loading method

Most of the above mentioned applications are limited to the smaller production strip mines. The bigger series front-end loaders with bucket sizes over $23\text{m}^3$ can be used effectively in the loading of larger trucks. Three loading methods dominate, depending on the compatibility or match between the loader and the truck in terms of sizes:

- Conventional method (single or double back-up)
- Tandem loading method - where the truck doesn’t need more than 2 bucket loads to fill it
- Chain loading (drive by) – where a truck needs 3 or more bucket loads and 2 or more FEL’s are available (generally a very unproductive option).
6.2.4 Hydraulic Shovels

Although hydraulic, the name only refers to the bucket drive system – these machines are electrically powered (larger types) or diesel driven (smaller types) (motor drives hydraulic pumps). These machines are currently being aggressively developed with ever-increasing bucket sizes. Two types of machine configuration are available for hydraulic shovels and whilst they share some similar application characteristics, their operation and loading methods are significantly different;

- Front (face) shovel configuration
- Back hoe excavator configuration.

The two variations of the machine are used in the same way except that:

- The back hoe digs under its working level to load (and trucks are usually also on the bottom working level or bench below to reduce the bucket’s hoist time and thus to reduce excavation cycle time - except if conditions are wet or inaccessible on the bottom level)
- The front shovel digs at or above its working level (or floor) and dumps in the trucks on the same working level as the machine itself.

Typical units have the characteristics as given in Table 6.6.

<table>
<thead>
<tr>
<th>MACHINE TYPE</th>
<th>BUCKET CAPACITY (m³)</th>
<th>POWER (kW)</th>
<th>WEIGHT (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Back hoe</td>
<td>3 - 23</td>
<td>210 - 650</td>
<td>60 - 235</td>
</tr>
<tr>
<td>Front shovel</td>
<td>3 - 35</td>
<td>210 - 1850</td>
<td>- 450</td>
</tr>
</tbody>
</table>

The basic concept of the hydraulic shovel/excavator consists of the undercarriage, upper carriage pivoting on a rotating tub and the attachment that consists of a boom, stick and shovel bucket as shown in Figure 6.17. To increase the effective application of the hydraulic
shovel, the boom and stick configuration can be changed to differentiate between a face shovel and a backhoe excavator. The backhoe version is primarily designed to excavate below level and the shovel version to load on or above the excavator level.

![Diagram of Front Shovel Configuration](image1)

**Front shovel configuration**

![Diagram of Backhoe Configuration](image2)

**Backhoe configuration**

Figure 6.17   Hydraulic excavator configurations

The hydraulic shovel/excavator was developed, in part, to compensate for the shortcomings of the rope shovel. The one major advantage over the rope shovel is that, with careful design, the forces generated by the hydraulic cylinders can be applied with maximum effect. The hydraulic shovel/excavator uses very effective crowd and breakout forces and to a lesser extent, a lifting action to perform the digging action. This implies that the digging action is not one of raking up the face but of crowding in and excavating the face from the top down or bottom up. The success of the hydraulic shovel/excavator lies in the versatility and adaptability to different tasks.

Both types are relatively new and have found increasing application in surface strip coal mining. They are often selected where digging conditions are difficult, due to consolidation or poor blasting. Compared with FEL’s, hydraulic shovels are used only for digging, not transport, and machine moves should ideally be infrequent. Thus for maximum machine productivity, the rock pile must be large so the machine does not need to frequently re-position, and a truck must be available within which to dump the rock. The back-hoe configuration
is used to dig below the level of the machine and since they have smaller buckets than the front shovel type, smaller rock pile and trucks are typical. Both machine types have a high breakout power which means they can dig quite hard, unblasted material. Figure 6.18 shows both types' excavation profiles.

![Excavation profile](image)

Figure 6.18 Excavation profile

The basic excavation cycle consists of cutting a path through the rock pile (crowding), loading of bucket on the hoist, swing towards the truck, dump and swing back to the rock pile. The two main and separate excavating movements, hoisting and crowding allows for various excavation possibilities to be applied whilst loading. Also because of the high digging breakout force of the machine, it is easier to maneuver the bucket to take selective rocks or layers while loading. The machine can be successfully applied to the excavation of medium consolidated ground without the need for blasting as would be the case for the FEL or rope shovel. (Whilst they can dig
quite hard material, in the hands of less experienced operators, high pressure build-ups for sustained periods causes adverse effects on availability and machine life).

The machine’s excavation characteristics are improved through the fine control that is available to move the bucket and to bring it into position. Advantages of this power and flexibility over other loaders are:

- Good floor control and floor cleaning (front shovel only)
- Selective removal of big rocks out of the face
- Bucket can penetrate rock pile or face at any angle, to get most advantage out of the strata and cracks
- Unstable face can be excavated from top to bottom to stabilize the face
- Shape of the face (angle of repose) can be easily maintained
- Selective mining and parting or interburden layer removal is possible

As opposed to the front-end loader, it is not necessary for the bucket to be the same width as the machine, because loading can be achieved through the hydraulic cylinders on the arm crowding the bucket. This allows for a bucket design that suits the specific material being mined, thus the breakout power (diggability) of the bucket is maximized if the bottom lip contact is minimised for the same crowding force. Buckets that are too wide can cause instability, and asymmetrical forces during loading should be avoided. For the same capacity, narrower buckets have to be deeper or higher – in both cases resulting in either the required loading height increasing to ensure proper dumping, and/or less control on the forces at the cutting edge. However, compared to the front dump bucket of the FEL, this bucket design has considerable advantages, the biggest being an increased dumping height. This is due to the fact that the front of the bucket is controlled hydraulically and lifts upwards, which means that larger trucks (higher dump height requirements) can be utilised.

Hydraulic shovels can operate efficiently on any face height up to 10-16m. The limitation on back-hoe configuration is typically 6-8 metres, although this is normally not a constraint because the machine can readily excavate higher faces in multiple passes. Very thin coal and
parting bands can cause low productivity due to the frequency of re-positioning of the machine with the rapid advance of the digging face. Neither machine type is greatly sensitive to block width and they can easily work in less than 25 metres pit working room (depends on truck type).

From the information presented earlier, it can be seen that the technology of hydraulic excavators is advancing substantially and the largest sizes are seriously challenging rope shovels. The key differences are:

- Hydraulic excavators have a lower initial capital cost than rope shovels for the same production rate, however they have a limited life. Over an extended period, the initial capital cost of the excavator plus the capital cost of major hydraulic overhaul or replacements may be more than the capital cost of the rope shovel.

- Rope shovels have a lower operating cost and the productivity of a rope shovel is less sensitive to the skill level of the operator.

Loading methods are the same for both varieties, the main aim is to follow a working plan which minimizes swing angle and hoist time. The swing angle is an important production parameter for most semi-mobile loaders; where excessive swing time is unproductive. Two loading methods are generally used to reduce swing time:

- Single side truck back-up loading
- Double side truck back-up loading.

Double sided truck loading maximises loader productivity (but reduces truck productivity since one truck must wait while the other one gets loaded), and also helps in maintaining a more productive face shape. With the single sided method, productivity is lower because of bigger swing angles especially during slot-cutting (the first cut into a rock pile to be loaded), when a new bench is sunk (drop-cut or box-cut) and the fact that the machine must wait for the next truck to get into position.
6.2.5 Electric Rope Shovel

The rope shovel with bucket sizes of 7-30 m$^3$ has historically been the primary loading tool in many large surface mines, including strip coal mines. The position is less so today with the advent of large wheel and hydraulic loaders. However, with the development of ultra-large mine haul trucks, some rope shovels of 60 m$^3$ bucket capacity are on the drawing boards to load these trucks in 3-4 passes, such as the Bucyrus 795 Electric Rope Shovel with hydraulic crowd, 135 ton nominal bucket payload and dipper capacities as required to suit material density.

Rope shovels are electrically driven and since an electric (powered) shovel is now generic in mining, the name rope shovel is in more common use for this type of equipment. They are used mainly for loading large trucks with capacities ranging from 80-280 t, the larger size more typical in overburden stripping, the smaller sizes in coal mining.

The machine is electrically driven and hoist and crowd motors are used to position the dipper (bucket). Rope shovel refers to the means by which the bucket is positioned (hoist and crowd arm) – in this case using ropes. The most important design characteristics of the rope shovel are the boom and dipper handle and the dipper itself. Figure 6.19 illustrates these components and the typical excavating profile of the machine. Various bucket and arm designs are available, including the super front type which aims to simulate the maneuverability of the hydraulic shovels – but with much less breakout force.

The basic concept of the cable shovel is to pull a bucket up the face and slice the material into the bucket. Essentially it uses a simple front-end structure and brute force for digging as can be seen from the digging profile. The boom is the structural member that supports the dipper handle and ultimately the dipper. The length of the boom determines the height that the dipper can be raised to and therefore the maximum bench height. The shovel dipper is designed for easy filling, abrasion protection whilst digging and easy emptying through the hinged rear door. Advances in dipper design allows dippers with different aspect ratios, typically 1.5:1 to 1.7:1. These high ratios allow higher fill factors in smaller face heights, leading to greater range of bench heights over which rope shovels are cost effective. While a wider dipper is advantageous to higher production rates, it does not improve the ability to load narrow seams of material.

Because of the raking action it is difficult to fill the dipper in one pass if there is insufficient face height. As a rule of thumb, the design bench height should be equal to the height above the floor of the boom point.
sheave. This should be reduced by 10% - 20% if the material is not well blasted. The minimum face height depends on the type of material but should generally be such that the digging cycle should not exceed 15 seconds. Based on the geometry of a PH 2300 shovel this equates to minimum and maximum productive face heights of between 8m and 17m.

Figure 6.19 Typical electric rope shovel excavating profile and structural members

Rope shovels have limited mobility due to the large weight of the machine as well as the restriction of the electric cable. The maximum distance that a rope shovel should be trammed for one shift’s production should not exceed 500m (up and back). This limitation imposes limits on the ability to blend different material types, which
are often required in coal mining. It also reduces the efficiency on lower bench heights because of the more rapid lateral advance required on these benches.

A typical loading cycle consists of excavation a slot, load, swing to the truck, dump and swing back. Cycle time varies between 25-35s for swing angles between 70° and 120°.

- The machine periodically gets repositioned to maximize excavation when the crowd action is used up (dipper can’t push out any further). Repositioning is a slow process and hence a machine with a long reach is preferable.

- The excavation slots are made through a combination of hoist (vertical) and crowding (horizontal) movements. Slot geometry is limited and it is usually in the form of a vertical arc from bottom to top. Floor clean up is poor with a rope shovel (and thus the use of supporting equipment is required, FEL or wheel dozer to clean and level the floor) and large rock blocks or a tight face make excavation difficult.

- Dumping is achieved by gravity only, through a door that is opened under the bucket once the bucket is in position over the truck, therefore it is not possible to control the off loading of the bucket into the truck and spillage is often problematic (where full as opposed to integer loads are required). The dumping mechanism means that the material that is excavated must not be sticking together, nor must large boulders be dug, since these materials would be difficult to empty from the bucket.

- Position of the truck must be carefully maintained, since maneuverability of the bucket is limited (especially with the double backup loading method).

A typical loading cycle consists of the components as given in Table 6.7.

Table 6.7 Typical rope shovel cycle component times

<table>
<thead>
<tr>
<th>Activity</th>
<th>Percentage of total cycle time</th>
</tr>
</thead>
<tbody>
<tr>
<td>Excavating slot to fill bucket</td>
<td>24%</td>
</tr>
<tr>
<td>Swing to truck</td>
<td>32%</td>
</tr>
<tr>
<td>Dump and empty on back swing</td>
<td>34%</td>
</tr>
<tr>
<td>Positioning for the following excavation slot</td>
<td>10%</td>
</tr>
</tbody>
</table>
Productivity is not as sensitive to block width as it is to face height. Block width minimum of 30m is preferred to allow sufficient rear-end clearance against highwalls and against the truck being loaded. (Most large trucks require 30 metres to turn anyway.) If necessary, rope shovels can swing 180° to load a truck behind. Normal design block width 50 metres.

Mostly single or double truck back-up loading methods are used, with the same comments applicable as for the hydraulic shovel. In all the cases the swing time and the truck positioning time must be minimized. Since the loader is electrically driven, special attention must be given to the trailing cable. Double back-up is difficult to apply especially in terms of positioning of the truck on the blind side of the operator (on the far side of the machine from the operators position) and in strip mining the block width is often too small to accommodate two trucks when loading the coal seam.

6.3 Cyclic Shovel Mining Methods

The effectiveness of a truck and shovel loading system is dependant on a number of factors, including the compatibility between truck and loader (the so-called ‘match’ - see later) and the loading method used. There are four major shovel mining methods used in the surface mining industry, but in strip coal mining, the most predominant are single back-up (for coal seam loading) and double backup (for overburden loading where more loading floor area is available). The methods depend basically upon the position of the shovel relative to the bench face, the position of the trucks when being loaded and the truck travel routes to and from the shovel, which is a function of loading floor available.

6.3.1 Loading Floor Area and Minimum Loading Bench Widths

Ideally, enough loading floor or block width should be available with which to accommodate the most productive loading method, typically a double back-up system. However, in strip mining, the block width (width of loading area available on the exposed coal seam) is more often a function of the overburden handling equipment selection and operating technique. This is because the control and minimisation of waste handling costs is often critical in the design and application of these systems and waste mining costs contribute a significant proportion of the total cost per ton mined.

Where no block width limitations exist (typically overburden excavation using truck and shovel, as opposed to dragline systems in
strip mining and terrace mining operations), the minimum amount of operating room required to accommodate the trucks and shovels involved in a parallel cut loading operation can be determined.

This dimension is the width of the working bench, or the bench in the process of being mined. This width (which is synonymous with the term ‘operating room’ or ‘loading floor’) is defined as the distance from the crest of the bench providing the floor for the loading operations to the bench toe being created as the bench face is being advanced. The minimum amount of operating room varies depending upon whether single or double back-up loading is used, with the latter obviously requiring somewhat more. The minimum width is equal to the width of the minimum required safety bench plus the width of the cut being taken.

As an example, assume the following parameters apply;

- Bench height = 14m, Bench face angle = 90°
- A safety berm is required.
- The minimum clearance between the outer truck tyre and the safety berm = 2m.
- Single back-up loading is used.
- Loading is by rope shovel with 28m$^3$ dipper
- Haulage is by 210 ton capacity trucks, truck width = 7m, tyre rolling radius = 1,46m.

The basic calculation entails;

**Safety berm width.** A safety berm is usually required along the edge of the bench. The height of the berm should be of the order of the tyre rolling radius. For this truck, the berm height would be about 1,5m. Assuming that the material has an angle of repose of 45°, the width of the safety berm is 3m. It is assumed that this berm is located with the outer edge at the crest.

**Distance from berm to truck centreline.** This distance is determined assuming parallel alignment. A 2m clearance distance between the safety berm and the wheels is used. Since the truck is 7m wide, the centerline to crest distance is 12m.
Distance from truck to shovel centreline. The appropriate shovel dimensions are read from the shovel specification sheet;

- Shovel centerline to truck centerline. This is assumed to be the dumping radius at maximum height, for example 10,3m

- The maximum dumping height should be more than sufficient to clear the truck body side – a smaller distance can be used where less dipper hoist is required (this will possibly also reduce loader cycle time too).

- The level floor radius dimension is the maximum distance from the shovel centerline which the floor can be cleaned. In this case typically 16,7m. This will be used as the maximum shovel centerline to toe distance.

The working bench dimension. This is found from the sum of the distances truck centreline to crest, shovel to truck centreline and shovel floor radius, i.e. 39m.

Width of cut. In this case it has been assumed that the shovel moves along a single path parallel to the crest. Information from the shovel manufacturer can be used to find the maximum cutting width, for example 24,3m. This applies to the width of the pile of broken material. Therefore, to allow for swell and throw of the material during blasting, the design cut width should be less than this value. Here a value of 18m has been assumed.

Knowing the width of the working bench and the cut width, the resulting safety bench has a width of 39 -18, i.e. 21m.

Check calculations. These should be made to check the assumptions and resulting width of cut;

- The maximum dig height of the shovel (sheave wheel at boom point) is should be greater than the bench height, so the shovel can reach to the top of the bench face for scaling.

- The flattest bench face angle which could be scaled is found from the arctangent of the difference between maximum shovel cutting radius and the maximum radius of the level floor. In this case, a slope angle of 82 means the shovel can easily scale the vertical bench face.
A somewhat simplified approach has been applied to determining working bench width. The problem becomes more complex when the optimum width from an overall economic viewpoint is required. Since the width of the working bench is approximately equal to the combined widths of the others in a bench stack set, it has a major impact on the overall slope angle. A wider working bench means that the slope angle is flatter with the extra costs related to earlier/more stripping, but the equipment operating efficiency is higher (with lower related costs). On the other hand a narrower working bench would provide a steeper overall slope at the cost of operating efficiency. Thus, there are other factors, beside those related to equipment geometries, which must be considered.

6.3.2 Loading Methods

The material being mined has little influence on which loading method is used, apart from its location in the mine. The main factors, which affect the choice of method, are:

- Bench slope (angle of repose of blasted or in-situ material)
- Loading floor area available around mining face and mining blockwidth
- Coal grading or blending constraints
- Operator skill level.

Any of the previously mentioned shovels can be used with these methods, but since the crawler-mounted pivotal loaders (rope and hydraulic front shovel) are less mobile than the FEL, these are used in the following discussion since the shovel position relative to the face is critical to avoid unnecessary and time-consuming repositioning.

**DOUBLE BACK-UP**

With this method the shovel crawlers are aligned at right angles to the face and the shovel digs straight ahead towards the back wall of the blast. Trucks reverse in and are loaded on both sides of the shovel. Any truck waiting time is used to position the vehicle ready for loading before the previous truck moves away. Hence minimized shovel waiting and maximum output (BCM/hr) is achieved. Truck travel routes to the shovel, as well as the position of shovel trailing cables and bridges are important (Figure 6.20 shows the sequence of operation with the loader moving from block 1-2-3 as each is excavated).
Double sided loading (provided there are sufficient trucks) basically eliminates the spotting delay - allowing the truck to be in position before the previous truck has finished loading. With front-end loaders, there is virtually no spotting delay anyway. Also, for shovel digging in hard faces, even with single sided loading there is often no lost time because the shovel gainfully uses the time between trucks for face clean-up and breaking out material. Table 6.8 summarises the main points of this method and Figure 6.21 illustrates the results of a simulation comparison between single and double sided loading for an optimum fleet size of 5 trucks, showing a 15% increase in productivity by using double-sided loading.

**SINGLE BACK-UP**

When a narrower face area is unavoidable, making it impossible to spot trucks either side of the shovel, a single back-up system can be used (Figure 6.22). This method is typically applied in strip coal mining operations by rope shovels and trucks loading the exposed strip of coal.
At optimum fleet size, difference in production - tempo = 15%  

Figure 6.21 Simulation results for a change from single to double sided loading.

Table 6.8 Double-sided loading method

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>• Overall swing angles are low</td>
<td>• Greater driver skill required to spot trucks - especially on blind side of truck or loader with bottom dump trucks</td>
</tr>
<tr>
<td>• Truck waiting time utilized for spotting</td>
<td>• Shovels load on blind side</td>
</tr>
<tr>
<td>• Well suited for mining irregular shaped faces</td>
<td>• Frequent moves required on a low face, therefore not suitable</td>
</tr>
<tr>
<td>• Dozer can clean up on one side while loading on other</td>
<td>• Shovel moves require more frequent tractor assistance</td>
</tr>
<tr>
<td>• Oversize can be moved from face with little disruption</td>
<td>• Pit floor grade is harder to maintain</td>
</tr>
<tr>
<td>• Trucks traveling over spillage in loading zone is limited</td>
<td>• A cable bridge is required</td>
</tr>
<tr>
<td>• Shovel position good for safety</td>
<td>• Adequate mining area is necessary</td>
</tr>
<tr>
<td>• Power cable is well clear of face and fallen rocks.</td>
<td>• A long truck spotting time results if the shovel is waiting to load when truck arrives (typical of an under-trucked operation).</td>
</tr>
</tbody>
</table>
Other typical application examples are:

- A bench slot blast (similar to box-cut)
- Berm blasting or bench pushback or drop-cut, usually when a pit is deepened
- Sinking a ramp.

The importance of having sufficient pit working room for double sided loading has long been a factor in truck and shovel pit design. Clearly, if the loader does not have to wait, its productivity is improved. However the importance of this is not as great as perhaps it once was due to changing economics of truck and loader system design.

In the mid-20th century, the capital and operating cost of rope shovels was very high in comparison to the associated trucks. In these circumstances, the most economic system always had the trucks queuing at the shovel - and in such an ‘overtrucking’ case, productivity was greatly improved by double sided loading. Nowadays there is not such a large relative difference, and truck and shovel systems become less economic if too many trucks are assigned to the shovel and the advantages of double sided loading are not as great (or at the very least need to be evaluated case by case). Table 6.9 summarises the main points of the single sided method.

6.4 Transport Equipment – Cyclic

In this section, cyclic haulage equipment is introduced, reviewed and a selection methodology defined. Large volume off-highway or mine haul trucks are an ideal solution in providing the flexibility mine planning and scheduling often require. In addition, with more demanding rehabilitation regulations and the very high cost of large walking draglines, truck and shovel systems are becoming more popular for overburden handling in preference to draglines in surface strip coal mines. Truck and shovel systems remain the dominant system for coal seam mining.
### Table 6.9  Single-sided loading method

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>▪ Caters for restricted mining areas</td>
<td>▪ Shovel can load on one side only</td>
</tr>
<tr>
<td>▪ Caters for irregular shaped faces</td>
<td>▪ Increased swing angles over double back-up, therefore lower productivity</td>
</tr>
<tr>
<td>▪ Pockets can be selectively mined to help grade control</td>
<td>▪ Long truck spotting and change over time reduce productivity of both shovel and trucks</td>
</tr>
<tr>
<td>▪ Trucks travelling over spillage in loading area is minimized</td>
<td>▪ Dozer clean up will stop loading operation</td>
</tr>
<tr>
<td>▪ Excellent regarding shovel safety</td>
<td>▪ Not suited to low face heights as frequent moves required</td>
</tr>
<tr>
<td>▪ A cable bridge is not essential</td>
<td>▪ Shovel moves require more frequent tractor assistance</td>
</tr>
<tr>
<td>▪ Less driver skill required than with double back-up system (no blind-side loading).</td>
<td>▪ Pit floor grades more difficult to maintain.</td>
</tr>
</tbody>
</table>

**Figure 6.22**  Single back-up loading method (showing bench slot blast application)
There are two basic designs of rigid frame haul truck in use as summarised in Table 6.10, mostly in 2-axle format for rear dump and three axle format for bottom-dump versions. The articulated dump truck - ADT - also falls into the category of a three axle 4- or 6-wheel drive truck (but at present is limited to 50t capacity – and does not have significant application in medium to large tonnage applications in surface strip coal mining).

The greater the number of axles a truck has, the better is its ability to handle poor ground conditions, but less is the maneuverability of the truck and with an articulated bottom dump truck the maneuverability is reduced even further. Thus the type of truck can have a significant influence on the loading method chosen.

The ADT was developed for transport under difficult conditions. Its mobility is such that it can take the shortest distance between two points, with little need for well-maintained haul roads. This leads to high efficiency and low cost per ton, which forms the foundation of successful haulage. The basis of the ADT is the connection between the front and rear frame. This makes it possible for tractor and trailer to move independently of each other, which drastically reduces stresses on the frame when driving off-road. Rigid haulers are fine on paved or graded roads, but on poor roads, ADTs are often used. An articulated truck's configuration allows it to turn 45° left or right, giving the operator a great deal of maneuverability in tight situations and, coupled with their ease of loading and high payload-to-weight ratio make them useful in short term overburden, sand, and hauling operations where a formal road would be expensive to provide. They are, however, limited in capacity.

Table 6.10  Haulage truck types

<table>
<thead>
<tr>
<th>Truck type</th>
<th>Load (t)</th>
<th>Drive power(kW)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rear-dump</td>
<td>50 - 350</td>
<td>300 - 2250</td>
</tr>
<tr>
<td>Bottom-dump</td>
<td>70 - 240</td>
<td>350 - 1100</td>
</tr>
</tbody>
</table>

Figure 6.23 shows a typical rear dump truck (RDT) and Figure 6.24 a bottom dump truck (BDT), the latter developed specifically for shallow surface mining operations.
The smaller series of trucks are generally mechanically driven, whilst the larger trucks are equipped with electric wheel motors in the rear axle, driven by a diesel-generator in the truck. The “electric” drive haul trucks are generally those with more than 130t capacity, as a result of the drive power and torque required. Recent developments in drive-train technology have enabled most manufacturers to offer mechanical drive on the larger trucks which competes directly with the electric systems. The subject of mechanical or electrical drive is still hotly debated. On the largest capacity trucks (280t capacity and upwards), electric drive is usually the only option – the limit in truck sizes offered being related to tyre technology limitations rather than drive train issues.
The general characteristics of the RDT and BDT are summarised comparatively in Table 6.11.

Table 6.11 Comparative analysis of RDT and BDT

<table>
<thead>
<tr>
<th>TRUCK TYPES</th>
<th>RDT</th>
<th>BDT</th>
</tr>
</thead>
<tbody>
<tr>
<td>Load to vehicle mass ratio (t:EVM) (Empty vehicle mass)</td>
<td>1,45</td>
<td>1,70</td>
</tr>
<tr>
<td>Propulsion to vehicle mass ratio (kW:EVM)</td>
<td>350</td>
<td>145</td>
</tr>
<tr>
<td>Propulsion per ton load (kW/t)</td>
<td>9,5</td>
<td>7,6</td>
</tr>
<tr>
<td>Slope climb ability (sustained ramp grades)</td>
<td>Max≈12%</td>
<td>Max≈8%</td>
</tr>
<tr>
<td>Turn radius</td>
<td>1,2 x length (2 axles)</td>
<td>1,80 x length (3 axles)</td>
</tr>
<tr>
<td>Body resistance to shock load</td>
<td>Good, must be free flowing rocks.</td>
<td>Poor, must be free flowing rocks. Large rocks damage the bottom door.</td>
</tr>
<tr>
<td>GVM (gross vehicle mass) distribution: 2 axle (%) (front and back) 3 axle (%) (front, drive and trailer)</td>
<td>32, 68</td>
<td>15, 40, 45</td>
</tr>
</tbody>
</table>

Factors to consider when choosing a haul truck are:

- Overburden or coal characteristics
- Transport route (grade, length and curves)
- Mine production requirements (ore or waste tonnages).
- Maneuvering space (especially turning)
- Dump conditions and dump design
- Mine road surface condition
Compatibility with existing fleet

Compatibility with loaders (match primarily, but also dipper dump height, truck body shape, etc.

Again, a life cycle costing approach is relevant here, but must not obscure the primary selection factor - that of “match” or the compatibility between the truck and the shovel loading it. The loading time per truck and the loader productivity are effected by the number of bucket loads or “passes” required to fill the truck. Too many passes (shovel too small or truck too big) leads to poor truck productivity, whilst the opposite leads to truck overloading and under-utilisation of the shovel’s capacity.

For strip mines, short ramps are used from the coal floor to the surface, but not to such a depth as open pit mines, and since the road also runs on the flatter surface too, either bottom or rear-dump trucks can be considered (although many new mines prefer the rear-dump truck for the other reasons stated in Table 6.11). The required road standards for the bottom dump trucks must be higher and if the ramp grade keeps on changing (with changing mining faces or concurrent rehabilitation), it requires a lot of road preparation and maintenance which is expensive.

To summarise, the main factors to consider when making a choice between BDT and RDT, are based on the following; analysis of each units gradeability, maneuverability, tyre choice, robustness, load material, dumping, loading and braking characteristics.

**RDT**

**Gradeability.** Good - high power to load ratio together with a good load distribution over the driving wheels makes it suitable for grades upto 12%.

**Maneuverability.** Good - short wheelbase allows unit to maneuver easily in the pit.

**Tyres.** Possible problems due to heat build up in abrasion resistant tyres - especially if used on long (+10km) hauls. Tyre loads high.

**Robustness.** Structure suited to tough working conditions and impact loading.
**Load material.** Can handle virtually any size of rock, wet or dry. For high density materials, a smaller body can be fitted, but low density materials requiring larger bodies may make the vehicle unstable.

**Dumping.** Free dumping good, but at hopper or discharge point it must reverse and spot load. Typical dump time 45 - 60 s.

**Loading.** High loading height may be a problem for LHD’s and result in irregular load distribution (load must be centralised in body). Better suited to rope or hydraulic shovels.

**Braking.** Good - short wheel base and axle spacing may induce skid on wet roads, but no jackknife problems.

**BDT**

**Gradeability.** Poor - generally limited productively to 7-8% on longer ramps or slightly steeper for short distances only.

**Maneuverability.** Good turning circle, but not on soft ground (for any maneuvers). Reversing is not easy.

**Tyres.** Good tyre loading and little heat build up, but more tyres per unit required (10 as opposed to 6 on RDT)

**Robustness.** Generally lighter construction suited to less rough work and well prepared loading and hauling areas. First load lands on discharge doors which can be damaged.

**Load material.** Suited to dry free flowing material, no large rocks.

**Dumping.** Free dump slow if not impossible, but at hopper very fast since no maneuvering is required.

**Loading.** Low loading height suits many loaders, but problem with large rocks damaging doors. Loading floor must be clean and maneuvering in this position is slow.

**Braking.** Tendency to jackknife on slippery surfaces.
6.4.1 Selection of Tyres

After fuel costs, truck tyres are the second most important cost item in the contribution to total cost per ton hauled. With a typical 250t truck fully laden to 400t (GVM), each back wheel (of 4 on a dual assembly) carries approximately 65t-70t and dynamic truck-road interaction adds about 20% extra to this amount.

Usually the front axle (the drive axle) is equipped with only single tyres while the back axle (on RDTs) or the horse drive and trailer axles (BDTs) are equipped with double (dual) tyres because of higher axle loadings. Heavy duty tyres are expensive items, typically 6-11% of the capital cost of a new truck, for a set of tyres and their life is ± 6 000 – 10 000 hours.

On longer haul routes, tyre design limitations effect the productivity (specially due to over-heating of the tyre, etc.) of single wheel rear-axle trucks. Under these conditions, double wheel rear–axle trucks are a better choice since the equivalent single wheel load (ESWL) is smaller and heat build-up reduced. The heat build-up limitation is known as the tyre ton.km/hr rating.

To choose the correct tyres depends on a number of factors. The tyre size itself is a function of the width (given in inches), radial or cross-ply construction and the rim size. For example, a 41.00-R65 tyre has a width of 104cm (41”), is built on radial (R) strengthening and fits on a rim of 165cm (65”) (tyre diameter is approx. 2,4m).

Other factors to consider are:

- Load
- Speed
- Durability
- Performance
- Cost.

Which are inter-related to a number of other factors as shown in Figure 6.25.

Most important of the criteria is the load and speed of the tyre, too much load on the tyre or too much speed will lead to failure of the tyre because of the heat load. Most tyres are made by vulcanising the rubber at 132ºC and thus if higher temperatures are generated the tyre will fail. Because of the high load on mine truck tyres, the repair of punctures is generally not possible, and the use of re-treads is only
a recent untested phenomenon. To keep heat accumulation within limits, the tyre must be selected according to the haul \( t.km/hr (tkph) \) grading.

Figure 6.25 Tyre selection criteria
The TKPH is given as:

\[ TKPH = \frac{\text{Average wheel load} \times \text{average truck speed}}{\text{speedtruckaverageloadwheelAverageTKPH}} \]

where

\[ \text{Average wheel load} = \frac{\text{Mass of load} + \text{mass of truck}}{2} \]

with due regard the load distribution between front and back axles of the truck.

As an example, take an RDT truck with a laden weight of 300t (GVM) and an unladen weight of 120t (EVM). The back axle is equipped with double wheels which carry 68% of the GVM. If the truck runs on a 5km haul (return) running 6 trips per hour, the minimum tyre TKPH rating is 765, comprising 30km at an average tyre load of 25.5t.

### 6.4.2 Mine Haul Roads

Surface strip coal mining uses haul roads to access the coal seam from the surface, using either highwall or low wall ramps, as described in an earlier module. The design and construction of the haul road is an important factor in the overall productivity and cost of hauling rock - strip mines spend about 7-15% of their total costs on coal truck transport, thus the road on which the truck is travelling must ensure:

- The lowest costs for the transport of material over the life of the mine
- A safe and quick access to and from the mining areas
- That it avoids unstable or soft areas of sub-grade
- Long life - to minimize construction and material cost (R300 000 to R1 500 000/km to build a new haul road)

Even though most mine roads are unpaved, the road surface conditions are of extreme importance. A typical haul road (or more importantly, the sub-grade or in-situ below the road) must be capable of withstanding ESWLs of 50–90t. Special construction techniques are required, which usually consist of placing a 0.3– 0.6m broken rock layer on top of the in-situ material and a 0.10- 0.15m layer of wearing course as shown in Figure 6.26.
A number of other factors must be taken into account with the design of haul roads, as listed below:

**Amount of lanes** – a function of transport volumes, truck speed and sight distances

**Width** – For two-way traffic the width must be 3–4 times the width of the truck, i.e. 25-30m road width. For the curves an extra 10–20% must be added (especially for longer bottom-dump trucks).

**Gradient** – The maximum gradient is in the vicinity of 8%-15% (4,5°-8,5°) for sustained gradients. Higher grades cause a drop in truck productivity, reduced engine life and safety limits on the downhill run (brake system capacity). A flatter grade causes a reduction in productivity because of a longer route length. Typical travel time for a specific truck can be determined from its productivity graph. It is usually seen that long flat roads are ‘slow’, short very steep roads are also ‘slow’. The optimum lies somewhere between these two extremes. In general, rear-dump trucks have a better hill climb ability than the slightly under-powered bottom-dump trucks.

When the distance to be hauled is approximately independent of the ramp gradient, there is little to be gained from using steeper gradients than strictly necessary - moreso in the case of RDTs. The above conclusions should not, however, be universally accepted without reference to other factors - especially the haul road resistance effect.
An increase of 1% resistance means an equivalent increase in grade (on the ramp) of 1% but also a reduction in acceleration on the flat too.

**Road surface** - Tyre cost increases because of the wear, impact and penetration on the tread and sidewalls. The most important factor to take in account is rolling resistance and it is a function of the material used to build the road surface or wearing course.

**Rolling resistance** - The amount of power used to overcome the resistance to motion caused by:

- Tyre deforming under the applied load
- Tyre penetration into the road surface because of too soft a road surface and/or the deformation of the road structure under the wheel load.

A basic formula for rolling resistance (RR) assessment that can be used is:

\[
RR = 0.02 + 0.0007B_p
\]

Where:

- \(RR\) = Rolling resistance (RRx100 gives rolling resistance as a percentage of GVM)
- \(B_p\) = Tyre penetration into the road surface (mm)

Typical values are 1-1.5% for tar, 3% for a well designed mine road, 2-4% for a compacted gravel surface, 5% for loose rocks, 7,5% for wet soft gravel.

The total rolling resistance which comprises the road rolling resistance and the slope (grade) resistance (effect of gravity - negative for uphill road) is used to determine the response of the truck to the road. Computer simulation can be used to demonstrate the effect of rolling resistance. Less than 4% total rolling resistance is ideal, greater than 8% is unacceptable (except on very short ramp roads) and 10% total rolling resistance will:

- Reduce hourly production with 47%
- Double the amount of trucks necessary (for the same t/hr production)
- Double the truck transport costs (R/t).
Curvature - must be maintained to keep good line of sight within the truck’s maximum stopping distance. Sharp curves will reduce the truck speed, which will result in a reduction in productivity and an increase in costs. Curves must be correctly elevated otherwise the trucks will slide at the apex of the curve. Tyre damage will develop if the amount of the super elevation is wrong.

Maintenance - Haul roads require continuous maintenance and there is a number of points to consider;

- Road surface routine maintenance; cross-fall, camber and longitudinal alignment, compaction, watering and replacement of lost material (± 20 mm per year). Rough surfaces give higher rolling resistance values which increase transport costs and thus scraping of the road every 2 – 3 days is necessary depending on the transport and surface type.

- Dust reduces the visibility and damages the engines and leads to a weakening and degradation of the road surface. The road is regularly sprayed wet to reduce dust (at about 0,5l/m²) and dust palliatives can be under certain circumstances applied if it can be justified from a cost perspective (reduces the use of water and the operating cost of the water truck).

- Moisture control; to allow the water to run off the road the slope (camber) of the road must be correct. Too much water causes the formation of potholes and an increase in the rolling resistance. 1½-2% is usually the best camber, except on a curve where super-elevation is applicable. Where road width and rainfall not problematic, a cross-fall from one side of the road to the other can be used, which assists in load sharing between wheels.
6.5 Factors Effecting Cyclic Systems Productivity

The selection of the most effective equipment combinations for a particular excavating and hauling application requires an estimate of the productivity of each component of the various possible truck/shovel combinations.

Depending on the level of detail required for the study, the process of productivity estimation can be tailored to generate suitable levels of sophistication;

- Long term planning work - Quick general estimation techniques are sufficient. Level of accuracy typically about 10%.

- Shorter term operation optimisation - Computer based modelling techniques are used to accurately define optimum choice of equipment for a particular mining operational environment. This method implicitly assumes that (for all but the most simple of applications) some form of truck allocation method will used to ensure productivity shovel and truck remains optimal.

For the purposes of this section, the general estimation technique will be illustrated, followed by a discussion of factors effecting productivity. The productivity of the system is assessed either in terms of the expected productivity of an established system, or the required fleet to maintain a design productivity. The component activities are discussed below.

6.5.1 Basic System Productivity Calculation

LOADING PRODUCTIVITY
Loader productivity is usually estimated in conjunction with an associated truck fleet, however for quick estimates on a stand-alone basis of the productivity of various loading shovel and truck combinations, the following example is given. Data is divided into three categories, namely:

1. Material data
2. Loader data
3. Truck data.
Basic data required is given in Table 6.12

Table 6.12 Basic shovel productivity data

<table>
<thead>
<tr>
<th>Material data</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Rock in-situ density (^1) (t/BCM)</td>
<td>2,7</td>
</tr>
<tr>
<td>Rock swell factor (^2) (%)</td>
<td>20</td>
</tr>
<tr>
<td>Rock bulk (loose) density (^3) (t/m(^3)) or (t/LCM)</td>
<td>2,35</td>
</tr>
<tr>
<td>Rock loaded swell factor (^4) (%)</td>
<td>33</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Shovel data</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket capacity (m(^3))</td>
<td>30</td>
</tr>
<tr>
<td>Bucket fill factor (^5)</td>
<td>1,00</td>
</tr>
<tr>
<td>Cycle time (per bucket loaded) (s)</td>
<td>33</td>
</tr>
<tr>
<td>Spot time (per truck) (s)</td>
<td>25</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Truck data</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Capacity (t)</td>
<td>172</td>
</tr>
<tr>
<td>Operating time (mins/hr)</td>
<td>50</td>
</tr>
</tbody>
</table>

Notes:

1. Rock density measured in-situ i.e before blasting takes place. In this case the term bank cubic meters (BCM) is used to describe the volume of the rock in-situ.

2. If the rocks is blasted it will swell because of the breaking process which introduces air voids between the fragmented rocks. In this case loose cubic meters (LCM) describe the blasted (loose) rock volume. The swell factor is established as:

\[
f = \frac{\text{In-situ density}}{\text{Loose density}}
\]

and is usually expressed as the percentage in excess of 1. For example, where in-situ density is 2t/BCM and loose density 1.7t/LCM, the swell factor \(f\) is 17% and \(f = 1.17\).
3. Bulk or loose density of rock calculated from swell and in-situ density.

4. Rock will swell or compact when it gets loaded and this effects the density of the rock (and thus tonnage) in the shovel bucket. A similar calculation applies to that given in (2) and the exact swell or compaction is usually a function of bucket dimensions.

5. Depending on the type of bucket on the shovel, different bucket fill factors are obtained. It is mostly ascribed to the aspect ratio of the bucket: - buckets with a large surface area (typical FEL) can carry more than their capacity due to the heaping of the rock which is a function of it's angle of repose (dry sandy types typically 15° - 20° and rock 35° - 40°). Capacity is mostly given as struck m$^3$ (loaded with rock level to the lip of the bucket, (bucket factor = 1)) or SAE heaped (above the lip of the bucket). The fill factor refers to heaped m$^3$ when the rock is heaped up in the bucket (bucket fill factor > 1).

Calculation of shovel productivity:

**FIRSTLY**

The actual bucket tonnage must be calculated according to different swell and fill factors of the material loaded.

**SECONDLY**

It must be decided which loading method will give the best shovel productivity. Two options are possible:

1. Load truck completely full (e.g. truck can carry 3½ bucket loads, so shovel loads 3 full and 1 load half full), i.e. 4 cycles

2. Load truck with integer bucket loads only (e.g. truck can carry 3⅓ bucket loads, but shovel loads only 3 full bucket loads), i.e. 3 cycles.

Calculation sequence is as follows:

1. Rock loose density (t/LCM) from in-situ density and swell factor gives 2.35t/LCM
2. Rock loaded density from loose density and bucket swell factor gives 1.70t/LCM loaded

3. Actual bucket capacity from rated capacity, loose density loaded and bucket fill factor gives 52.8t per bucket load

4. For a 172t capacity truck, the number of bucket loads is given by truck capacity divided by actual bucket capacity, giving 3.25 bucket loads

5. For a full load, the cycle time is calculated for 3.25 bucket loads. This is actually 4 loads but the last load is only 0.25 from a full bucket and the truck is loaded with 172t

6. For integer loads, the cycle time is calculated for 3 bucket loads only and the truck is loaded with 158.4t only

7. Calculate productivity for each option by considering total cycle time to fill truck. The cycle consists of spot time (to position truck correct under the shovel – boom point) and the time per bucket load (load, swing, dump, swing back). Thus for four bucket loads \( [28 + (4 \times 33)] = 2.67 \text{min} \) is necessary to load 172t and thus for three (integer) bucket loads 2.12min is necessary to load 158.4t

8. Full loads:

\[
\frac{50 \times 172}{28 + (4 \times 33)} = 3220 \frac{t}{hr} = 1370 \frac{LCM}{hr} = 1192 \frac{BCM}{hr}
\]

9. Integer loads

\[
\frac{50 \times 154.8}{28 + (3 \times 33)} = 3735 \frac{t}{hr} = 1589 \frac{LCM}{hr} = 1383 \frac{BCM}{hr}
\]

In terms of the loader-hauler ‘match’, a shovel should load a truck in 2-3 passes for short hauls and 4-7 passes for long hauls (as a general rule). Hence select truck to match loader and problem geometry if possible.

The “best” shovel productivity in this example is thus with integer bucket loads (truck not fully loaded), but take note, it is perhaps not the lowest cost option because of the fact that the trucks transport 8% less rock per trip. The final decision is calculated on a combination of costs/t loaded and cost/t transported with the view to minimizing total
cost. It doesn’t help to send a truck away with for example 92% of its total load if there is no other truck waiting, because then the shovel stands. That is known as an under-trucked operation. On the other hand, under certain circumstances it is better to send the truck away with only 92% of it’s load if there is other trucks waiting - the over-trucked condition.

The importance of dipper size is most critical where the ‘nominal’ match falls in the range between 25% and 75% of the ‘last’ dipper load. By increasing slightly the capacity of the bucket on the loader, significant productivity improvements can be realised. In this example, by increasing bucket capacity to 33.9m³ (an increase of 13%, equivalent to an increase in bucket load of 4.5t or 8%), an increase in productivity of 25% can be gained by integer loading the truck to manufacturers specified capacity. This is often a cheap and practicable method to optimise productivity of the system.

HAUL PRODUCTIVITY
Components of the haul cycle, excluding spotting and waiting at the loader, are;

- Turn and dump
- Haul and return

Turn and dump times are very site specific and conditions to avoid are limited reversing room, poor road conditions and difficult tip site combinations. The type of hauler also effects the turn and dump times with ADTs and RDTs being slightly slower to tip than BDTs (assuming the tip is suitable for bottom dumping).

Haul and return time calculations are based on the vehicles performance and the haul road segment rolling and grade resistance as discussed previously. To determine the haul and return productivity, reference is usually made to the manufacturer’s productivity estimation charts, rimpull charts and brake/retarder performance charts. These graphs are based on simulation results for specific model types and tyres used, in which the vehicle’s performance is determined for specified hauling parameters. This estimate is normally done by computer simulation programs such as FPC, TALPAC, etc. The above graphs only apply for calculations over segments where the acceleration is a constant. Since the acceleration may vary greatly during the course of a complete cycle, calculation of the total travel time requires the haul route to be subdivided into components such that each component has the same...
characteristics, and is small enough so that acceleration can be considered a constant throughout the truck travel time within that component. Two sub-divisions are undertaken;

- Subdivision of the haul route into segments where the basic operating constraints are constant over the segment (grade, rolling resistance, etc.).
- Subdivision of each segment into as many additional sub-segments as are necessary such that the acceleration can be considered a constant.

In all programs, the initial sub-division (haulroad segments) are input by the user, together with specific limits, such as speed, oncoming traffic, single or dual road width, etc. From this data, a truck travel time is estimated for the laden and unladen haul components.

**SYSTEM PRODUCTIVITY**

The cycle times for the load and haul sections can now be summed and a job efficiency factor applied. This is in effect an operating hour correction which may vary from 55mins to 40mins in poorly organised environments. The correction is applied to calculate cycles per hour, from operating hour divided by total cycle time.

The available operating time and utility corrections are then also applied, as discussed in an earlier Module;

\[
\text{Available operating time} = \text{Shift time} - \text{maintenance and mechanical/ electrical downtime}
\]

\[
\text{Availability} \% = \frac{\text{Shift time} - \text{maintenance and mechanical/ electrical downtime}}{\text{Shift time}}
\]

\[
\text{Utilisation} \% = \frac{\text{Available operating time} - \text{non load/haul activity time}}{\text{Available operating time}}
\]

Availability, utilisation and performance can be represented by a systems’ inherent and realised capacity. Inherent capacity refers to the maximum production capability of equipment, design, people, processes, and environment components combined. It effectively defines the capability limit that is potentially achievable. Realised capacity refers to the mine’s current output level and is a function of availability, utilisation, and performance. Realised mine capacity may approach but never surpass the inherent capacity limit. Deficiencies in the utilisation, availability and performance of the system, or one or
more individual subsystems, result in the gap between the inherent capacity limit and the realised capacity (lost capacity or improvement potential) as illustrated in Figure 6.27.

![Figure 6.27 Potential activities available to increase inherent system capacity](image)

In order to harness the improvement potential, the gap between realised and inherent capacities must be reduced by typically optimising asset utilisation. This results in maximising system efficiencies and overall mine productivity. The ability to accomplish this is largely the motivator for truck and shovel (and overall mine) management system development. Taking the utilisation issue as an example, significant increases in realised capacity can be made by adopting computerised (dynamic), as opposed to manual (static) truck dispatch or allocation and control systems. The interplay between various truck/shovel combinations results in two scenarios;

1. More trucks because loader ‘waits’ for truck
   - Loader productivity is maximised (no waiting for truck)
   - Truck fleet productivity is reduced due to possible queuing at loader
2. More loaders because truck queues for loader

- Loader productivity reduces
- Truck fleet productivity increases
- More mining faces necessary to accommodate loaders – complicates mining plan or needs larger pit room.

A static dispatch system is clearly not the best way to optimise transport operations of a mine. A dynamic dispatch method, which includes computer simulations is better, but is critically dependant on the production information (loading tempo, travelling times, truck condition and position, etc.) and production and management objectives and control to make a dispatch decision, as shown schematically in Figure 6.28.

![Figure 6.28 Basic approach to truck control by computer](image)

The main object of truck allocation systems is to:

- Maximize the loader productivity, while
- Maximizing transport productivity and
- Maintain a balance in the application of trucks and loaders (park extra truck or loader if necessary)
- Satisfy the mine’s production requirements (tons of ore and waste to be mined per day) at the lowest overall cost per ton.
- Satisfy the mine’s production sourcing and tonnage scheduling requirements
The comparison process between input and decision process must in general aim to reduce waiting time to the minimum and to maintain production control according to waste and ore tonnage (and different ore tonnages where ore mixing takes place). The organisation system used to manage the decision making process is shown in Figure 6.29.

![Organisation system used to manage decision making process.](image)

To improve production it is necessary to reduce waiting times right through the production cycle. Waiting time is usually experienced where:

- Loader waits for truck
- Truck waits long while it is loaded (especially when the loader is too small) or a poor match exists
- Truck waits at single carriageway road or at entrance to trolley line
- Truck waits at dumping point (waste, stockpile or ore)
- Truck waits for loader (repositioning)
- Loader and/or truck wait for service or maintenance (scheduled services during the shift)
Loader waits for support equipment to move, or to clean loading area.

Another important aspect of the central dispatch process is the ability to accommodate ‘exceptions’ and, when the unit is available again, to re-establish it as fast and efficiently in the system as possible, e.g:

- Trucks that go to the wrong loader
- Loader break down (temporarily)
- Truck break down (temporarily)
- Mechanical service or fuel required
- Truck tyre t.km/hr limitation on haul route
- Production disruptions by blasting.

This is where the gathering of real-time information and quick assimilation of data is important. Various computerized dispatch systems also have the facility to monitor the machine productivity and availability. Daily, weekly and monthly summaries can be used by mine management to bring their planning up to date and to change the logic of the central allocation control (usually priority changes). Quite often however, the amount of information generated by these systems is excessive and some sifting of information is required to enable the basic facts to be presented as a management tool for improvement, rather than purely as a source of monitoring data. Typical features of advanced asset communication and allocation systems are:

- Flexible production and equipment time data capture and analysis
- Equipment tracking
- Monitoring of mine consumables
- Data management capabilities for reporting standards using relational database and online analytical processing solutions
- Next-generation maintenance monitoring and management
- Payload monitoring
- Global Positioning System (GPS) and HP-GPS applications
- Real-time production management
- Interface capabilities to original equipment manufacturer sensors.

Through the implementation of these types of systems, the gap between realised and new inherent capacity can be reduced and design capacity increased. Integration between operations, engineering and planning, and maintenance can be achieved. The ability to accurately capture, monitor, trend and analyse data can result in a focus on key performance indicator (KPI) measuring and analysis, providing mines with a much better insight and understanding of their cyclic excavating and transport system operational practices and processes.

6.5.2 Factors Effecting System Productivity

Some of the more critical areas of truck and loader system design are given below.

**TRUCK AND LOADER MATCH**

An approximate indication of a good truck/shovel match was given in section 6.5.1 for long and short hauls. As mentioned previously, the number of trucks assigned to a loader is also dependent on the length of haul trip and the method of loading. The correctness of the match is more critical for small fleets with short travel times than for larger fleets. Also, the effect of shift start and end effects is such as to decrease productivity slightly for the trucks on longer hauls.

The definitive method of determining match is, however, based on an economic assessment. The most reliable method for assessing this is the discounted average cost method. With this technique, the cost is the price ‘charged’ to move the quantity of material, assuming normal commercial terms, for example, repaying capital investment, paying tax, profit, etc. The basic elements in such an analysis are given below in Table 6.13. Refer to a later Module for definitions of the financial terms and derivation of the analysis.
Table 6.13  Typical data requirements for a discounted cash-flow analysis of optimum truck and loader match.

<table>
<thead>
<tr>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Capital Cost</td>
</tr>
<tr>
<td>Trade-in Value</td>
</tr>
<tr>
<td>Depreciation Rate</td>
</tr>
<tr>
<td>Operating Cost (per Operating Hour)</td>
</tr>
<tr>
<td>Operating Life (in Operating Hours)</td>
</tr>
<tr>
<td>Depreciation Method</td>
</tr>
<tr>
<td>Marginal Tax Rate</td>
</tr>
<tr>
<td>Required Return on Investment</td>
</tr>
<tr>
<td>Scheduled Operating Time per Year</td>
</tr>
<tr>
<td>Shift Delays in Operating Time</td>
</tr>
<tr>
<td>Loader Availability in Scheduled Shift Time</td>
</tr>
</tbody>
</table>

The method is very specific in terms of depreciation rate and method, required return on investment, truck and loader cost and operating cost and life, but in general terms;

- If too few trucks are used, the capital and operating cost of the loader must be borne by a very small quantity of product (or revenue generating material - even waste). In this case, the cost per unit excavated is high.

- If too many trucks are assigned to the loader, this leads to the situation where the productivity of the extra truck compared to the cost of running it becomes higher than the fleet average including the cost of running the loader. The optimum fleet size occurs thus when overall cost is minimised.

Note that an optimum truck match would, in reality, represent an average operating fleet of several more units as a result of availability, breakdowns, etc.

**MINE LAYOUT AND LOAD ELEVATION**

The most significant factor affecting the productivity and costs of running trucks is load elevation. Efficient mine layout therefore requires a minimum amount of load elevation, sometimes even at the expense of additional removal of waste material for improved haul routes.
In a strip mining operation, as the mine deepened from 25 metres to 77.5 metres, BDT productivity declined from 337 tph to 238 tph. This represents a productivity reduction of 30% for 52.5 metres of additional elevation, or approximately 6.5% lower productivity for each extra 10 metres of elevation. From the analysis of cyclic transport systems, it may be anticipated that if RDT trucks were used, by virtue of their better grade climbing ability, this decrease in productivity could be offset somewhat.

In the case of coal haulage in strip mines, this increase in costs is simply a fact of life - there is very little that mine planners can change about the location of the coal in the ground. However in the case of waste haulage, haul routes are very much under the control of the planner, and are influenced greatly by such factors as;

- location and shape of outside dumps
- proportion of waste material placed in outside dumps compared to in-pit refill, and the system balance determined by shallow benches generating waste to dumps and deep benches generating waste to in-pit dumps
- shape and length of refill benches compared to excavation benches
- vertical distance between in-pit dumping levels, and
- positioning of access ramps within refill spoil areas.

The cyclic truck and loader system is certainly the most flexible method of large scale earthmoving used in strip mining. By comparison, other large scale earthmoving methods are economic only under much more restrictive circumstances. However, the efficiency and cost-effectiveness of trucks is sensitive to many more factors, but often to a lesser extent. The combination of the factors given above, especially load elevation, can result in truck and loader system costs more than double those of seemingly similar systems and requiring a mine to re-evaluate the transition point from cyclic to continuous systems changeover.
6.6 Loading Equipment – Continuous

The transition to continuous mining and rock handling systems has already been briefly discussed. Surface mining is increasingly taking place at greater depth, which means that to make it economic, more overburden must be moved at lower unit costs (R/ton). Such a reduction in waste mining costs is achieved primarily by reducing transport costs – since with depth, drill, blast and excavating costs do not change significantly, it is only the transport component that increases. One way to control or even reduce the rate of cost increase with depth is to use continuous mining systems, instead of the cyclic based blasting, loading and truck based systems.

Figure 6.1 shows the starting point of the continuous system is based on free-digging, where no drill and blast is required. In a later section, the possibility of combining cyclic with continuous transport systems will be discussed.

One method to remove large volumes of soft overburden cheaply is through the use of a continuous excavator, the Bucket Wheel Excavator (BWE). Depending on the design used, it can excavate and also transport material over a limited distance. Single bucket shovels (rope, hydraulic and FEL) and draglines are cyclic (discontinuous) excavators, since the swing movements interrupt the excavation process and material generation is discrete. The bucket wheel excavator excavates continuously by means of small buckets that are mounted on a rotating wheel. When the wheel swings over the excavating face, the rotating buckets have an upward excavating action and are filled with material (individual bucket digging profile and operation is similar to a dipper on a rope shovel – the only difference being that there are many buckets on the wheel). As the wheel rotates, the material falls out of the buckets onto an internal conveyor belt which transports the material out of the machine to an over-the-pit or round-the-pit conveyor belt system. The excavating wheel can be lifted or lowered, to make subsequent bow shape cuts to shape the face as it is desired in terms of stability or blending considerations. Figure 6.30 illustrates the bucket wheel of a large C-frame machine.

The conveying component of the machine on which the excavated material is dumped can operate in different modes as shown in Figure 6.31, namely:

- Dumping directly on waste dump – for narrow pit and strip-mines only.
- Dumping on an intermediate conveyor to the main round-the-pit or over-the-pit conveyor, then to the waste dump. This method is mostly used at strip or terrace mining.
operation, together with a stacker or reclaimer for dumping the excavated material.

Figure 6.30  Typical bucket wheel arrangement on ‘giant’ BWE

The second option is most widely used, but it also requires the largest capital output and an extensive fleet of support equipment. If a system comprises more and more components, then the availability of the whole system is negatively effected, since all these components need to be working before the machine can actually produce. Fortunately, conveyors have a high availability and thus the system availability as a whole often exceeds 60-70%.

Dump direct to trucks is not recommended, because a cyclic method (trucks) are not easily reconcilable with a continuous mining system, unless a specialised machine with on-board storage is purchased, or a silo or intermediate storage system used to accommodate the change between continuous supply and discrete transport.

The BWE works in block operations at the face, with parallel advance of the machine in a direction parallel to the face conveyor belt. The face is excavated in a couple of blocks from an initial sump cut, by swinging and pushing the wheel out over the length of the working face. The BWE is then moved forward to make the next sump cut and the transfer point on the face conveyor belt moves with it. After a couple of forward movements the whole face conveyor belt must be moved forward. The number of movements of the transfer point and
face conveyor belt is the function of the machines outreach. The bigger the outreach; the further the machine can cut before the conveyor belt must be moved. Thus the outreach is an important factor in the selection of the type of machine, since face conveyor moves are time consuming.

![Compact BWE dumping directly](image)

![Compact BWE dumping to intermediate conveyor](image)

Figure 6.31 Discharge options for compact BWEs using cross-pit or round-the-pit conveyors

Cuts are made on both sides of the face, so that the BWE can turn around and come back (thereby reducing deadheading time considerably). Planning of BWE cuts and the associated conveyor belt maneuvers is a complex and specialised process.

### 6.6.1 Selection of BWE

There are two main factors which initially establishes the feasibility of the system in a mine or particular area of a mine:

1. The production volume, stripping ratio and duration of the surface mine must justify the high capital costs for the equipment. That is to say a long lifetime and/or high
dividends to pay the large CAPEX off over a longer payback period.

2. The material which the machine will excavate must be soft enough to be mechanically cut (free digging) without any blasting (otherwise the system becomes basically cyclic).

The type of material mostly suited for BWE loading is:

- Sand, gravel and medium strength clay’s with a compressive strength of less than 10 – 15 MPa
- In special circumstances with specially designed machines, hard rock formations can be excavated (up to 30 MPa).

The following machine applications and limitations must be recognized when determining if a BWE will be feasible on a mine:

- Avoid materials that contain single isolated rock blocks or boulders in a soft matrix.
- Cohesive material must not be excavated because it will block the buckets – especially wet clay with high cohesive values.
- The excavation face must be very stable and self-supporting – more so if large excavators are used over a long cut distance since the required stand-up time is long. Flatter slopes can be mined, but this effectively reduces the useful outreach of the machine.
- Overburden is not so problematic since benches can be cut to suit the machine – only areal extent may increase the stripping ratio – as would shallower bench angles and wider berms.
- In the pit conditions must be dry or well drained and the bearing capacity of the rock under the machine strong (high bearing capacity for heavy machines).
- As the machine size increases (up to 20 000 BCM/hr), the degree of application increases with a specific mine location. It is important to take the application limitations of the machines into account and the expected performance over the full duration of the mine’s life.

In South Africa, the geological setting of the coal reserves does not typically include material soft enough to be excavated by BWE’s. In isolated cases, alluvial deposits of river sand, etc. may be found as a softs covering the overburden, but these would have to be of
sufficient depth and extent to warrant the purchase of this type of equipment.

Bucket wheel excavators are available in a wide range of sizes and capacities. Bucket sizes varies from 60–6300 litres. The bigger machines (Giant BWEs) have a theoretical daily production of over 240 000BCM. The weight of such a machine is approximately 12 800t, hence the need for a strong strata under the machine. In South African surface strip coal mining context, only the smaller hydraulic machines could be considered, as shown in Figure 6.32. These machines are a good choice where it is proven that a BWE will work, but mining rate or life of mine do not justify a large unit. In addition, the machine itself is “harder” and as such will accommodate digging harder materials – even up to 50MPa – due to the additional rigidity of these smaller units. The main advantages are;

- Shorter delivering time
- Lower weight
- Cheaper – for the same capacity as the smallest C-frame machines
- Can handle harder material (because of shorter but stronger arm), but gives lower hourly production than the comparable C-frame.

Figure 6.32 Typical compact BWE units
The compact unit is designed to provide for the requirements of strip mines with soft overburden. The beam length is in general 10 – 30 meters with a maximum delivery of approximately 5000 m$^3$/hr for the biggest (C-6300) series. The main design points of the compact units are:

- Crawler mounted – low bearing pressure suitable for weaker floors
- Low-placed counter weight – low centre point of gravity and good machine stability on non-level floors
- Hydraulics are used right through, especially for the lift and positioning of the cutting wheel
- Materials are directly transferred to the face conveyor (certain models is also equipped to handle trucks together with an on-board storage system).

### 6.6.2 BWE Productivity Estimation

It is easy to calculate a theoretical production figure for BWEs, taking into account the volume of the buckets and the wheel revolutions per minute. Operation application, excavator cycle management and planning and availability of machines and transport system reduce the productivity greatly and various analyses have found that the most accurate production figures are found by combinations of the theoretical calculations with empirical system availability values.

The main factors that influence productivity of a BWE are:

- The type of material that is excavated (digging resistance, $KA_{corr}$, N/cm$^2$)
- Face dimensions – small face height or length reduces actual digging time and increases repositioning time
- Excavation cycle management (ancillary operations – belt moves, etc.)
- Material handling system on BWE;
- Climatic conditions (amount of rain, etc.)
- The size of the machine.
6.6.3 Continuous Miners

Continuous mining machines include two basic types of machines, classified according to their cutting system;

- **Face attack**: The cutting head is mounted on a boom which approaches the vertical face of the material to be excavated. Machine sits on floor of pit and excavates coal face to floor depth. Ideal where full seam width is mined and seam height greater than 4m (thinner seams would require a faster rate of advance which is problematic for the machine and transport system).

- **Surface attack**: The cutting mechanism is similar, but mounted underneath the machine and can handle thin partings or coal seams. The cutting action involves the large cutter head impacting downwards on to the seam, and drawing the coal towards the rear of the machine.

Both types of machines would only be considered for coal seam mining applications, with the preference in strip mining being for the surface attack configuration, these being best suited to relatively soft parting or coal, preferably with pronounced banding. They have high cutting capacities, typically about 2500t/op.hour and cut a path up to 4m in width, 1-600mm deep through solid coal.

Units in use are diesel powered, and are sufficiently mobile to be working in several pit areas for parts of a shift. The miner units drive on the coal/partings being cut. Although a continuous machine, as will be seen in the next section, the strip mining method is not well suited to the use of a continuous transport system, so these units are normally coupled with a discontinuous transport system. Typical loading time for a 240t BDT is around 6 minutes, but the machine must stop cutting whilst the trucks changeout and the discharge boom of the machine is short, requiring the truck to have access to the top or side of the seam as it is mined.

For adequate maneuvering room and high productivity, cutting runs should be 150-300 metres (before the machine has to turn around and cut back). Pit widths of 40 metres minimum recommended to allow ease of turning and the working area should be well drained, but ground bearing pressures are not a problem.
6.7 Transport Equipment – Continuous

A continuous transport system is rarely used for coal (as ROM) or primary transport since, to maximise system efficiency, it would need to be fed by a continuous excavator. Alternatively, both systems can be combined by using a cyclic loader together with in-pit crusher, from which the conveyor is fed. However, due to the rate of advance of the coal mining faces in strip mines and the associated repositioning of the belt, these systems are seldom considered. In terrace mining however, conditions more favourable to the use of cyclic-continuous combined systems may arise and it is useful to review the motivation and selection of these systems.

Truck haulage maintains an inherent flexibility and improvements continue to be made in performance and capacity. Conveyor systems may be cheaper to run than trucks – but this only really becomes apparent with increasing load elevation, as shown in Figure 6.33, an effect magnified by the much steeper grades (hence shorter grade distances) that can be used with conveyor installations.

![Figure 6.33](image)

Figure 6.33 Comparison between relative costs of diesel-electric trucks and conveyors.

A further disadvantage associated with the use of conveyors is the necessity to move the installation as the mining face moves. In strip mining, when loading coal, the rate of advance of the face is rapid and the conveyor would need frequent repositioning. Conversely, for overburden transport, cross-pit or around the pit systems are used, only the face feed conveyor need be regularly repositioned.
Trucks have a poor energy efficiency. Only 40% of the energy used is for the transport of the payload (rock), the remaining energy is consumed in moving of the truck itself. Conveyor systems have a higher energy efficiency: They use approximately 80% of the energy consumption for the transport of the payload. Energy costs for conveyors are 30% of the energy cost for trucks on level route and as little as 15% for increased payloads (this applies particularly where electricity costs/tariffs are low in comparison with diesel fuel). Recent international political events have tended to regenerate concerns over future oil supply, price stability, carbon combustion (Kyoto Protocol) and engine emissions in general. For such reasons, conveyor based transport is becoming increasingly attractive. Truck haulage improvements continue to be made in performance and capacity. Doubts must exist, however, over the extent to which further economies of scale may be made. Conversely, belt conveyor systems do not suffer this constraint and exhibit some of the general advantages listed in Table 6.14.

Table 6.14 Advantages of continuous transport systems over cyclic systems

<table>
<thead>
<tr>
<th>Advantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>Electrical power is used and the specific energy consumption is at least half that required for truck haulage</td>
</tr>
<tr>
<td>Electrical power demand is uniform</td>
</tr>
<tr>
<td>Conveyor installations are less sensitive to cost inflation</td>
</tr>
<tr>
<td>Operating costs and skilled labour requirements are lower</td>
</tr>
<tr>
<td>Equipment has superior longevity and is “inflation proof”</td>
</tr>
<tr>
<td>They are less prone to breakdown and bad weather delays</td>
</tr>
<tr>
<td>In long haul, deep, high capacity operations conveyors can save substantially on capital cost (fleet replacement and fleet growth with depth of working and age of equipment)</td>
</tr>
<tr>
<td>They can negotiate poor terrain and require less road width than a truck-based transport system</td>
</tr>
<tr>
<td>Environmental impact is improved with reduced noise level, dust, carbon combustion products and atmospheric pollution.</td>
</tr>
</tbody>
</table>

Conveyors are most cost effective when moving large volumes of material at regular flow rates between two fixed points, operating
costs rise greatly the more transfer points and other complexities are built into the system. Cost efficiency reduces with;

- Irregularity of flow rates. The high capital cost of conveyor based systems means that they must be fully utilised, or their cost competitiveness can be quickly eroded.

- Low volumes. In many mining applications, the size of conveyor is fixed by the size distribution of material to be conveyed. The capital cost of a low volume installation may be no less than a high volume installation.

- End-points not fixed. Whenever feed or discharge points on conveyor-based systems must be moveable, substantial additional capital cost inputs are required. Extra complexity also requires more supervision, and higher operating costs.

Although conveyor systems are less flexible in operation than truck haulage and although a breakdown in any part of the system can immobilize the total system, mechanical availabilities of >80% are constantly reported by large operators. Modern technology and high quality components as in pulley’s, bearings and support structures and conveyor belts, make conveyor belts a lot more reliable with minimal operating maintenance. Single conveyor belts have availabilities of >95%.

6.7.1 Types of Conveyor Systems

A conveyor is a moving continuous rubber belt supported by troughing idlers (laden upper belt) and return rollers (lower, empty belt) suspended on frame which can be either fixed, or relocatable or track shiftable. A drive unit is used to transfer energy from an electric motor via a gear box to a large pulley which in turn pulls the conveyor belt. The size and complexity of the drive station depends on the length, elevation and power requirements for the particular system. A tail pulley is normally on the opposite end of the conveyor to the drive station.

Various types of conveying systems exist, the design being primarily a function of the required mobility (permanence) of the system, extensibility and the desired angle of conveying. Some designs of conveyor will allow for almost vertical conveying, but generally these systems are limited to low capacity, well prepared fine crushed materials, as shown in Figure 6.34. Table 6.15 summarises the types of conveyor systems.
Table 6.15 Characteristics of types of conveyor belt systems

<table>
<thead>
<tr>
<th>Type</th>
<th>High angle capability?</th>
<th>Extensible?</th>
<th>Characteristics</th>
</tr>
</thead>
<tbody>
<tr>
<td>Conventional pulley driven</td>
<td>No</td>
<td>No</td>
<td>Pre-fabricated structural frame units between head and tail sections. Semi-permanent intermediate length (&lt;3km) and capacity.</td>
</tr>
<tr>
<td>Conventional pulley driven</td>
<td>No</td>
<td>Yes</td>
<td>As conventional, but the conveyor is designed to be extended in a linear direction. Modern systems allow this to be done quickly by using “take up” belt or belt storage system in the drive head.</td>
</tr>
<tr>
<td>extentable</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cable driven or supported</td>
<td>No</td>
<td>No</td>
<td>Designed for long distances (10s km) and use steel cables to support or drive the load and to take the tension, with the rubber belt simply holding the material. Permanent high speed high capacity system</td>
</tr>
<tr>
<td>belt</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>High angle conveyors</td>
<td>Yes</td>
<td>No</td>
<td>Several designs available, including pipe configurations (the flat belt is rolled into a tube by a series of idlers which contains the load and allows the material to be taken up much steeper grades, sandwich belts - the material is sandwiched between two conveyor belts whilst going up incline; and pocket conveyors - a specific belt type is used to avoid the material falling backwards. High angle conveyors are high capital cost items, require finer feed sizing and are low capacity at present.</td>
</tr>
<tr>
<td>Transportable (piggyback)</td>
<td>No</td>
<td>No</td>
<td>A modular system containing drive and tail and intermediate support structure. When the conveyor advances, another unit is inserted into the conveyor stream. They are very flexible, but because of the number of transfer points they have a greater probability of breakdown. Variable capacity designs for short term variable applications only, up to 200m length per unit</td>
</tr>
<tr>
<td>conveyors</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Track shiftable belts  No  No  Conventional units with sturdier frame which are able to be moved in a lateral direction by use of a special pulley attached to the side of the conveyor system. These conveyors have wide applications especially for face and dumping systems. Bridge conveyors have a track mechanism on each end and therefore are highly mobile.

Bandwagon belts  No  Yes These are conventional belt systems, which are track mounted and usually act as an intermediate transport system between the crushing/excavating system and the main face conveyor system. They can be fixed or slewing and can vary in complexity. Their astute use in mining applications gives the conveyor system far more flexibility than would otherwise be possible.

Figure 6.34  Conveyor systems classified according to type, capacity and range of operating angles
The design of any of the above systems is specialised and normally left to the manufacturer, based on data supplied by the mine. Major cost-related design information includes:

- Type of belt design chosen
- Capacity required (tons/hour)
- Belt width, speed and type of belting (related also to capacity)
- Load elevation and transport distance
- Material characteristics including unit weight, swell and lump size variation (related also to capacity)

6.7.3 Crushing

**SIZING REQUIREMENTS**
Conveyor based systems offer a means to create continuous mining systems with potential to innovate in the basic design of mining methods. The physical and mechanical properties of the materials is the principal factor in determining whether overburden and coal can be removed by continuous mining methods. The absence of a suitable continuous miner to remove medium to hard coal will mean that, in the short term, conveyor systems will be required to work in situations where single bucket loaders are used.

Belt conveyor systems require smaller lump sizes of material than trucks. In most cases crushing is necessary prior to loading onto conveyors, particularly when using large shovels. Two main options are available to combine loading and transport in conjunction with semi-mobile crushers;

- The TCC systems (truck-crusher-conveyor).
- The LCC systems (loader-crusher-conveyor).

In the TCC system trucks are used for level transport only on the individual benches where they are loaded by shovels. In this way the adaptability of trucks is best utilised (specifically when the shovels change from bench to bench). Restricting trucks to level transport has the advantage of considerably reducing diesel consumption (because trucks are no longer employed for uphill transport, when consumption is much higher). In the LCC system, for longer term shovel locations, it is possible to use a loader feeding directly to a crusher, which follows the loader along the mining face. Conveyors are applied where the material is moved over the same route for a
longer period of time, usually on a specially constructed conveyor road connecting the individual benches with the surface and along the onward route to the tipping point. The primary ore crusher is removed from the surface down into the pit in front of the conveyors and is designed to be a mobile or semi-mobile unit. As such, it can be relocated from time to time as required by the mining advance and system (TCC or LCC).

The criteria used in applying in-pit crushing techniques require careful analysis. Product lump size, in particular, is a decisive constraint on belt conveyor design, as well as the crushing ratio of the primary crushing unit. Trucks are able to handle nearly any size of lump, subject to the dipper size of the loader. The maximum lump size that can be transported on conveyors depends on belt width (W) speed, troughing angle and conveyor inclination. Figure 6.35 summarises the lump size limits for the various belts types previously introduced. The design of a conveyor will also be dependent on how frequently large lumps occur. As a rule of thumb, the maximum individual lump in a material of standard size distribution should be no greater than 0.4W when W>1000mm and 0.25-0.3W when W>700mm. The frequency of large lumps is an important conveyor design factor and because fluctuations are usually beyond the control of the conveyor operator, a cutoff limit of about 250mm to 350mm maximum lump size for belts of 1000 mm width and greater is usually adhered to.

![Figure 6.35 Lump size limitations in belt conveying](image)

Figure 6.35    Lump size limitations in belt conveying
Various methods exist to prepare material for belt conveying, each reducing the maximum lump size to some predetermined limit. They are as summarised follows:

1. Separation of oversize lumps by screening and feeding only the undersize to the conveyor. Oversize lumps are handled separately, either by truck, or they are recycled after secondary blasting. This method applies for very small percentages of oversizes and where crushing costs are higher than separate handling costs.

2. Separation of oversize lumps by screening and feeding the undersize directly onto the conveyor and feeding the oversize to a crusher for sizing. This method applies for high percentages of oversize and where the total flowrate cannot be passed through the appropriate crusher type.

3. Total material flow is passed through a crusher prior to feeding onto a belt conveyor. This method, which is commonly used, applies if the total material flowrate is within the capacity of an appropriate crusher and where the overall costs are less than for a combination of screening and crushing.

MOBILITY AND LOCATION OF IN-PIT-CRUSHERS

Since the mining faces are dynamic, and the feed system (TCC or LCC) options dependant on the mining method, some consideration needs to be given to both the mobility of the crushing system in the pit and its location. Semi-mobile plant locations apply when there are a multitude of excavating faces which are not each worked continuously, and when the excavators need frequent relocation to different levels.

The following types of crusher propelling devices are available;

- Pneumatic tyres. A crusher plant mounted on pneumatic tyres offers high manoeuvrability and is suited to an application where relocation is frequent. The ground must be stable and the loads on the carriage must be moderate during relocation. During crushing operations the pneumatic tyred carriage must be removed or the wheels must be retracted.

- Walking mechanism. This is suitable for very high loads but it is a system with a very slow propelling speed. Ground conditions must be stable. The system would not be suited to applications which require frequent relocation. The walking mechanism is retracted or removed during crushing operations.
Crawlers. They provide unlimited universal mobility without any of the above-mentioned restrictions. Gradients of up to 10% can be negotiated by fully mobile units which are permanently mounted on crawlers. The same is true for semi-mobile units where a separate transport crawler transports components of up to 1000 tons to the new crusher site.

 Crushers with a reduction ratio of 1:10 - so called primary crushers – are used almost exclusively for in-pit crushing. The choice of a crusher is dependent on the following factors:

- Hardness of material to be crushed
- Size and shape of rock to be crushed
- Crusher mass to be the minimum for maximum throughput
- Economic considerations.

With the trend toward higher throughput, the mass of the crusher becomes more important, as it constitutes a large part of the crushing system. The gyratory crusher has a much higher throughput for the same, or lower mass, than other types of crushers. Direct dumping is another advantage of the gyratory crusher. No feeding equipment is needed and this achieves a mass and monetary saving. Maintenance is infrequent and operating costs are also low. In general terms, the following basic characteristics are required for a primary movable crushing and conveyor system:

- A large capacity (3 000 to 4 000 tph)
- High system reliability (more than 85%)
- Compatibility with trucks
- Large crusher feed openings (bigger than 1,3m)
- Hard-rock crushing ability
- Freedom from clogging and bridging
- Relocatable within a minimum of time (time depends on mobility of option chosen – hours or weeks)
- Transportation of components up a 10% incline.
For any one of the crusher feed methods mentioned (TCC or LCC) the correct location is essential. The choice exists between two principal locations as follows;

- The digging face (LCC). In this case the plant is fully mobile and fed directly by an excavator (rope shovel, hydraulic loader or front-end loader). The plant discharges directly, or, in most cases, via a belt-wagon onto a shiftable face conveyor. This location is good only if one or more excavating faces exist, face height is high and lateral advance rates not excessive and the excavators stay permanently at the same face. For this location, only the preparation methods which include crushing, (2) and (3) as listed above, are appropriate.

- One strategic point in the mine (TCC). The plant is semi-mobile and follows the pit advance at appropriate time intervals. It is fed by trucks which haul material from the face to the crusher. The haulage route is kept as short as practicable and, if possible, any uphill haulage is avoided. The semi-mobile crushing plant discharges onto permanently installed conveyors which are only relocated or lengthened in conjunction with the plant.

Figures 6.36 and 6.37 depict these options in a terrace mining method.

![Figure 6.36 TCC and LCC combined cyclic loading, transport and continuous transport systems](image-url)
Further consideration should be given to site preparation as well as to haul road construction. Fully mobile plants, particularly on crawlers, need little or no site preparation at all, as they can generally move around under the same traction conditions as the loading excavator. Semi-mobile plants need some site preparation, mainly excavation, and sometimes some civil works, particularly for bigger units. The frequency of relocation and site preparation works have to be considered in cost optimisation. Site preparation for shiftable conveyors is minimal and is usually a temporary measure as preparation for the shifting operation itself. Site preparation for stationary conveyors is required to a certain extent, but it is minimal compared with the construction of truck haul roads.

6.7.4 Waste Handling

An in-pit crusher-belt conveyor system may be used to transport overburden or coal from the mine. In coal transport, although the degree of application of these systems is generally poor for both strip and terrace mining, this can be done by belt conveyor at lower cost than by trucks. Dumping will take place directly into the plant or plant stockpile.
For overburden conveying, the necessary preparation and stacking (dumping) costs are considerably more and are offset by savings in transport costs. Only large quantities of overburden to be transported over fairly long distances at considerable elevation would justify such crusher-conveyor-dumping systems.

In the case of waste, the dumping system usually requires a spreader and or tripper unit, as outlined in Figure 6.1. The tripper unit allows the flow of material to be separated from the feed belt (overland semi-permanent installation feeding the shiftable dump belt) and transferred onto the receiving belt of the spreader. The tripper moves as the spreader moves and is usually crawler mounted.

A spreader is usually a crawler mounted structure with a receiving and discharge boom. The receiving boom brings the flow of material from the shiftable dump conveyor onto the discharge boom. The tripper and discharge boom are interconnected in such a way so as to allow coordinated movement to occur. The discharge boom simply discharges the material onto the dump. When required the discharge boom can slew to ensure the dump is built up in an orderly fashion.

If instability is a major problem, a long boom spreader is required to ensure that the spreader is a sufficient distance back from the potential slip/failure surface. Spreaders can also operate in high and low dumping modes, enabling upper and lower dump levels to be constructed which minimises the number of conveyors shifts and moves required.

### 6.7.5 Continuous System Costing

The substantial capital outlay required for an in-pit crusher and conveyor system, compared with total truck haulage system, necessitates a thorough cash-flow analysis that includes a life cycle costing approach, operating and maintenance costs assessment and an economic evaluation during the life of the mine. There are several alternatives to consider when choosing a materials handling system and as every mine is unique, a separate cash-flow analysis should be done for each new situation:

- RDT/BDTs hauling material out of the pit to the end point (fixed crushing station or waste dump).
- Similar to above, but the trucks are fitted with a pantograph system for upgrade haulage.
- RDTs feed an in-pit semi-mobile crusher with conveyor transport out of the pit (TCC).
A shovel and mobile crusher advance together along the mining face. Transportation is done by conveyors (LCC).

Results of a typical feasibility study for a 3000t/hr mobile in the pit crusher at a terrace mining operation gave the following results:

1. RDT/BDTs hauling material out of the pit to the waste dump – cost basis 100%

2. Truck feeds a fixed crusher outside the pit and a conveyor belt system runs from the crusher to the waste dump 75%

3. Truck feeds a mobile crusher outside the pit and a conveyor belt system runs from the crusher to the waste dump 75%

4. Truck feeds a mobile crusher in the pit and a conveyor belt system runs from in-pit to the waste dump 66%

5. Loader loads a mobile crusher in the pit and a conveyor belt system runs from the crusher to the waste dump 42%

These numbers reflect specific financial and mining-method related factors that would make the actual waste mining cost savings difficult to translate to other operations, especially the associated decisions regarding purchase of new fleet equipment or the supplementation of the current fleet with the new system. Figure 6.38 illustrates a typical comparison between a TCC and conventional surface crusher and truck haulage system, for one particular application.

Since load elevation is the principle economic driver for these types of systems (especially where no productivity improvements are possible due to the necessity to use cyclic loading systems), it is necessary to also investigate the phasing of the equipment purchase over time.

- Early in the life of a mine, where depth is not an issue, a truck and shovel system will be the most economic option.

- As depth increases, costs can be reduced by additional capital expenditure on trolley-assist, in which electric power is fed directly into the wheel motors of diesel electric trucks on the laden haul out of the pit.
With further increase in depth, capital expenditure is required to increase the hauling capacity of the truck fleet to maintain the same production tempo, expenditure will also be required to replace ageing units also. At this point in time, a continuous transport system may have the best possibility of direct cost competitiveness. Figure 6.39 illustrates a typical expenditure comparison between these systems, based on an increasing depth of pit over time.

It is important to note the role of depth (and an associated increase in stripping ratio) does not often apply to South African coal reserves, which are in the main horizontally stratified and limited in depth. Most benefit under these conditions would probably be derived from waste handling in terrace operations.
Figure 6.39  Typical capital cost profiles of the cyclic and continuous systems over time
THE SELECTION AND APPLICATION OF DRAGLINES

Learning outcomes

Knowledge and understanding of

- Application limitations for draglines in surface strip coal mining
- Advantages and disadvantages of use
- Basic machine operation and operational parameters
- Bucket dumping options and effect on productivity
- Pit geometry influence on machine selection
- Pit geotechnical influence on machine selection
- Highwall and spoil stability issues
- Factors influencing general dragline-pit dimensional relationships
- Bucket size determination and factors to consider
- Mine planning considerations for the use of draglines
- Dragline selection methodology – life-of-mine approach
- Top and side cut box-cuts
- Simple side casting
- Advance bench chop down
- Extended bench working
- Managing and optimising dragline productivity
- Measures of dragline productivity
- Approach and issues surrounding dragline relocation

Apply, calculate or predict

- Situational analysis of possible machine applications
- Pit dimensions to machine selection and sizing – reach factor
- Rehandle, TCMs and prime CMs
- Bucket capacity determination for a specific production tempo
- Most suitable method from specifications of pit and machine
- Effect of overburden depth on prime productivity and incremental costing of increase in cover

Evaluate or design

- Apply situational analysis in the selection of dragline for single seam coal mining
- Carry out an analysis of the merits of dragline applications under a given set of mining specifications
A selection methodology for a life-of-mine application of a dragline
Dragline productivity from key performance indicators
7.1 Introduction to Draglines

The characteristics and selection parameters of draglines will be discussed, following which the analyses of the development and operation of a surface strip coal mine using draglines is given, initially using simple side casting, following which more advanced techniques are introduced. Finally, those factors which impact on dragline performance are discussed.

The dragline is a cyclic excavator and transport system combined; no intermediate transport is employed when using a dragline, it excavates and dumps directly to the waste or spoil side of the strip mine. It is critically important to note that draglines of the type and size discussed in this Module are ONLY USED FOR STRIP MINING OVERBURDEN REMOVAL and they have no application in any of the other type of mining material handling systems.

Draglines are used where the material that is excavated, must be transported over a short distance only (maximum approximately 100m). The machine is thus ideal for strip mining applications where the width of strip (exposed coal) rarely exceeds 50-60m. In general the larger draglines with bucket capacities of up to 68m$^3$ are used in the mining industry in South Africa. They are electric driven, mounted on a rotating base (tub), the base itself on walking shoes on both sides of the tub. The machine excavates by dragging the bucket up the mining face to fill it before it gets hoisted, swung and dumped on the spoil-side of the mine. The machine digs primarily below its working position (pad). Figure 7.1 illustrates the digging profile of the machine and the Figure 7.2 the available digging forces.

The deposit geology exercises a primary role in determining if a dragline can be applied cost-efficiently for overburden removal. Basic requirements are;

- Relatively simple seam geometry. Because of their relative immobility, draglines are most cost effective excavating thick horizons of overburden. Large numbers of thin partings requiring many dragline passes will cause large losses in machine productivity. However, draglines are rarely cost effective where the overburden thickness exceeds 50-60 metres. Beyond this depth, geometrical considerations increasingly require more waste to be rehandled, effectively increasing the volume of material moved to expose the coal seam below.

- Sufficient pit working room. Scheduling of dragline operations for multiple sequential tasks requires adequate pit length - elapsed time between sequential tasks. If insufficient pit length
is available, the relative immobility of the dragline means that it may not be able to be deployed elsewhere whilst waiting for in- or ex-pit activities - causing low utilisation and loss of cost efficiency.

Figure 7.1 Dragline excavating profile

Figure 7.2 Dragline digging forces
Coal quality. Draglines must move waste horizons sequentially. Therefore, coal from an upper seam must be removed before coal from the lower seam is exposed. This makes blending out of the pit very difficult and would require extensive blend stockpiles to be created if blending were a product requirement.

Geotechnical characteristics of the deposit. Draglines are not very sensitive to geotechnical factors, but some consideration needs to be given to key geotechnical issues before a decision is made to utilise the dragline. Usually, a geotechnical investigation would be commissioned before a dragline purchase is considered – it is a high capital cost item and if such a machine is deployed on the mine, its success (as the only piece of bulk waste excavating equipment) must be ensured by adequate planning and investigation.

The main advantages and disadvantages associated with the use of a dragline (where the geology and mine layout allow the use of the machine) are given in Table 7.1.

The dragline is the largest excavating machine in South Africa's strip coal mines. The machine is used to remove the overburden to expose the coal seams underneath. It is expensive but a highly productive machine, the expected production tempo is 35 BCM/m³ bucket capacity/hr (approximately 1MBCM/month) at a cost that varies according to the mining method, typically R1,50-R2,50/BCM. The bucket size can be as big as 80m³ (but not currently in South Africa) and the machine's power consumption is 1MW and higher. It can excavate material up to 50-60m lower than it’s operating level and can dump the material up to 30m above it’s operating level over a reach of up to 100m as shown in Figure 7.1.

Since draglines are the single most expensive piece of capital equipment which a mine will buy and as in the case of the South African strip coal mines, the only prime overburden excavating machine, they must be fully utilized. At most of the mines the machines work 24 hours a day and 363 day’s per year and their availability is often in excess of 95%. Several new mines however have found that, due to the CAPEX cost of the machines, the long order lead, construction and commission time and their relative inflexibility in mining, the newest series of hydraulic or rope shovels, coupled with trucks of >280t capacity, can compete with draglines for overburden handling. Today, it is not a foregone conclusion that when establishing a strip mine, a dragline is required.
Table 7.1 Main advantages and disadvantages of draglines in strip coal mining application

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>High productivity (BCM/hr) and low operating cost (R/t) – typically R1,50-R2,50 per BCM prime</td>
<td>The use of the machine requires bench and pad preparation which entails the following:</td>
</tr>
<tr>
<td>Dumping over a large distance from the excavation face negates the needs for intermediate transport.</td>
<td></td>
</tr>
</tbody>
</table>
  - Leveling of highwall block after blasting |
| Material with large rock blocks can be easily handled (lower PDF thus lower explosive costs) |  
  - Maintenance of sufficient height for dumping |
| Flexible operation – various applications are possible – but limited to strip mining method |  
  - Distribution of fine aggregate on cohesive materials on pad |
| Safe against flooding and failures of muck piles since the machine is sitting on the high wall and not in the pit |  
  - Profiling of highwall since the machine cannot walk on gradients exceeding 5-6%. |
| High availability (because of the simplicity of the machine and its being electrical driven) |  
  - Blasting must be enough to break strata, but must not throw the material – the maintenance of dump height is important (cast blasting usually not applicable except in areas of deeper overburden) |
|                                                                          |  
  - Can’t be used on an unstable high wall – especially where there is wet material together with water |
|                                                                          |  
  - Less productive than a rope shovel loader in terms of BCM/m³ bucket capacity/hr |
|                                                                          |  
  - Very expensive to purchase, thus large reserves and long duration of mining activities required to repay capital |
|                                                                          |  
  - Becomes unproductive if overburden depth exceeds hoist drum capacity (typical 50 – 60m). |
7.1.1 Basic Machine Operation

The machine walks on two shoes either side of the machine tub. An individual step is usually a couple of meters in length, each step takes 40-50s and the walking speed is not more than 1kph. This results in a lot of the machine’s available time being spent on walking and re-positioning. The use (utility) of the machine (time spent on the actual excavation) is thus lower, typical 50 – 70%, the lesser value being typical of applications with thinner overburden, where the excavation face advances rapidly.

The machine functions in various modes depending on the pit geometry. The main aim of the operating method is to move most overburden only once (prime handling). Although it may be necessary that the machine excavates and handles the material twice, due to reach or digging depth limitations. This is known as re-handling and it is obviously unproductive since it effectively multiplies the total overburden handled and increases the stripping ratio. Mine planning and dragline planning aim to reduce re-handling and it is thus a highly specialised and complex aspect of strip-mining management.

Draglines are usually the machine on a strip mine that pace all the other operations. In the selection of the dragline and the dragline mining method, the rate at which the dragline exposes coal surface area is defined by the required production rate of the coal. The basic operating technique is a side casting sequence, which includes:

- Swing (45° - 120°) from dump to excavating point while the bucket lowers to the bottom of the highwall (larger swing angles are unproductive)
- Loading of bucket, when the bucket is dragged in and up towards the machine from the bottom of the highwall. Ideally, the bucket should be full after dragging for a distance equivalent to three bucket lengths.
- Swing, hoist bucket and dump on waste side
- Swing back, etc.

The bucket is free to swing on the hoist apparatus and the dragline cannot “force” the bucket to dig in certain place. Therefore, a lateral limit is necessary if a high wall is to be excavated and shaped. The key cut is used for this purpose since, being the first cut into the overburden, the blasted rock helps to control the bucket position as it cuts a path against the new highwall. During normal functioning the machine is periodically re-positioned to optimize the bucket control and the face shaping.
Although the dragline is normally used for excavation work under its own level, chop down is used now and again where the dragline prepares the next block pad by digging and dragging of the material above it’s own level and by swinging through 180° to dump. It is however a time-consuming and unproductive method of working and if available dragline time is limited, it is often better and more cost-effective to use a truck and shovel pre-strip operation.

Blasting must be carried out to almost the full depth of the overburden to ensure its diggability – but must not damage the coal seam below. Blasting the overburden leads to an increase in the volume of material because of the swell factor, therefore it is often necessary that chop down is used (if buffer blasting especially), so that the machine can sit at the right height above the coal layer, especially where dump height is limited.

7.1.2 Machine Operational Parameters

For a general understanding of a dragline working system, the design of the machine should be understood. The most important characteristic which distinguishes one dragline from another dragline is the boom. Characteristics which are related to boom configuration are:

**BOOM LENGTH**
Measured from the boom foot to the axial line of the boom point sheave wheel. Typical length (large draglines) - ± 100 meters.

**BOOM ANGLE**
Typical 38º (between limits of 30º to 40º). The boom angle is in general selected to approximate the angle of repose of the dump spoils. Steeper angles allow a larger dump height, but it also reduces reach of the machine. Flatter angles (30º) allow longer reach distances, but reduces the dump height. Take note that the boom angle on the machine can’t be changed as is the case with other cyclic excavator machines. The boom is constructed from of tubular steel and designed with a low safety factor (the relationship between maximum and applied load) of between 1,15 and 1,25 and is filled with gas. If cracks develop, the gas leaks out and the alarms systems shows where the crack is located in the boom. Thus the bucket size is also a function of the chosen boom angle – steeper angles may accommodate larger buckets.

**OPERATING RADIUS**
It is the distance from the axial line of the dragline tub (pivot center line) to the point directly under the boom point (position where the hoist rope is vertical). For a typical dragline with a 99 meter boom at
38º, the operating radius will be approximately 87 meters. The reach depends on how near the dragline sits to the edge of the highwall (referred to as positioning factor) and is thus the distance from boom point to edge of highwall.

**MAXIMUM SUSPENDED LOAD**

It is the weight of the bucket (frame included) and the material in the bucket which the boom was designed to carry. A general rule is that one cubic meter (of nominal bucket capacity) is equal to 3000kg maximum suspended load, thus a 70m³ bucket represents a MSL (Maximum Suspended Load) of 210t.

Take note that different from a crane, the dragline is so designed that it can carry the full specification MSL directly under the boom point.

The dragline size is mainly determined through the maximum suspended load. A certain size of dragline can carry a bigger bucket if a shorter boom is chosen. A general rule is to choose the longest boom for the required bucket capacity, because it makes it possible to use the efficient normal side-casting working method with larger excavating depths.

**HOIST ROPE AND EXCAVATION DEPTH**

The hoist drum (inside the dragline) accommodates only a certain amount of rope between the full out (maximum digging depth) and the full in position (maximum dump height position). The digging depth, which is quoted in manufacturers specification sheets, refers to the amount of rope which is accommodated on the hoist drum. Greater excavating depths can only be reached by the sacrifice of dump height. Note that on some machines it may still not be physically possible to reach this depth because of the geometrical limitations and workface angles.

**TAIL-END WORKING (SWING) CLEARANCE**

This is the radius from the centreline of the machine to the rear or tail of the machine and represents the closest the dragline could stand to a wall or bench above its working level. It normally includes 2-4m freeboard and is important for;

- positioning the machine on the lower pass of many two-seam mining methods
- excavation of advance benches
- spoiling at maximum dump height where the base of the spoil pile is adjacent to the back of the machine, often encountered in side box-cut excavation techniques.
The range of geometrical conditions in which a dragline can work is largely a function of its operating radius. For the same operating radius, a larger dragline will simply move more material – it doesn’t accommodate a greater range of geometrical conditions to be handled. The following rules-of-thumb can be used in the selection of the boom length, boom angle and maximum suspended load for draglines:

- For any given size dragline, choose the configuration with the largest bucket – provided that the operating radius is sufficient to do the work.
- Choose the steepest boom angle, which will allow you to maximise dump height. It is valid even if it is not necessary to dump at this height - the loss in the bucket capacity on shallow angle booms is usually a lot more than what the saving in rehandling is, derived from greater reach.

### 7.1.3 Bucket Dumping Options

The simplicity and cost effectiveness of the dragline originates from the bucket configuration and the dumping mechanism. The center of gravity of the dragline bucket is forward ahead of hoist rope. The bucket is restrained from dumping by the tension in the drag rope, attached by pulley to the drag rope. There is always tension in the drag ropes until the bucket reaches under boom point. At this point, tension forces in both ropes are equal. Providing the balance of the bucket is correct, tension in the drag ropes will be transferred to tension in the dump ropes – and the bucket won’t dump. Figure 7.3 illustrates the rigging of the bucket.

As soon as the bucket is near or at the boom point, the hoist rope is vertical and if there is no longer tension in the dump ropes the bucket will dump. A properly constructed dragline bucket can as a rule not dump, except under the boom point. The dragline operator has little control over the bucket while it is in the dumping orientation.

Different procedures are used to actually cause the bucket to dump. The two most general are:

- Dump under boom point

The cleanest dumping is achieved right under boom point. To dump under boom point the operator must judge when the bucket is well placed and then stop for a moment to feed dragrope. That stops the outwards movement of the bucket and the lack of balance causes the bucket to dump - while dumping
is made easier at the same time by the further feeding out of dragrope. This form of dumping leads to all material being dumped in the same place. The bucket isn’t moved during dumping. It is achieved by the minimum amount of dragrope out and therefore also the minimum of dragrope to pull back before the restarting of the procedure. The technique is very important when dumping on extended benches, since any extra rope will drag on the ground. That will lead to dirt being picked up which will disrupt the smooth return of the bucket. A boom point dump does, however, add seconds to the dragline cycle time (in comparison to casting).

Figure 7.3 Dragline bucket rigging configuration

- Casting

This is the traditional dumping technique. It simply results in more dragrope being fed out until the bucket dumps. The bucket will start dumping as soon as it goes past the point directly under the boom point. Unless it is brought to a halt, the swing effect will carry the bucket past the boom point – while it dumps all the time. For the bucket to completely dump, significant extra amount of dragrope must be fed out as the bucket continues to move outwards and also because the (dumping) rotation of the bucket uses additional rope.
7.2 Dragline Selection

To understand and use the dragline’s excavating potential to the full, it is necessary that several analyses are undertaken prior to deciding on the use of a dragline. Foremost amongst these is the geotechnical investigation of the overburden in the proposed mining area.

7.2.1 Geotechnical Investigations for Dragline Mining

Although all types of surface mining are sensitive to geotechnical factors to some degree, it is in dragline strip mines that they assume their greatest importance. This situation arises from the basic geometric limitation of the dragline, specifically its dumping radius, a function of its boom length. Thus, while the dragline is ideal to remove overburden from a coal seam at minimum cost, it requires favourable stability conditions to be fully effective. It can still be used where stability conditions are not favourable, but only at the expense of modified geometry (longer boom/smaller bucket) or high rehandle volumes, either of which will inevitably erode its basic economic advantages. Thus, the dragline may not always be the optimum overburden removal method, even for overburden depths less than 50-60m.

The principal geotechnical parameters are:

- spoil pile stability
- highwall stability (including extended bench)
- floor stability

The variety of geological conditions, seam geometries and mining methods occurring in strip mines implies that geotechnical conditions cannot easily be generalised and specific deposit investigations are required, most usefully at the mine design – feasibility study stage.

SPOIL PILE STABILITY

Spoil pile stability is usually the most critical geotechnical parameter in strip mining, since the spoil pile is basically an uncontrolled, uncompacted heap of disturbed overburden, which can only be placed a limited distance from where it is excavated. Spoil pile heights are generally in the range 80-100m for current maximum dragline digging depths of around 60m.
When dumped, the overburden will roll to its angle of repose, which may range from 30° or less for clayey material to 40° or more for hard rock, but which is most commonly in the range 35 – 40°. The angle of repose applies only to the loose material on the free surface of the pile. Material at greater depth will undergo some degree of self-weight compaction and it is possible to re-excavate piles to steeper slopes. For example, the lower part of the pile is commonly cut to angles of 45° when the dragline bench is rehandled in the extended bench method, which is widely used to increase the basic dumping reach of the dragline.

Spoil pile failure can occur in two ways:

1. slumping of the face of the pile due to internal failure of the spoil material
2. translational failure of the whole spoil pile, usually as a result of foundation failure.

The first type of failure is usually the result of over-steepening of the face, as discussed above, and generally does not involve large volumes of material. Foundation failures are by far the most important, in terms of volume of material.

The stability of the pile is not very sensitive to spoil material strength, but it is highly sensitive to floor dip and foundation strength. Because of the large shearing deformations generated in the foundation during building of the spoil pile, cohesion is taken as zero and typical foundation (residual) angles of friction range from 14° for a horizontal foundation to 26° for a foundation dipping at 15°.

Low foundation strength can arise from three sources:

- weak, slaking prone material placed in the base of the pile,
- weak material lying on the previous floor of the pit,
- weak layers beneath the floor of the pit.

Water is also a major contributor to foundation failure, in the form of:

- water on the floor of the pit prior to placing the pile
- surface water ponding behind the spoil pile peaks and seeping through the pile and along the floor
- groundwater seeping upwards from the floor.

In dipping seams, there are important differences in stability conditions between advancing down dip and advancing along strike (see discussion in earlier Module). A conventional dragline operation advances down dip and the spoil pile must support itself on a dipping foundation, as discussed above. On the other hand, if the strip is oriented down dip and advances along strike, the spoil pile is buttressed against the final down dip wall and stability problems are substantially reduced. This method can be used with draglines under certain conditions, but it is generally more suited to shovel/truck or similar operations.

When it is necessary, the stability of the spoil pile can be improved through the use of a variety of additional mining method variations, namely;

- Selective placing of the waste (soft, slippery material for example clay and shale) on top and not in the bottom of the waste dumps

- Stripping or the blasting of the floor surface of the pit to give a better friction surface (especially if floor is a low-friction material – carbonaceous shale, etc.)

- Excavation of clay materials (typically shales) from the floor to expose material with more frictional resistance. That will normally require a bench extension working method together with the additional specialised handling of clay material (which will result in increased waste handling costs and therefore reduced profitability).

Note that small scale changes in floor (or coal seam) elevation or inclination are not as problematic from a geotechnical point of view, but nevertheless create extremely difficult working conditions when dragline mining is used.

**HIGHWALL STABILITY**

In a dragline operation, two long walls are formed in the initial box-cut. One of these, the low wall, remains and is eventually covered by spoil. The other, the highwall, is progressively excavated and occupies a new position with each strip. The maximum highwall height is usually of the order of 45-50m, with a variable depth of chopping or prestripping above the dragline working level.
The stability of these walls will be dependent on:

- for soft materials, the nature and strength of the material
- for hard materials, the patterns of bedding planes, joints and faults including weak layers within the overburden and coal seams (joint fabric)
- groundwater conditions.

In soft materials, e.g. alluvium, weakly cemented sediments and highly weathered rocks, wall failure will occur by slumping along a curved failure surface passing through intact material. In materials where intact strength precludes this failure mechanism failure can occur along simple or complex surfaces formed by pre-existing defects in the rock mass. Bedding dip is again an important factor in wall stability and specific design procedures are required for steeply dipping seams.

In harder materials, backbreak from overburden blasting can have an important influence on wall stability. For this reason, the use of initiation sequences designed to minimise backbreak is highly recommended.

Groundwater pressures can have an influence on stability if high pressures exist close to the face of the wall. In most cases, conditions will be more difficult in the initial box-cut than in subsequent strips, because of natural drawdown which occurs with time.

Wall stability and the design highwall slope must also be considered in terms of the consequences of failure. For example, in a dragline operation, a more conservative approach must be adopted for a free-standing high wall than when the high wall is buttressed by a spoil bench, as in the extended bench method. The safety of the coal extraction operation, in relation to rocks falling from the high wall, is also of paramount importance. Time dependant effects must also be considered, based on the turn over time of a strip and the length of time the highwall must remain standing. Lastly, when using extended benching methods and sourcing material from an advance bench chopdown, care must be taken if this material is weak and weathered (as is usually the case near the surface). Used as a bench extension, it could cause foundation failure or translational slides as the dragline walks out onto the extension to spoil the remaining overburden. Again, the role of groundwater would compound the stability problem.

A special set of considerations apply when mining over previously mined areas. The concept has already been discussed in an earlier
module and the significant working method, that of a subsidence hazard plans, already identified. Underground workings may give rise to a void migration that may extend upwards to more than half the height of the excavated slope. Specific highwall instability due to partially extracted seams or shallow mine workings and voids within slopes, can be classified as transitional slides, toppling failures, span failures, and slab sliding.

- **TRANSITIONAL SLIDES.** Three forms of transitional slides can be identified, the first of which concerns the formation of ancient bell pits. The compaction of infill debris is variable, but, in general, the pre-formed, bell-shaped mining voids are often stable within the excavated slopes. However, under-cutting of the slope can occur. A second mode of transitional slide is a result of movement along bedding planes which have been downwarped and tilted towards the apex of a migrated room void. The third type of transitional slide may be either plane or wedgelike. Here, open-jointed overlying strata aggravate these failures, which are triggered primarily by the failure of arch infill and sometimes by failure of sheared coal pillars at the toe of the slope.

- **TOPPING FAILURES.** Where a strong bed overlies arch infill which intersects a working face, a toppling failure is likely to occur. Forward movement and collapse of infill, as a consequence of excavation, leaves an unsupported, overhanging beam of strata in the upper part of the slope. This slope fails by toppling, usually with some rockfall.

- **SPAN FAILURES.** Failure of rocks spanning a void can occur because the inner edge of a pillar fails at the toe of the slope. The slope toe collapses into the void and the face topples or slides forward.

- **SLAB SLIDING.** The interaction of room and pillar workings and dip become important where dips are in excess of 18° and slope heights are greater than 25 metres. This type of failure leads either to compaction of the arch infill or to the buckling of the roof strata which spans the infill.

The strip block geometry also has a large influence on the highwall stability. In terms of strip width, slope stability over previously mined areas can be improved by selecting the strip width so that the slope toe rests on a whole pillar, or by working over elongated rectangular pillars with their short sides parallel to the excavated face. In terms of strip layout (direction), two factors are critical, namely;
- Favourable or unfavourable joint sets, shear planes or fault orientation. The location of joints or discontinuities are signs of potential weakness in the rock mass and may affect highwall stability, but their influence may usually be greatly reduced by adopting a highwall direction at right angles to the major feature.

- The nature of the underground workings. If the strip is orientated parallel to the workings, then the highwall will sometimes intersect bords with no pillar support. If the highwall runs at an angle to the workings, there will always be some pillars at the face. Even though a regular pattern of pillars is present underground, the axes of the pillars can change at intervals to cater for the changing mining conditions encountered. In addition, the mining control and survey quality may result in unforseen axis changes which may affect stability.

Stability of highwalls over previously mined areas is also a function of the mining method selected (how the voids are treated during the blasting and coal recovery operation) and the dragline working method. In extreme cases, it may be necessary to consider spoil pull-back with draglines on the spoil-side.

**FLOOR STABILITY**

Floor stability problems can arise from;

- floor buckling as a result of spoil pile failure by sliding on a weak layer beneath the pit floor level

- heaving as a result of water pressure in a confined aquifer beneath the pit floor level.

The first mechanism has been discussed above and, although it may generate operating problems with coal extraction, is not generally critical. Heaving due to subfloor water pressures is potentially more serious since uncontrolled floor heave could lead to rupture of the aquifer confining layer and flooding of the strip pit.

The rock types and structural domains within and beneath an excavation determine the behaviour of ground water. The aquifers in the coal measure are typically coal and sandstone. The impermeable zones, aquicludes, include mudstone, seat earth and fault gouge. Siltstone is an intermediate material and is termed an aquitard. Structural discontinuities within the rock mass can act as preferential flow paths.
7.2.2 General Dimensional Relationships for Dragline Mining

The relationship between the strip mine geometry, geotechnical design and the dragline geometry form the basis of dragline selection. Two techniques can be used to relate mine- to dragline geometry, namely:

- A mathematical derivation – a two dimensional estimation of what is in reality a three dimensional problem. The effect of the third dimension (pit length) is not so important except where the dumping space is limited (e.g. in the vicinity of low wall roads and ends of the strips)

- A graphical technique (range diagram) – used by the majority of computer-based pit design packages that include a rigorous 3-dimensional approach to pit design and dragline position during excavation and dumping.

For a basic understanding of the geometric relationships, assume a single bench of overburden above the coal seam and a dragline using a normal side-casting method. The parameters, which are used in these calculations, are:

- \( \beta \) = High wall inclination (from horizontal)
- \( \Theta \) = Waste spoil pile inclination (from horizontal)
- \( D \) = Overburden depth (m)
- \( O_R \) = Dragline operating radius (m)
- \( P_f \) = Dragline positioning factor
- \( R_F \) = Dragline reach (m)
- \( d \) = Dragline tub diameter (m)
- \( s \) = Swell factor (of overburden material) (%) 
- \( W \) = Block or strip width (m)
- \( h \) = Height of spoil peak above coal (m)
- \( T \) = Coal thickness (m)

These values are further defined in Figure 7.4 and 7.5.
A – Swing radius
B – Operating radius (OR)
C – Boom foot radius
D – Tail clearance height
E – Boom foot height
F – Dump clearance
G – Dump height
H – Boom point height
J – Digging depth
K – Point sheave pitch diameter
L – Tub diameter (d)
M – Boom angle

Figure 7.4 Dragline dimension terminology

Figure 7.5 General strip-mine geometry for simple casting method, with dragline axis of rotation located at ‘C’.

The two components of the analyses are the strip, which is going to be excavated (from the highwall) and the area that represents this material on the waste side (material is dumped on the lowwall side) of the strip (Figure 7.5). Waste pile area ($A_w$) is the area cut ($A_c$),...
increased with the swell factor (S), or the area can be calculated mathematically as (D x W).

Area of cut $A_c$ is thus:

$$A_c = D \times W$$

And the equivalent spoil waste area $A_s$ is thus:

$$A_s = D \times W(1 + s)$$

Through geometry the area $A_s$ can be found from Figure 7.6 if;

$$A_s = WT + Wh - \frac{1}{2} W \frac{W}{2} Tan\theta$$

combining gives:

$$DW(1 + s) = WT + Wh - \frac{1}{2} W \frac{W}{2} Tan\theta$$

therefore:

$$D(1 + s) = T + h - \frac{W}{4} Tan\theta$$

or

$$h = D(1 + s) + \frac{W}{4} Tan\theta - T$$

The reach factor ($R_f$) is the sum of the horizontal projection of the overburden and the highwall slope. Thus:

$$R_f = DCot\beta + (h + T)Cot\theta$$

or

$$h = \frac{(R_f - DCot\beta)}{Cot\theta} - T$$
Again, combining expressions for h gives:

\[ h = D(1 + s) + \frac{W}{4} \tan \theta - T = \frac{(R_f - D \cot \beta)}{\cot \theta} - T \]

thus:

\[ R_f = \frac{D(1 + s)}{\tan \theta} + \frac{W}{4} + \frac{D}{\tan \beta} \]

If the waste runs up against the coal face to a height (t), then there will be a loss of coal (since some coal will be left to "support" the toe of the spoil) and, in addition to possible coal dilution, such a situation will lead to poor drainage conditions. The equation of the reach factor is therefore also changed, as shown in Figure 7.7. Similar modifications are required to the formula for \( R_f \) if a drain is left right next to the coal, or a space is left next to spoils for dumping parting as the coal is mined, as shown in Figure 7.8.

For coal run-up:

\[ R_f = \frac{D(1 + s)}{\tan \theta} + \frac{W}{4} + \frac{D}{\tan \beta} - \frac{t}{\tan \theta} \]
Figure 7.7 Spoil run-up against coal

For drain of width $W_d$:

$$R_f = \frac{D(1+s)}{\tan \theta} + \frac{W}{4} + \frac{D}{\tan \beta} + W_d$$
Figure 7.8 Drainage slot or parting dump next to spoils

The operating radius ($O_R$) of the machine is the distance from the machine's rotation axis ('C' in Figure 7.5) up to the boom point vertical position – usually the maximum extent of the spoil peak when the machine is rotated normal to the edge of the highwall. The operating radius is a design number which is used in the choice of a dragline.

Thus:

$$O_R = R_f + P_f d$$

During operation, the dragline takes a part of the total operating radius up as positioning. That is the space which the machine needs to position itself safely on the work level. If the tub of the dragline is just on the edge of the bench then $P_f = 0.5$. In this case there is no space on one side of the machine for the walking shoe and the machine must walk backwards perpendicular to the bench face (it is dangerous and wastes time). Usually, positioning includes 25 – 30% more freeboard to ensure that the machine is not exposed to bench failures or instability and can move safely in any direction, especially parallel to the face. Figure 7.9 illustrates the concept.

![Diagram of dragline positioning](image)

Figure 7.9 Positioning of the dragline on the bench

In this case, the required operating radius is given by:
In conclusion, the dragline dump height \( H_s \) (the height above the dragline working level at which waste is dumped) is given by:

\[
H_s = h - D
\]

Dumping height is usually not a limiting factor except when a box-cut is made, where the dragline must dump on its working level, as shown in Figure 7.10. In this case, since there is no previous strip in which to throw the spoils, the final height of the box-cut spoil will be much higher than the dragline maximum dump height (depending also on angle of repose of spoils). Under certain circumstances (e.g. at the end of the pit or where low wall roads pass through the waste), dumping height limitations can develop even when dumping into the previous strip void.

Figure 7.10 Dumping height when a box-cut is excavated.

A final selection dragline selection would be much more detailed than the basic analysis presented here. Some of the questions that would additionally be checked would be as follows;

- Check that changes in highwall and spoil pile angles with changing material types across the property would be accommodated
- Check possible swell factor changes with material type
- Check the maximum digging depth requirement (or use a least-costs approach to define dragline and pre-strip optimum depths)
- Check the stacking height requirement at maximum bench height
- Check if there are any areas in the mine which would require extended periods of stripping depths greater than design - could coal production be maintained throughout these areas?
- Check machine bearing pressure against proposed bench material specifications.

As can be seen, a whole range of questions must now be checked to ensure the selected machine will perform its proposed task. The final selection will be made following a balancing of all these residual requirements.

7.3 Factors Influencing General Dimensional Relationships

HIGHWALL AND SPOIL ANGLES
Stability of highwalls and the spoil angle should be evaluated during the geotechnical investigation stage of mine design since the principal effect of this on dragline operations is to limit the effective operating radius. This limits the total thickness that can be spoiled, or the total depth that can be dug. It does not impact greatly on rehandle, except if the highwalls are excavated too steeply and then collapse.

OVERBURDEN DIGGABILITY
If overburden is properly blasted, dragline productivity is largely insensitive to this. The greatest problems occur when harder ground is encountered; bucket fill times and wear rates increase. Since it is common to have several months blasted inventory ahead of the dragline, the blast design problem is often not recognised and corrected before considerable hard digging and reduced productivity.

Large fragments in the overburden causes the operator to slow the drag speed and ‘feel’ through when filling the bucket to prevent stalling the drag motors causing large currents through the motors which will adversely affect the motor life. The fill of the dragline
bucket will also be affected as more of the bucket volume will be taken up by voids. The larger the fragments the less BCMs will be filled into the bucket. Well fragmented material may result in a 92% fill of the bucket, whereas poor fragmentation may result in a lesser fill of 85% and a swell of 30%.

OVERBURDEN SWELL FACTOR
High swell factors limit the total thickness of waste which can be handled, because of the increased spoil height and reach required. On the other hand, when using extended benches, rehandle may actually be reduced for higher swells, because less material is needed to make the extensions.

INFLUENCE OF PIT WIDTH
Pit (or block) width was identified as a factor in dragline size selection, based on the reach and operating radius required. If dragline reach is reduced, a larger bucket can be employed, improving the rate of overburden removal (but also, in deeper overburden areas, requiring more rehandle since reach is limited). Aside from dragline selection, there are a number of operational considerations surrounding the selection of pit width.

The choice of pit width is usually made in the planning stage with little or no leeway for change once mining starts. Only when a dragline is stripping cover well within its maximum capability is there any real choice of width. A wide pit, of say 50m, is less congested and easier to operate as regards coal loading and hauling than a narrow pit of say 25m width. One of the primary considerations is will the haul trucks have to turn and if so what width do they require for an efficient operation. For a single seam dragline operation haul trucks do not have to pass the stripping equipment in the cut. Other less important factors might be the possible location of sumps, dewatering pumps, in-pit dumping of coal seam inter-burdens or partings in the void between spoil and coal and catering for coal drilling and blasting operations. For most properties a target width of 40m, squeezing down to 30m in problem areas, produces a good efficient operation.

Other considerations associated with pit width and in-pit operations centre on the interaction between the dragline and the coal loading and hauling operation. These tend to be application specific, but the most important parameter is the variations in rate of coal exposure as compared with the required coal mining rate. This is especially important where cover heights above the coal vary widely along the length of the cut, since coal exposure rate will vary and cut width...
variation (to balance exposure rates) is not a viable option. It must be ensured that the dragline or draglines can expose coal at the same rate at which it must be mined, perhaps not at maximum depths of cover, but if for certain short periods the coal exposure rate is lower than the mining rate, the dragline must be in a position to catch up quickly without the likelihood of loss of production.

7.3.1 Selection of Dragline Bucket Capacity

Once the physical parameters of the dragline have been established, there is another equally important factor that must be found, namely the bucket size. It is usually calculated in rated cubic meters and is critically dependant on reliable estimations of the following parameters;

Mine and design factors;

- Proposed coal production rate and coal seam thickness and recovery ratios
- Overburden stripping ratio
- Material swell factor
- Scheduled operating time

Machine factors

- Bucket fill factor
- Dragline cycle time
- Utility (available time dragline spends actually excavating material).

The machine factors are the most problematic to determine and usually show high variability depending on the actual block of ground being mined.

The bucket fill factor is used to determine the amount of material in the bucket while digging. Digging above the dragline horizon (chopdown) normally results in a lower bucket factor than normal operations. Also, deep digging results in lower bucket factors. As a rule-of-thumb for the bucket fill factor, 77% would be typical. I.e. a nominal bucket size of 68m$^3$ would actually contain 52.3m$^3$. 

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The dragline cycle time is comprised of the following activities:

- **Bucket fill time**: To achieve fast, efficient bucket loading, it is essential to maintain a steep working face in order to use gravity effects as much as possible. This is, however, much more important in rock material than the softer clays. Filling times vary from as little as 5 seconds to 20 seconds; digging (dragging) rehandle is a fast fill, normal bench excavation intermediate fill and long fill times are associated with deep digging requiring casting, clean-up of the pit floor; and key-cuts. Drag speed is generally less important than drag power and payout speed. The operator can seldom achieve top speed while dragging to fill the bucket. As a rule, maximum drag payout speed must be used to keep pace with the hoist speed when the bucket is full and being swung and elevated simultaneously to dump on the spoil pile.

- **Swing time** (laden bucket): The time from when a bucket is full and lifted out of the working face, until it is dumped. It is dependent on either the time to hoist the bucket to the required dump height, or the time to swing around to the dumping position. Most swings are ‘swing-time dependent’, the typical swing time for 90° swing - 18 to 22 seconds. ‘Hoist-time dependent’ swings occur for high hoisting in the lowest passes in multi-seam mining applications and during spoiling of boxcut waste.

- **Bucket dump time**: The dump time is usually zero since the bucket can usually be dumped in the final stages of the outward swing, and in the initial stages of the return swing (casting, not boom point dumping). Only in special circumstances is it necessary to allow the swing to come to standstill before starting bucket dump.

- **Swing time** (empty bucket): This is rarely hoist dependent because draglines can lower the bucket faster than they can raise it. Swing return times are 1-2 seconds faster than swing out times. The angle through which the machine swings between the dig and dump positions is known as the Swing Angle and this angle is ideally minimised by working method and machine position on the bench.

- **Bucket positioning**: Good operating practice involves starting the bucket digging before the dragline has completed the return swing – this is a means of better controlling the dig positioning and drags material in from the edge of the bench and helps maintain a steeper digging face.
The above five components determine how long it takes to complete a single cycle excavating a particular block of overburden. The product of the cycle time and the number of swings is the total excavation time for the block, excluding additional time losses due to reduced digging time (utility).

Utility is determined as a factor to account for delays covering short periods of time included in the mining cycle time. These additional delays include:

- Trimming and dressing the digging face – related to the quality of blasting and diggability of the resulting muck-pile
- Brief waits for other equipment – dozer backup for cable handling, etc.
- Operator/personnel delays
- Small adjustments to the dragline – most usually unscheduled delays to change or inspect bucket rigging or teeth
- Occasional bad passes requiring drag payout and bucket repositioning
- Cleaning to the coal surface or roof contact
- Short moves for machine positioning and walks – these are often the most time consuming element since from the completion of a swing to the start of a walk, it is necessary to bring the bucket into position, orient the dragline in the direction of walk, wait for a signal from the ground crew that the cable is secured or moved, engage the propel motor, and commence the walk. A similar process is followed in reverse at the completion of walking. Typical times for set-up (including time to commence digging again after the walk) range from 3-4 minutes (small walk away from cable), to 7-9 minutes (longer walk requiring lining up, walking towards the cable and cable handling).

There are a few additional points that should be carefully noted. The first of these concerns stripping methods that involve rehandle. The annual BCM requirement must contain all the overburden above the coal as well as the additional rehandle volumes. One of the most important considerations is adjustment of the cycle time and bucket fill factor that will be necessary when different modes of dragline operation are included in the stripping method. For example the rate of dragline production will be normally greatly reduced when the
The dragline is used in the chop down mode as compared with the conventional operation. Similarly a pull back operation may be at a reduced production rate, especially when extra deadheading time, etc., needed to get on to the spoil pile, is included.

When these factor variations are fully enumerated, bucket capacity $K$ ($m^3$) can be found from:

$$K = \frac{Y C (1 + s)}{O A U B}$$

Where:

- $Y$ = Yearly overburden stripping (BCM) required to expose annual coal sales
- $C$ = Cycle time ($s$) per bucket load
- $s$ = Swell factor ($\%$)
- $O$ = Operating time per year ($s$)
- $A$ = Availability ($\%$)
- $U$ = Utility ($\%$)
- $B$ = Bucket fill factor ($\%$)

The final stage of the calculation is to determine the rated suspended load ($RSL$). The rated suspended load is the maximum load that may be suspended from the boom of the dragline, excluding the ropes, comprising the weight of the bucket and the weight of the material in the bucket.

Thus:

$$RSL = K \left( \rho_{\text{bucket}} + \rho_b \right)$$

Where:

- $\rho_{\text{Bucket}}$ = Bucket unit weight per cubic meter bucket capacity
- $\rho_b$ = Rock bulk density ($kg/m^3$)

In general the bucket unit weight is ± 700-1400 kg/m$^3$ of the bucket capacity, depending on digging duty.
The RSL must not exceed the manufacturer's maximum suspended load (MSL) specification, which usually includes the mass of the bucket rigging and ropes at full (hoist) out.

The density of the material as encountered in the pit after blasting (also referred to as the rock bulk density) is difficult to determine, as it will vary from blast to blast. The amount of explosives used and the blasting technique applied plays a role in determining the bulk density of the material. To determine the bulk density in the field, the blast profiles may be used and the swell determined from these profiles may be applied to the relative density to estimate the bulk density. However, during the excavation process the material is compacted to a certain extent, giving rise to a higher bulk density than the estimated density. To compound the problem further, the material being excavated is also compacted in the bucket as the bucket is dragged through the material to fill.

As the density estimate is the major criterion in calculating bucket capacity, a system of estimating density by using the hoist current of the dragline is now commonly used. The model is such that it discounts the weight of the ropes during the digging cycle as this does not form part of the load calculations. The weight of the bucket can be measured and the weight of the material determined through deduction.

In conclusion, the design parameters calculated namely:

- Reach factor
- Operating radius
- Excavating depth
- Dumping height
- Bucket capacity
- Maximum suspended load (MSL)

can be used in conjunction with the manufacturer's specifications to decide on the specific type and series of a machine. Simple extensions to the basic principles described here in the dragline size selection procedure for the simple side casting case are, with care, usually sufficient for alternative stripping methods. This is the case as regards the size selection alone but is not the case as regards the selection of the most economical stripping method. With the introduction of rehandle, chop down, extended benching, etc. the
basic principles must be enlarged. The dragline size selection and operating method permutations now become greatly increased when the economics of these various operating modes are considered.

For example it is not necessarily more economic to select a stripping method with no rehandle as opposed to one that has rehandle. A dragline having a long boom, and correspondingly smaller bucket size, stripping without rehandle may be a more costly operation than the same machine with a shorter boom, with a larger bucket and greater stripping rate, operating with a certain amount of rehandle. Similarly, a limiting depth will be found for each machine size at which a more economic combination of overburden handling – dragline side casting, advanced benching, extended benching or truck and shovel pre-stripping, can be found. A rigorous selection methodology framework is required within which to make this selection.

7.3.2 Dragline Selection Methodology

In the geometric and bucket selection methodologies presented in the previous section, the assumption was made that the coal seam had no dip, or was of such a value that the difference in waste block volumes and spoiling volumes was minimal. In most cases this will suffice for a strip-by-strip analysis of dimensions, but it nevertheless does not indicate to what depth dragline working is cost-efficient, nor does it allow for the selection of the most suitable dragline size.

To determine the most suitable dragline size for an operation, it is necessary to consider the volumes of overburden to be moved and how they vary over time. In as much as truck and shovel systems would not be considered to the full depth of the overburden, neither would a dragline be considered able to handle (cost-efficiently) the full depth. The question is what size of machine and to what depth?

One approach involves a block-by-block analysis of dragline productivity and the incremental cost of increasing overburden depth. Using this approach it is possible to determine the cost-effectiveness of a dragline system, compared to another system (e.g. truck and shovel) and what is the economically optimal depth for each. Figure 7.11 shows a typical example of dragline productivity and BCM incremental costs with increasing depth of overburden. Note that with increasing depth of overburden, the extra cost of the extra BCMs increases rapidly and at some depth, it would be cheaper to use a pre-strip (for example, in Figure 7.11, the 5m overburden cover above 50m costs an extra R4.80/BCM to strip with a dragline - a cheaper system may allow this material to be stripped at a lower cost).
Figure 7.11 Typical cost variation with increasing overburden thickness

For a mine-wide life-of-mine approach, the solution is based on simulation of alternative mining sequences, sizing and costing of pre-stripping equipment and draglines based on the sequence, and determination of the least cost alternative from a financial point of view, present worth, discounted average cost, etc. Figure 7.12 shows a flow-chart of the approach used.

Figure 7.12 Dragline selection methodology
In the approach, it is necessary to simulate a mining sequence and generate overburden requirements in terms of quantities and depths on an annual basis. The annual coal production requirements, resource summary, and planned pre-stripping depths must be known. The summary provides an indication of the characteristics of the resource in terms of depth and quantity of overburden relative to the resource. Cumulative coal to a particular depth is a key in the evaluation of digging methodologies and thus the economics associated with the resource.

Given coal production requirements over the life of the mine and the expected mining recovery, the total in-situ resources required to meet the production can be calculated by summing annual production divided by mining recovery. With the in-situ resource determined, the total overburden associated with the resource can be calculated by summing the overburden increments until the coal resource equals the in-situ production requirement. This assumes that;

- coal with the lowest overburden cover would be mined first
- the geometry of the property is such that this is a feasible alternative.

The maximum mining depth satisfying the total in-situ tonnage requirement is estimated by assuming the overburden increases at a linear rate within the increment.

In a multiple-seam resource, the overburden increments would include overburden plus interburden depths and quantities and coal in-situ would be cumulative tonnage of all seams. Using this approach on a multiple-seam property would assume simultaneous mining of all seams, not seam by seam removal, as in an actual operation. This is not a problem if dragline overburden production versus depth data properly accounts for average production rates in multiple pass stripping.

Pre-stripping is the removal of overburden ahead of the primary dragline stripping operation. Given a maximum dragline digging depth, all overburden in excess of this maximum depth is considered pre-stripping material and is assigned to the pre-stripping equipment. Material below that depth is removed by the dragline(s).

With the mining limits determined, two basic approaches may be taken to sequencing the actual mining;

- The first is an averaging sequence that involves mining from the coal outcrop or sub-outcrop to the mining limit, or from low to
high overburden increments in an effort to “average” the overburden depth within a given pit. It should be apparent that the equipment selected must be capable of handling the maximum depth and average quantities in the initial phases of the operation. With constant production, these requirements will remain constant throughout the life of the property.

- A second approach involves selection of a sequence that minimizes the overburden depth and quantities in the early years of operation by mining shallow coal first. This tends to minimise the stripping ratio and overburden quantities and, thus, equipment requirements in the early years of the operation. However, the equipment requirements will be at a maximum in the final years of operation when a high stripping rate and overburden depth is the norm. It is apparent that these requirements will be greater than those in the averaging case.

With the stripping requirements known, the next step in the involves calculation of scheduled hours, based on production rates, and machine requirements and costs, based on scheduled hours for various models of draglines available. Of the draglines tried, an economic evaluation is performed to determine the least-cost alternative to meet the production requirements. It is important here to use the appropriate production rates - for a given machine, property, and pit design, the production rate can be reduced to a relationship of rate versus depth of overburden. This is important in machine selection, since the rate of production in terms of BCMs per hour may vary by 50% or more over a typical range of machine-digging depths. Thus, in order to be realistic in estimating machine requirements and also responsive to actual coal resource characteristics, this factor must be built into the selection procedure.

The second critical factor in the selection procedure is the appropriate treatment in an economic analysis. When selecting a specific model from an array of possible machines available to perform the necessary stripping, mutually exclusive alternatives are being considered. Consequently, an approach that accounts for differences in capital and operating costs, depreciation, etc. and timing of these costs is needed.

The number of machines is based on annual scheduled stripping hour requirements and then comparing these to the specified maximum dragline scheduled hours per machine. Annual scheduled hours are a function of the annual overburden which is known from the sequence simulation, and thus the production rate can be
determined from the production rate versus depth curve of the dragline being evaluated.

With dragline requirements expressed in terms of number of machines and annual scheduled hours, it becomes a matter of apportioning capital costs and calculating annual operating costs, given the appropriate capital and operating cost inputs. When pre-stripping is employed, the capital and operating cost calculations are similar to those discussed for the dragline, with the exception that production rate is input as a fixed rate per year. In addition, since a portion of the pre-strip system may have to be replaced, provision is made for replacing a percentage of the initial capital investment after a specified number of pre-stripping system operating hours. This provision is employed, for example, to account for replacement of trucks in a shovel-truck system.

It is clear that the single dragline size selection can become a very complex undertaking when a series of different stripping methods appear economic. These problems are not diminished with the addition of one or more machines. The combinations of machine dimensions, modes of operation etc. can be many but most operations, however, usually have some restrictions that greatly reduce the possible stripping combinations even to the point where one combination is specifically indicated.

Some additional points to be analysed include the careful evaluation of rehandle volumes. Almost all multiple machine stripping methods include rehandle and some include the rehandle of material more than once. On occasions the result can be rehandle values of 100% or more. Additionally, each machine should be carefully analysed to establish its exact movement requirement and the various machine stripping rates must be matched to ensure a smooth continuous operation. If one machine takes appreciably longer to strip its requirement then the whole operation must be geared down to that rate.
7.4 Dragline Operating Methods

This section will cover the development of various dragline operating methods for a single coal seam strip mine. It is assumed that the following mine planning considerations (see Module ten for information on the planning process) are established and tested for the suitability of dragline mining:

- Coal sales and reserves can justify the capital cost of the machine.
- Depth, thickness and geological characteristics of the overburden is known.
- Planning of the surface facilities, streams, roads, informal settlements, etc. are redirected or re-established away from the mining area.
- The blocks of coal to be developed are identified and the strip length maximized. The blocks must where possible be rectangular, typically 50m or so wide and 200 – 300m long, at least 10-15 blocks in a strip. The dragline cannot easily mine blocks that are not rectangular. Short strips result in excessive dragline maneuvering time, and not enough in-pit or floor reserves of drilled and blasted overburden and drilled and blasted coal.
- The whole block layout must allow that the dragline always has a dump area available. That avoids the necessity of extra box-cut – as a result, each strip can be smaller in length than the previous – but never longer.
- The position of the box-cut is established – usually coincident with the minimum stripping ratio or maximum profitability contour. Also, the direction of mining is determined with due regard to:
  - Profitability and surface contours
  - The dip of the coal layer and the variations in the quality
  - Geological displacements and intrusion, etc
  - Drainage factors
  - Old surface or underground workings
- The general mining plan – where possible the strips must be laid out in a convex shape with a development direction from the inside towards the back (to maximize the waste spoiling area)

- Topographical variations – thinner overburden improves the immediate stripping ratio while a lot of ridges or valleys result in expensive ground preparation or pre-stripping work (to walk dragline on even surface)

- Rehabilitation design of original box-cut and final strip (usually a valley gets left along the last strip and a hill alongside the original box-cut)

- Profit contours, which will indicate most profitable areas (function of overburden depth, dragline mining method and coal quality).

A single seam of coal can be exploited through the use of one or more draglines. The choice of the machine size depends on the parameters of the pit, as described previously. If the mine is a new operation, the pit parameters can be designed to fit the selected dragline operating radius. Draglines from another area or property can be relocated from an older mine also (as discussed later), but in either case, eventually the overburden or coal geology can vary so much from the estimates that the available dragline parameters tend to prescribe which stripping method must be used.

The main activities which lead to the exposure of the coal layer with a dragline based mining method are:

i. Removal of vegetation and the top soil (usually referred to as soil A and B horizons) – bowl scrapers are used for the removal of the top layers and the transportation of it to a special top soil storage pile, for later replacement on the flat and the leveled waste (start of the rehabilitation process). Depth of top soil stripping is established by the Director of Mineral Development and/or the EMP.

ii. Drilling and blasting of the overburden material.

iii. Preparation for the dragline pad (leveling, smoothing, covering with gravel or shales (if the ground is clayey) if the method requires it).

iv. Box-cut (for new mine or new area, once only to establish strip)
End-cut

Side-cut method

v. Removal of the overburden (next strip after a box-cut or existing strip) with casting method:

- Simple
- With pre-strip (or chop-down)
- With bench extension
- Or other more advanced modifications of these basic methods

The last two activities (iv and v) will be discussed individually in detail.

7.4.1 Box-cut

The first cut made to start a strip coal mine is known as the box-cut. The name box-cut refers to a cut in the shape of a box, which is made in the blasted overburden to form the new high wall and to expose the coal. Three sides of the pit consist of high walls, the fourth (usually parallel to the direction in which the dragline works) is ramped down to give the trucks and loaders access to the coal seam. The ramp runs at a slope of 5-8% from the surface. Such ramps can also be placed at the end of the cut or on the high wall side, but it complicates the scheduling of the operation (even though it is very advantageous for rehabilitation). Figure 7.13 shows a typical box end cut method.

It has already been shown that dragline dumping height (G in Figure 7.4) is a limiting design factor in the application of the machine. Because the dragline rather dumps the waste on the surface instead of in the previous strip void, the available dumping height is often reduced. This limitation can be over come in two ways:

i. By dumping the material of the box-cut to both sides of the box-cut and the material on the high wall side gets rehandled with shovels and trucks and transported to the waste side, or if the overburden is already blasted underneath the high wall waste dump, then the dragline handles it again on the next cut (if the reach and the excavation depth of the machine allow it). Figure 7.14 show the method.

ii. Borrow pit: a shallow pit is firstly dug with a dragline next to the proposed box-cut position. The depth of the borrow pit
must be sufficient to allow all the material of the 'true' box-cut to be dumped on the waste side of the cut. Figure 7.15 shows the method. The disadvantage of the method is that it sterilizes coal reserves below due to the large surface area taken-up by the spoil piles (of the borrow-pit and box-cut).

Figure 7.13 Plan and cross sectional view of a box pit, end cut method

The two methods for the development of the box-cut; the end cut or side cut method are similar, the only difference is the dragline position. The end cut is used in areas where the overburden is thin and has the advantage of smaller swing angles for the dragline. When the overburden is thicker, the volume of waste that has to be excavated requires more dragline reach. That can be achieved through the placement of the dragline on the side of the cut, between the box-cut and the waste, consequently the reach is effectively increased. The main disadvantage of this method is the swing angle, which is much larger and thus the productivity of the machine...
(BCM/hr) reduces, also the fact that more surface area is taken up by the dragline (reserves are sterilized or more surface must be purchased). Figure 7.16 shows this method.

**Figure 7.14** Plan and cross sectional view of a box pit, rehandle end cut method.

**Figure 7.15** Cross sectional view of a box pit, borrow-pit rehandle side cut method
7.4.2 Simple Side Casting

Simple side casting is the most common stripping method. The dragline stands on the prepared pad (smoothed and blasted overburden) and digs down to the coal seam and dumps the overburden material on the waste dump which is in the void of the previous cut (once the coal has been removed).

Side casting a block of overburden consists of two separate processes:

i. Key cut

The dragline stands in line with the required new high wall position and excavates the key cut. The remaining overburden serves to stabilize the bucket, thus a stable high wall can be dug (otherwise there will be no material to keep the bucket near the high wall).
Other advantages of the use of the key cut are given in Table 7.2 and Figure 7.17 illustrates the position of the dragline during key cutting.

Table 7.2  Key cut advantages

<table>
<thead>
<tr>
<th>Advantages</th>
</tr>
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<tbody>
<tr>
<td>Reduction of excavation and highwall stability problems</td>
</tr>
<tr>
<td>Reduction of the average swing angles for the total cut.</td>
</tr>
<tr>
<td>It allows for the excavation of the material in a direction away from</td>
</tr>
<tr>
<td>the new highwall – it doesn’t get damaged or undermined etc.</td>
</tr>
</tbody>
</table>

Figure 7.17  Plan and cross sectional view of the simple side casting method illustrating the removal of the initial key cut
ii. Casting

The dragline is moved to the outer edge of the blasted overburden block and the remaining material is cast to the waste side.

The main reasons for this apparent time consuming action is that:

- The dragline can use it's maximum reach distance – it stops the waste pushing up against the coal seam with the accompanied losses and reduces the necessity for rehandling.
- Gives better visibility to the operator.
- It allows for a steeper loading face to be maintained and the bucket is filled quicker and the cycle time is reduced.
- The wear on the drag rope is reduced with the development of the excavation path.

Figure 7.18 illustrates the removal of the remaining overburden once the key cut is taken and Figure 7.19 shows the dragline repositioning to maximize reach and minimize swing angles.

The dragline moves back after casting is completed to lengthen the key cut and the cycle starts again. While dumping is taking place the operator must observe the following points:

- Acid material (rock types or minerals in the overburden that oxidize) and large rock blocks must be positioned to the bottom of the spoil pile.
- Clay soil must be kept high in the waste pile and spread to avoid formation of slip surfaces. It would otherwise reduce the spoil pile stability.
- The operator must not ‘peak’ the waste if the overburden is shallow, i.e. - a continuous crest must be maintained, which at a later stage will help with the implementation of rehabilitation.

7.4.3 Advanced Bench Chopdown

The simple casting method is often modified by use of the advanced bench chopdown method. It is done by forming an excavation pad on a level lower than the ground surface. The main reasons for this are:
- It makes a continuous smooth surface for all areas even where it usually is naturally hilly

![Diagram](image)

Figure 7.18 Plan and cross sectional view of the simple side casting method illustrating the final cut removal

- It makes a firm surface where the top layer or the overburden is soft or weathered

- Where the dump height is sufficient the dragline is standing a bit closer to the waste side of the pit (if the highwall slope is less than 90°).

Figure 7.20 shows the method. The bench is usually cut one block and one strip ahead of the operating cut by chopdown. This is done before the key cut and it creates large swing angles if the chopdown material is used for the dragline pad extension.
Figure 7.19 Typical dragline moves used in the simple side casting stripping method.

The term ‘chopdown’ refers to the dragline operation where material is dug at or above the elevation of the dragline bench floor. The action is a ‘chop’ in as much as the bucket teeth are dropped into the material thus giving bucket penetration. The bucket is then dragged down and towards the dragline. This operation is best performed with an arched bucket with a strong front ring, which is necessary for bucket weight and, in particular, gives the bucket the required balance.

The chopdown efficiency varies widely. In general the actual production loss is found to be in the 25% - 70% range. This loss results from longer bucket loading times, decreased bucket fill factors,
as well as increased dragline moves. One factor frequently not considered is the increased bucket wear, ground engaging tool wear and maintenance time and cost on the machine resulting from this type of operation. Dozer time is increased by the necessity of pushing back material and bench relevelling required by the bucket’s inability to cleanly fill itself when being dragged towards the machine. Initially the dragline can walk back from this spill material, but eventually dozers must be used to relevel the bench.

Figure 7.20 Plan and cross sectional view of the advance bench being produced by the dragline chopdown method
The effect of this inefficiency on stripping costs can be substantial. For instance, if the chopdown efficiency is found to be 50% then the result is equivalent to a stripping cost increase for the chopdown material of at least 80%, and probably more when increased maintenance costs are included. For most stripping methods this chopdown material may be only 10 - 20% of the total stripping. This means, for an operation which has a 50% chopdown efficiency the overall increase in stripping cost, as compared with a 100% conventional operation, would be in the 8 - 16% range. When the decision to use chopdown as an integral part of the stripping method is being made, it is very important to use both the actual production rate and cost associated with chopdown when comparing it with other alternative methods.

Some important operation disadvantages are clearly observed with this method, as shown in Table 7.3.

Table 7.3 Disadvantages associated with the advance bench chopdown method

<table>
<thead>
<tr>
<th>Disadvantages</th>
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<tbody>
<tr>
<td>Low dragline productivity</td>
</tr>
<tr>
<td>Heavy wear of the dragline bucket and teeth (especially during formation of</td>
</tr>
<tr>
<td>the small advance bench highwall)</td>
</tr>
<tr>
<td>Bulldozers must be used to clean in front of the dragline, because during</td>
</tr>
<tr>
<td>chopdown with a dragline material is dragged nearer the machine – but does</td>
</tr>
<tr>
<td>not load easily into the bucket.</td>
</tr>
</tbody>
</table>

7.4.4 Bench Extension

This variation of the casting method includes the concept of re-handling or double handling of the waste. When the reach of the dragline is insufficient to dump the waste at the correct position (i.e. The peak position of the spoil pile as calculated or determined from a range diagram), the excavation pad needs to be extended into the pit and the dragline then moves towards the spoils and sits on top of the bench extension pad to reach the correct dump distance. Material for the bench extension is generated from either the key cut, during normal casting or chopdown. The extension is usually prepared one block ahead of the dragline by casting. A bulldozer is used to prepare the bench extension for the dragline to walk on. Figure 7.21 and Figure 7.22 show this method.
In general 65 - 85% of the material that is used to make the bench extension is later rehandled by moving it to the waste side. The exact amount of rehandling can easily be calculated from range diagrams or by simple geometric calculation. The rehandling volume must then be added to the total bench cubic meters of the overburden that is handled, thereby determining the actual capacity and productivity of the dragline.
Figure 7.22 Plan and cross sectional view of a dragline utilizing an extended bench to perform the final cut

REHANDLE

Rehandling is the double handling of waste, to build an extended bench, or to be able to dump all the material on the waste dump without any loss of coal. Two types of rehandle are often defined;

- **Direct**: This type of rehandle is usually incurred as a direct result of the chosen stripping sequence for any particular pit geometry. It normally includes the volume of material required to construct an extended bench and the dragline pad.
- **Indirect**: This is the unavoidable rehandle incurred at the ends of cuts and at positions where haulage ramps are bridged. In many instances the exact quantities depend on each specific cut layout. At these positions rehandle will be incurred and the dragline’s effective productivity must be derated accordingly. Obviously in both situations deeper overburden would imply more rehandle.

Rehandle can be expressed as a part of the total (or prime) waste material volume which must be excavated.

\[
\text{Rehandle(\%)} = \frac{\text{BCMs of extension}}{\text{Prime BCMs}} .100
\]

In this equation, rehandling is expressed in terms of BCM or Prime in-situ cubic meters – that means the volume waste before it is blasted. In some cases, reference can be made to loose (swelled) cubic meters (LCMs) since this is the state of the overburden when it is actually excavated.

Total Cubic Metres (TCM) or Effective BCMs is also used in the context of rehandle and is defined as the sum of BCMs and the unswollen rehandle BCMs and is normally expressed in cubic metres.

A productivity factor is also used, expressed in terms of the prime volume to be moved, such that;

\[
\text{Prime productivity factor} = \frac{\text{Prime BCMs}}{\text{TCMs}.\text{Prime BCMs}}
\]

If 30% rehandle is incurred, this would be equivalent to a Prime Productivity Factor of 0.769, or on average, 76.9 percent of dragline swings are moving prime overburden.

It is important to determine to what rehandle refers when calculating dragline working methods. Also – note that when extending a bench using blasted material – this is actually swelled BCM’s and has to be unswelled if rehandle is calculated in total BCM’s and not LCM’s. If cast blasting is used (or even normal blasting techniques with narrow pit widths), a ‘bonus blast-over’ can be encountered, described as the volume of material moved to its final position by explosive energy and not handled by the dragline. This volume is often expressed as a percentage of BCMs and it may also form a fraction of the rehandle bench volume.
7.5 Managing and Optimising Dragline Productivity

Draglines have been shown to be flexible mining tools with excellent cost efficiency across a broad range of application variables, but subject to an understanding of the effect on productivity of these variables. Some of the key variables are introduced and discussed in the following sections.

7.5.1 Dragline Utility and Operational Efficiency

An important method to increase production is a detailed analysis of dragline utility. Of all the cyclic loading machines discussed, the dragline is certainly the most comprehensively instrumented piece of equipment; not only from an electronic diagnostics point of view, but also in terms of its operational activity. The reason for this is both related to the cost of the machine, its pivotal role in overburden removal and the volumes excavated, where relatively small improvements to efficiency or cycle management pay dividends in terms of extra tonnages stripped.

The primary function of such a control system, apart from the progress or performance measurement, must be to identify possible opportunities where the process may be improved. Any control system consists of the following aspects:

- standards
- measurement
- evaluation
- reporting of results
- corrective action or implementation of improvements

STANDARDS

The standards for a dragline operation, where continuous improvement is the main goal, are derived from first principles. The digging method the dragline uses is normally evaluated from software that calculates a number of key operating performance parameters.

The data recorded can be used as a basis for universal productivity and Key Performance Indicator (KPI) comparisons. Such comparisons could cover, for instance;
- Annual output vs. rated suspended load (RSL). This is a measure of the relative output of the dragline against the RSL of the machine. Note that the RSL is the load designated by the manufacturer and the evaluation is based on the extent to which the ‘purchased’ digging capacity was used.

- Dig rate vs. RSL: Dig rate can be specified as either an equivalent annual output or an equivalent annual dig rate. The dig rate reduces the productivity back to a “per dig hour” comparison to assess how productive the machine was when digging.

- Fill Time
- Swing Angle
- Swing Out Time
- Swing Return Time
- Cycle Time
- Spot Time: The difference between the cycle time measured and the sum of its components. In reality it is a measure of the efficiency of the operators. That is, the more efficient the operation, the less anomalous events occur and the lower the spot time.

- Bucket Efficiency Ratio: Bucket payload / Bucket capacity
- Payload Efficiency Ratio: Weight of spoil / Weight of bucket steel

MEASUREMENT
The measurement of the operation is a critical part of the whole operation. In the case of draglines, the measurement base is volume and time. Inaccurate measurements of any of these invalidate any management decisions. Whilst a continuous monitoring system will provide 90% of the data required, as a check, survey measurement of the dragline volumes remains the most reliable method to determine the dragline’s performance.

The process starts with the measurement of the blast profiles to determine the in situ material remaining after blasting. The dragline will perform certain functions to enable it to eventually spoil all the overburden, such as extending the pad or elevating the pad. The
survey must measure all the volumes to be able to classify the material as prime or rehandle.

The second component of measurement is available digging time. The starting point for utility management is the analysis of operational time records over extended periods of time. The operating time and delay times should be broken down into specific categories so that the total time usage can be recorded. Table 7.4 presents a typical summary of time usage.

<table>
<thead>
<tr>
<th>TABLE 7.4 Typical dragline operational recording and time analysis</th>
</tr>
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<tbody>
<tr>
<td><strong>TOTAL HOURS</strong></td>
</tr>
<tr>
<td><strong>TOTAL HOURS SCHEDULED</strong></td>
</tr>
<tr>
<td><strong>STAND DOWN HOURS</strong></td>
</tr>
<tr>
<td>No Power</td>
</tr>
<tr>
<td>Change Cable</td>
</tr>
<tr>
<td>Wait Ancillary Equipment</td>
</tr>
<tr>
<td>Service</td>
</tr>
<tr>
<td>Bucket</td>
</tr>
<tr>
<td>Ropes</td>
</tr>
<tr>
<td>Deadheading</td>
</tr>
<tr>
<td>Pad Soft or Preparation</td>
</tr>
<tr>
<td>Blasting</td>
</tr>
<tr>
<td>Clean coal</td>
</tr>
<tr>
<td>Clean face</td>
</tr>
<tr>
<td><strong>TOTAL STAND DOWN HOURS</strong></td>
</tr>
<tr>
<td><strong>SCHEDULED MAINTENANCE</strong></td>
</tr>
<tr>
<td>Mechanical</td>
</tr>
<tr>
<td>Electrical</td>
</tr>
<tr>
<td><strong>TOTAL SCHEDULED MAINTENANCE</strong></td>
</tr>
<tr>
<td><strong>UNSCHEDULED MAINTENANCE</strong></td>
</tr>
<tr>
<td>Mechanical</td>
</tr>
<tr>
<td>Electrical</td>
</tr>
<tr>
<td><strong>TOTAL UNSCHEDULED MAINTENANCE</strong></td>
</tr>
<tr>
<td><strong>ELEC/MECH AVAILABILITY</strong></td>
</tr>
<tr>
<td><strong>UTILISATION OF AVAILABLE TIME</strong></td>
</tr>
</tbody>
</table>
EVALUATION OF MEASUREMENT
The results achieved from the measurements must be evaluated against planning parameters and deviations from the planned figures must be analysed to determine the causes of the deviations. These causes can then be evaluated by indicating the areas in the operation which cause large delays, these being the areas requiring close attention for possible improvements. Bear in mind that whilst most modern draglines have an on board recording system to gather measurement data –some operator input is required and ultimately the accuracy of especially the ‘stand-down’ data is dependant on how the operator books this time.

Where performance exceeds planned parameters, evaluations should also be undertaken to establish the reasons for this performance. These may be applied to the other draglines or incorporated in to future planning parameters to improve on dragline performance.

REPORTING OF RESULTS
Since the machine is a high-volume excavator, the results of the dragline performance should be reported on a daily basis, with month to date performance and a total month estimate trends. If used, the report can be updated weekly with the accurately surveyed volumes to reconcile both systems.

CORRECTIVE ACTIONS
Allied to dragline performance monitoring is operators performance monitoring. By recording some the following operational critical alarms and messages, training deficiencies to be recognised and rectified or the working method modified to reduce their incidence;

- Drag stalls: monitors and records drag stalls in excess of 3 seconds - reducing motor/generator maintenance.
- Hoist stalls: monitors and records hoist stalls during long swings angles - reducing motor/generator maintenance.
- Hoist overloads: monitors and records overloading the bucket by 25% - overloading can be costly in terms of maintenance and light loading is costly in terms of lost productivity.
- Hoist slack rope: monitors and records excessive hoist slack rope at the hoist drum, to prevent over-wrap/over spooling - reducing rope replacement and lost productivity due to down time.
- Drag slack rope: monitors and records excessive drag slack rope at the drag drum, to prevent over-wrap/over spooling - reducing rope replacement and lost productivity due to down time.

- Tight lining: monitors and records the bucket in this curve during the hoist to dump cycle - reducing excessive boom stresses.

- Pre-tight line: monitors and records when the loaded bucket is hoisted within a predetermined best path hoist to dump - tight-lining slows down the hoist speed, increases cycle time resulting in lost productivity.

- Over swing left or right: monitors and records the boom point sheave tilt angle during the dig or dump mode and excessive bucket lag or lead during the swinging. This bad operating technique places enormous stresses on the boom and will result in costly boom repairs and lost productivity due to down time.

- Fly dumps: monitors and records the position of the bucket in excess past the dig-dump radius during the dumping mode, resulting in roller circle, rails and tub repairs, lost productivity due to walking and waiting on the dozer whilst re-leveling the dragline pad.

- Dragline not level: monitors and records the level of the machine during operation.

- Boom stress: monitors and records excessive shocks to the boom from the bucket via the hoist ropes - this reduces boom damage and lost productivity due to down time.

- Control lever jockeying: monitors and records excessive hoist control movement – also reduces boom stresses, bucket rigging damage and lost productivity due to down time.

Additionally, these systems record the cycle time components reviewed previously. Examination of this data, together with the utility data may be useful in highlighting downtime that is related to a mining method which, whilst appearing as an attractive working method on paper, show up as being expensive in terms of down-time, walking, deadheading, repositioning, excessive swing angles, etc.
7.5.2 Dragline Productivity in Deep Overburden

Most of the critical performance factors of dragline productivity are directly related to increasing overburden depth. At the start of a strip mine operation economic factors generally dictate that the position of the initial cut be situated in the shallower reserve areas. This facilitates easier blasting of the box cut burden, lower volumes of spoil placed on virgin ground and a relatively rapid rate of coal exposure. Should the mine be planned in this way it would generally mean that in later years of the mine’s productive life the stripping operation will gradually progress into deeper deposits where the factors that influence dragline performance will change, to a point where the mining method may need to be modified or even totally revised. It is therefore essential that mine planners identify, at the initial planning stage, factors that will necessitate changes in the deeper burden areas and that they understand the implications that these factors could have on dragline productivity and costs.

In multi-strip operations, the deep stripping of overburden by a dragline in one particular area can be largely offset by the stripping of shallower overburden, by a second dragline in another area. Ideally this balancing of burden depths can facilitate constant rates of coal exposure but it requires accurate mine planning and scheduling as well as strict control and monitoring. Although in this situation acceptable average overburden stripping rates may well be maintained, the fundamental fact that draglines in deep areas have lower effective productivity rates (for a particular stripping sequence) is not addressed. To address the problem, a number of alternatives need to be reviewed, including:

- Shovel and truck combination in pre-stripping
- Loader/BWE/conveyor systems in pre-stripping
- Cast blasting or doze over
- Spoil-side stripping
- Bucket design

Some of the concepts have been discussed in other Modules, but when evaluating any of the above methods or combinations of methods each entity should not be viewed in isolation, but as a integral part of the total mining system, from in-situ overburden position through to the final rehabilitation of the spoils. Each of the systems is briefly discussed in this context below.
PRE-STRIPPING

Pre-stripping involves the partial removal of overburden in advance of the dragline, haulage around the end of the pit and dumping on top of spoil several rows behind the dragline. The type of pre-stripping equipment selected must be capable of achieving this and is therefore the most critical step in the evaluation. This system must be compatible with the dragline operation, cost effective and capable of doing the work at the required rate.

The truck and shovel system has numerous advantages, the main one being that it is a proven system and is flexible, as evidenced by the wide range of mining applications in which it is used. Critical issues associated with this type of operation need, however, to be carefully analysed. These include volumes of overburden required to be moved, equipment selection, haul distances and the relatively high operating costs. Haul distances specifically need to be carefully monitored so as to ensure they remain as short and as constant as possible. This is usually in conflict with the principle of long cut length required for effective dragline operations. Long cuts may have to be bridged which in turn could increase dragline rehandle. Truck and shovel pre-stripping has certain advantages also, as top soil must, in any case, be hauled to levelled spoils in the rehabilitation process. When additional overburden is trucked to the spoil side this spoil can be evenly distributed and could assist in the spoils levelling process. This operation therefore cannot be economically evaluated in isolation as the spoil levelling and top soil placing advantage must be included.

A second alternative which could be considered is a system using cross pit or around the pit conveyors and stackers loaded by shovels or BWEs. The disadvantages of this option are however, their high capital cost, lack of operational flexibility and their reliability in difficult or unpredictable material. This last disadvantage is of particular concern for the harsher South African overburden conditions.

BLASTING TECHNIQUES

Correct blasting of burden is extremely critical to all dragline operations and all blast designs must take cognisance of fragmentation and burden heave, especially in deep burden areas. Excess burden movement, when not planned, could result in additional dragline rehandle. In the use of explosive energy to move burden – cast-blasting - the bonus blast-over in some instances can offset the increase in rehandle incurred in deeper areas, but additional bulldozer pad preparation is, however, often necessary and key cut excavation is often problematic. Care should be taken so as not to drop the dragline pad elevation to the extent where dump height difficulties can be experienced. Also, the potential coal seam damage
(and hence coal losses) requires constant blast design control and monitoring.

To evaluate the potential of changes in blast design to deliver cost-effective results, it is necessary to determine the value of increased dragline productivity or reduced costs. Using the cost structure example in an earlier Module, assume total waste stripping costs, representing 34% of the operating cost comprise stripping costs of 30% and overburden preparation (drill and blast) represents 4%. Assume that by improving overburden blasting it is possible to increase dragline production by 5%. Of dragline operating costs only power is approximately directly proportional to the production rate, and the power cost represents approximately 40% of the total. Other costs remain the same with the exception of maintenance, which may even decrease if improved ground engaging tool- and rope-life with decreased bucket wear are experienced. In all certainty, there would be a 3% reduction in dragline unit costs.

To evaluate the possible cost savings – or value of overburden blasting improvements, assume that the mine operating cost is R30/t coal. The total stripping cost would be R10,20 per ton of which the blasting cost R1,20/t. A saving of 3% on the stripping cost would then represent R0,27/t. It was originally assumed that this cost decrease was solely due to improved overburden blasting. Thus the overburden blasting cost could be increased by up to R0,27/t to achieve this result.

Basically this means a more efficient operation is achieved if, when increasing the overburden blasting cost by 22%, an increase of 5% or greater is experienced in dragline production. Furthermore, when it is considered that the explosives cost may represent about 40% of the overburden blasting cost, then it may well be possible to increase the explosive consumption by up to 50% for this same 5% production increase.

Cast blasting of overburden is often the least expensive option to follow if the coal exposure rate is to be increased. This technique is carried out in conjunction with a change in the dragline digging method, since cast blasting in most cases requires the chopping of material by the dragline. In some cases the overburden moved by the explosive energy may be as high as 50% of the total insitu volume. The cycle times in certain digging positions may be longer, resulting in a slightly longer overall cycle time than under normal digging conditions. This is due to the higher hoisting distance, larger swing angles and more difficult digging conditions along the highwalls. However, when considering the total operation, the coal exposure rate will be much higher due to less rehandle and less insitu material that must be excavated to expose the coal.
DOZE OVER
To reduce the volume of material that the dragline must excavate to expose the coal even further, bulldozers may be used to push some material past the required limit to expose coal. In most dragline operations a dozer is allocated to the machine for pad preparation, cable handling occasionally and extensione pushover purposes. The machine therefore has a utilisation as low as 30%. Initially the dragline pad dozer can be used to push material over to assist with the coal exposure. As the volume of material increases to maintain the coal exposure rate, more dozers may have to be added to assist the pad dozer to keep ahead of the dragline. Coal exposure rates may increase up to 30% with a dozer assisted dragline operation. The cost of moving the material by dozer may range from R2,00 to R3.50 per BCM depending on the type of material being dozed and it may be cheaper than the average cost of overburden TCMs moved by the dragline if large bench extensions are required.

SPOIL-SIDE DRAGLINE PASS
This stripping sequence implies that burden is removed in two or more stages by either one dragline executing two (or more) passes or two drag lines operating in tandem. During the initial pass the upper portion of the burden is placed in the void in a position where it is used as the dragline operating pad for the second pass. In a single seam operation the various operating levels can be determined according to the pit geometry and physical dragline constraints as shown earlier. In a multi seam operation however, these levels are normally dictated by the position of the coal seams.

The advantage of the spoil-side sequence is the reduction in direct rehandle volumes which would have been incurred using only a highwall pass. Consideration must be given to spoil side ramp positions and frequency, highwall to spoil-side bridging, end of cut configuration, length of cut, cut widths, spoil stability, power reticulation and the impact on coal exposure rates.

BUCKET DESIGN
One way to increase dragline productivity is by increasing the size of the dragline bucket. To ensure the larger bucket does not increase the maximum allowable load that can be suspended from the dragline’s boom, new lightweight steel with some innovative engineering design is required. The dragline design dictates certain capabilities in terms of material handling. One of the major aspects of the design is the rated suspended load (RSL) capability of the machine. This, with the material type to be excavated determines the
bucket size of the dragline. The bucket size may be increased by reducing the actual bucket weight or increasing the material load per bucket. The increased material load results in more being excavated per time period, thereby increasing productivity in deep overburden areas.

One approach to bucket design is a heavy duty front ring, as this is the area that endures the most stresses and designed to last for, typically, five years before major repair work or replacement takes place. The basket section behind the ring retains the material once dug and is not exposed to such high stresses and wear as the front ring.

It can therefore be made of lighter steel, saving on the weight of the bucket which can be converted into carrying capacity. The lightweight basket is replaced once per annum with a new basket and the traditional approach of repairing and rebuilding buckets is replaced with one of manufacturing new buckets or baskets. It also eliminates the problem of adding excessive amounts of wear plate, reducing capacity and increasing weight.

Dragline buckets can be classified into the following categories:

- light weight (890—950 kg/m$^3$ of bucket size)
- medium weight (1009—1068 kg/m$^3$ of bucket size)
- heavy duty (1068—1247 kg/m$^3$ of bucket size)
- severe duty (1086—1425 kg/m$^3$ of bucket size).

The current trend is to reduce the weight of the bucket, thereby increasing the amount of material that can be carried in the bucket. For typical South African overburdens, this is somewhat problematic due to the hard, blocky nature of the material. For a successful increase in productivity through bucket weight reduction, fit-for-purpose bucket designs are required. The lighter the bucket, the more maintenance may be required and in some types of material a lightweight bucket may not have the desired life, negating the possible benefits that may be achieved by the larger size. Dragline bucket size becomes a trade off between maintenance intervals and the economics of repairing or replacing the bucket as well as the associated machinery. A user may decide to opt for the largest possible bucket, which will dictate the use of the thinnest possible plate for fabrication. This bucket may only last a month for a given situation, resulting in the removal and replacement or repair of the bucket. The digging time loss to replace a bucket as well as the repair or replacement cost must be taken into consideration when choosing the bucket size and design.
7.6 Dragline Relocation

Draglines, due to their size and cost, are purchased and erected for the life of the mine and are usually scrapped or mothballed on closure of the mine. The life of the first generation of strip coal mines which exploited the shallower reserves in South Africa are now coming to an end and it is possible to relocate draglines from these mines if they still have a serviceable life. In addition to the above, with the task of relocating draglines becoming more feasible, the possibility of relocating draglines to exploit smaller and previously uneconomical resource blocks is now becoming economically viable.

7.6.1 Transportation Options

There are various methods of relocating draglines, namely by walking them, dismantling and transporting the components via road or transporting them on multiple axle self-propelled trailers or crawlers.

Walking: In walking mode the machine moves in reverse. A dragline can walk only slowly because of the re-routing of the power supply cables and skid mounted transformers supplying the power to the dragline. An alternative source of power for walking draglines over long distances is a transportable power supply from diesel powered generators mounted on trailers, consisting of diesel powered generators varying in size from 500 kVa to 1.25MVa. The benefits of relocating draglines by walking using transportable power are:

- Temporary overhead power lines do not need to be constructed along the route
- The risks associated with HT switching when moving the dragline cables and transformer skids are avoided
- Long relocations are more cost effective using transportable power since the dragline walks faster, getting to its destination sooner.

The disadvantages are:

- The high risk of mechanical breakdown on route
- The additional wear and tear on the machine, particularly the walking mechanism
The perceived negative environmental impacts arising from constructing a 32 m wide relocation route and the high cost of rehabilitating the route to restore it to its original state.

**Multiple axle, self-propelled trailers:** The option exists to transport a dragline on multiple axle, self-propelled trailers. The disadvantage of this option is the high specification the constructed roadway must conform to, the high cost of rehabilitating such a route and the high cost of delay in terms of trailer hire if there are unforeseen delays at the obstacles that require crossing along the route. However, the risk of mechanical breakdown is low and the time required to relocate the machine is much shorter than the walking options.

**Crawler transporters:** Use three crawler transporter units to support a frame that the dragline sits on. This method is used frequently in Australia for relocating draglines. The advantage of this method is that less earthworks and the associated rehabilitation needs to be carried out in preparing the roadway. As with the other trailer options there is the trailer hire risk.

**Dismantling the machine:** The cost of relocating a dragline by dismantling transporting the sections and re-ereciting them on site is high. This option is only viable if environmental factors or obstacles en route preclude the dragline being moved as whole. The duration required to carry out the work depends on the degree to which the machine is dismantled, but for on-road transport, the process could take up to 18 months.

### 7.6.2 Considerations for Walking a Dragline

**GEOTECHNICAL INVESTIGATIONS AND DESIGN**

The primary considerations are the terrain, and obstacles to be traversed and the time of the year the relocation route is to be constructed and used. The relocation route must be designed to prevent the following:

- Collapse below the dragline walkway due to previously mined areas – pillar collapse or migration of voids
- Walkway bearing capacity failure below a shoe
- Excessive differential settlement of the shoes or tub on the walkway
- The formation of a roll of clay or wave of sand which accumulates below the tub as the dragline walks.

Once the preferred relocation route alignment has been selected, a detailed investigation into buried services and underground stability must be carried out, together with the excavation of borrow-pits and a centre-line profile of the surface stratigraphy and material types compiled before a suitable structural design and wearing course is specified.

**ENVIRONMENTAL CONSIDERATIONS**

An Environmental Impact Assessment (EIA) and/or Environmental Management Programme Report, together with the relevant water use licenses have to be prepared and submitted. The EIA process is followed which includes; scoping, baseline environment studies, considering alternatives, determination of the route, impact assessment, mitigation measures and compilation of an Environmental Management Plan (EMP). The studies must address all aspects of the receiving environment, including; climate, topography, surface water resources, geology and soils, land capability, land use surface and infrastructure, fauna and flora, aquatic ecology, existing mining activities, archaeological and cultural history sites and human settlements.

The EMP, based on the mitigation measures to minimize impacts proposed for the route construction, dragline relocation and route rehabilitation provides for the inclusion of environmental specifications from the outset of the project when procuring contractors and post-rehabilitation monitoring programmes for the rivers, wetlands, natural vegetation and cultivated land.

**RIGHT OF WAY AGREEMENTS**

A typical right of way agreement should include a description of the properties, the right of way, consideration, duration, access, rehabilitation and crop loss, damage, insurance and other general clauses that make up a formal agreement. An early start in concluding these agreements is important as this activity carries a high risk in terms of delaying the project.
LEGISLATIVE REQUIREMENTS
As dragline relocation projects are of a mining nature they have to comply with the local regulatory framework. The crossing of farmers’ land and public servitudes falls within the jurisdiction of various Departments and legislation other than Mining and Minerals. Discussions with the parties involved have to be concluded to confirm which piece of legislation would be applicable under which circumstances.
# EQUIPMENT COST ESTIMATION TECHNIQUES

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**Apply, calculate or predict**
- Effect of inflation, escalation and real interest rates of
- Present and future values
- Cash flow and DCF schedules and analysis
- Discounted average cost for earthmoving
- Replacement options for equipment
- Ownership cost – annual investment or EAC method
- Discounted average cost of ownership

**Evaluate or design**
- A DCF analysis for equipment purchase or comparison
- Owning and operating costs for a piece of equipment
8.1 Introduction to Cost Estimation

In the previous Modules, much attention has been given to equipment selection and operational analysis, based on a ‘fit-for-purpose’ approach. In many instances, several similar competing systems or products can be identified and a selection decision is required, based finally on an economic analysis or comparison. An economic analysis is based on economic criteria and, in the case of whole projects, what is commonly termed an “economic analysis” normally implies estimation of capital and operating costs as well as revenue from sale of products. However how the capital is financed, or how the venture is structured is not normally considered at this stage.

In the mine planning phase, as will be seen in the next Module, the provision of reliable and realistic costs for equipment purchases is critical to the economic evaluation of a mining project. In the case of large volume, high capital, low unit excavating or transporting cost equipment, the reliability and impact of these assessments becomes all the more critical in the overall economic model and resulting mine plan. This Module introduces some of the basic terms, concepts and techniques involved. It is not intended to be a guide to costing complete in itself but rather to demonstrate some of the typical approaches that can be used.

8.1.1 Life Cycle Costing Approach and Cost Elements

When using a life cycle costing approach for equipment selection, a framework is developed in which various alternatives are compared. It is a uniform test for application, maintenance, infrastructure requirements and replacement, following the principles of asset management. The basis of the approach is to develop a value chain model that starts at the market demand, through to the life of mine planning, short-term mine planning and maintenance, together with other associated activities. Figure 8.1 illustrates the components of a typical life cycle costing approach, including;

- Acquisition cost
- Operating cost
- Maintenance cost
- Repair costs
- Modification or decommissioning (scrapping) costs
The concept of life cycle costs is the ‘systematic analytical process of evaluating various alternative courses of action with the objective of choosing the best way to employ scarce resources’. The life cycle cost model is at the heart of the cost projections used in the equipment selection model. The life cycle cost model can be compiled based on a number of different approaches; either the British Standard, BS5760 (BSJ 1983) or the SAE model (adapted) as illustrated in Figure 8.1.

Most elements of the business, or value chain, have some influence on the equipment performance, life, and the life cycle cost. Therefore, from an equipment selection point of view it is imperative that the decision is made after accounting for the different stakeholders. This is essentially why life cycle costing is so important. It ensures that a long-term focus is maintained in order to achieve the lowest possible cost through effective teamwork from the various elements constituting the value chain in a business. Life cycle costing is, however, not without its limitations. However, by using sound judgement the effect of these limitations can be reduced.

Figure 8.1 A life cycle cost model for loader selection
The costs incurred by a mining company over the equipment life cycle are;

- Acquisition or capital (or ownership) costs include such items as purchase price of equipment, administration engineering, installation and transport, training and conversion or special options. For tax purposes, these costs are not regarded as directly deductible expenses, but must be either amortised, depreciated or depleted. The equipment may have some residual value as scrap, effectively reducing the depreciation amount, but there are additional costs to consider such as decommissioning, modification and site costs also.

- Operating costs (or expenses) are for such items as labour, utilities, consumables and other supplies, and are deductible, in total, in the year of occurrence. Costs are also treated as direct or indirect, with fixed or variable cost components, depending on their relationship to coal production. It is usual to consider maintenance and repair costs in this category also.

It is the estimation and analyses of these costs that form the basis of economic analyses of equipment selection options and forms input to the mine planning exercise.

8.1.2 Definitions and Techniques

In the planning, scheduling and controlling (not to mention budgeting) of almost any mining project, all significant aspects or criteria of that project are generally analysed, evaluated and compared with some other criterion or alternative. Whatever can be meaningfully quantified, should be, since most decision-makers find it easier to compare numbers (figures) than other qualities. In order to quantify and compare such criteria, a common denominator for most purposes is a monetary (Rand) value. For this reason, and for the purpose of this Module, most of the significant criteria or attributes of a project are reduced to a monetary value whenever possible.

Obviously, this monetary value will have different “values” over time - every reasonably educated person knows that a Rand received “tomorrow” is not equivalent to a Rand in hand “today”.

The two key concepts making up the time value of money are ‘future value’ and ‘present value’, each with their common derivatives. Future-value calculations are needed to evaluate future amounts resulting from current investments in some benefit-bearing (interest)
medium. They are also useful in determining interest or growth rates of future money streams. Present-value calculations are inversely related to future value. They are used to assess the current value of a project or an investment with all of the future costs and benefits expected to result from certain actions today.

**BASIC DEFINITIONS**

**Inflation**  Inflation is the decrease in purchasing power of a currency with time. It refers to the change in price of a “basket” or group of items over time. The most common measure is the Consumer Price Index (CPI) which is a weighted index of prices of common commodities purchased. Inflation indices are also published for many other baskets of commodities. Inflation depreciates the value of money over time, since in an inflationary environment the same amount of cash will not purchase the same amount of goods in the future.

The CPI is indexed at 100 at a specific time and expressed relative to this base in future periods. Equivalent annual inflation rates can be calculated from monthly CPI’s. For example, a change in CPI’s from 112.8 to 113.6 respectively indicates an increase of 0.8 on 112.8, or 0.71% for one month. The annual rate is thus;

\[ i = (1 + 0.0071)^{12} - 1 = 8.8\% \]

**Escalation**  When the rate of inflation quoted for categories of project cost differs from that obtained using the CPI, that difference is termed escalation. For example, if the general inflation rate is 8.8% and an escalation rate of 5% is applicable to capital expenditures (CAPEX), the specific inflation rate to be applied to CAPEX items is thus 1.088 x 1.05 = 1.1424 or 14.24%.

**Interest**  It is the price paid for the use of (someone else’s) money. Economic theory considers two components; an “originary” component which represents the difference between the present values of present and future goods, and an “entrepreneurial” component which represents the uncertainty element as to whether the money is likely to be repaid or not. Hence a Rand today is (likely to be) more valuable than a Rand received in a year’s time.

**Real Interest Rate**  Reflects the price paid for use of money over and above the amount paid just to keep up to inflation. Obtained by adjusting the nominal interest rate (quoted interest rate) for the rate of inflation. For example, if the nominal interest rate on a loan is 15% and inflation is 8.8%, then the real interest rate is;
\[
\frac{1.15}{1.088} = 1.057
\]

or 5.7%. As an approximation the nominal interest rate minus inflation equals the real interest rate.

**TIME-VALUE RELATIONSHIPS**

Time-value relationships are used to provide a monetary connection between cash flow values estimated at different points in time. Values must either be brought to a constant point in time (usually, *today’s* money terms or ‘time zero’), or spread evenly over a common period of time.

**Cash Flow**  The difference between cash flowing-in to a business or operation (receipts, or revenue) and cash flowing-out (actual cash costs, including tax). Cash flows are normally grouped by time period in which they are expected to appear. Components of the cash flow may differ, but will still comply with the above flows. Cash flows generated when evaluating projects are generally expressed in today’s (or constant) money values and related to a specific date, usually ‘now’ which is taken to mean the end of year 0 of the project.

**Current (Nominal) and Constant (Real) Money Terms**  A cash flow in current (nominal) money terms, shows the actual amount of money spent (or earned) in a specific period where the influence of inflation has been taken into account. The amounts of money reported in current money terms are generally larger than those in constant (real) terms as the value of money has decreased. To be comparable, these numbers must be discounted by the inflation rate to give a value at a specific time. When the impact of inflation is considered, the decision on the feasibility of a project may differ from that made on current money basis.

The concepts of present value (PV) and future value (FV) are used to estimate the effect of time on a value or item.

**Future Value**  Determines the value of present value at some point \( n \) years into the future, as a function of inflation \( i \);

\[
FV = PV(1+i)^n
\]

Determination of FV is also referred to as compounding, where \((1+i)^n\) is the compounding factor.
**Present Value** Determines the value of future money \( n \) years into the future in today’s terms, as a function of inflation \( i \);

\[
P V = \frac{F V}{(1 + i)^n}
\]

Determination of PV is also referred to as discounting, where \((1+i)^n\) is the discounting factor.

**Annuity Factor** A series of fixed payments (or equal annual cash flows) over a period of time are known as annuities. The value of an annuity \((A)\) comprising a number of fixed payments \((P)\) is given by;

\[
A = P \left[ \frac{1 - (1 + i)^n}{i} \right]
\]

An annuity factor is thus;

\[
\left[ \frac{1 - (1 + i)^n}{i} \right]
\]

**Net Present Value** When a series of annual net cash-flows (the difference between cash spent and/or received at various times in the future) is discounted to the present value (in a tabulation called a ‘discounted cash flow’ (DCF) – discounting is used synonymously with net present value). The sum total of these discounted amounts is the Net Present Value (NPV). Developing a DCF involves estimating a project’s cash flows, discounting these cash flows at the company’s required rate of return (cost of capital) and subtracting the PV cost of the investment from the sum of cashflow PVs. For revenue generating investments, the investment with the highest positive NPV is the preferred option, whilst for non-revenue generating applications, the investment with the lowest negative NPV will be preferred.

NPV is determined by;

\[
NPV = \sum_{t=0}^{t=n} \left[ \frac{CF_t}{(1 + i)^t} \right] - PV_{investment}
\]

where;

\[
i = \text{cost of capital} \\
CF_t = \text{cash flow in year } t,
\]
As an example, consider the purchase of a piece of equipment. Equipment A costs R10M today (Year 0) and generates the annual net cash flows shown in Table 8.1 for operating years 1-5. Equipment B costs the same, but generates slightly different annual net cash flows, as also shown in Table 8.1. If a return on investment of 10% is specified as a criteria to cover cost of borrowing and desired profit, then an investment in equipment A would be favourable for the cash-flows shown in Table 8.1.

Table 8.1 DCF analysis of investment

<table>
<thead>
<tr>
<th>Year</th>
<th>Equipment A</th>
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<th>Equipment B</th>
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<tbody>
<tr>
<td></td>
<td>Net cash flow (CF) (R.M)</td>
<td>PV (R.M) of annual cash flow</td>
<td>Net cash flow (CF) (R.M)</td>
<td>PV (R.M) of annual cash flow</td>
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<td>-10.0000</td>
<td>-10.00</td>
<td>-10.0000</td>
</tr>
<tr>
<td>1</td>
<td>+2.00</td>
<td>1.8182</td>
<td>0.50</td>
<td>0.4545</td>
</tr>
<tr>
<td>2</td>
<td>+2.50</td>
<td>2.0661</td>
<td>1.00</td>
<td>0.8264</td>
</tr>
<tr>
<td>3</td>
<td>+3.50</td>
<td>2.6296</td>
<td>3.50</td>
<td>2.6296</td>
</tr>
<tr>
<td>4</td>
<td>+3.50</td>
<td>2.3905</td>
<td>5.00</td>
<td>3.4151</td>
</tr>
<tr>
<td>5</td>
<td>+3.50</td>
<td>2.1732</td>
<td>5.00</td>
<td>3.1046</td>
</tr>
<tr>
<td>NPV</td>
<td>+1.0777</td>
<td>NPV</td>
<td>+0.4303</td>
<td></td>
</tr>
</tbody>
</table>

From the above example, although both pieces of equipment have the same total annual net cashflows (R15M), and both cost the same to purchase (R10M), it is seen that the best investment decision, based on NPV, is to purchase Equipment A, due to the higher NPV. This results from the higher annual net cashflows that occur earlier in the life of the equipment than those associated with Equipment B.

The financial evaluation of various investment possibilities, as illustrated in simple terms above, is an integral part of the capital budgeting process. It is necessary to establish, firstly, whether
proposed investment projects are financially viable in terms of capital budgeting criteria and, secondly, whether it merits inclusion in the capital budget in terms of financial strategy.

The decision criteria for using the NPV as a decision tool are the following;

- Accept all independent projects with a positive NPV (NPV > 0).
- Reject all independent projects with a negative NPV (NPV < 0).
- Projects with an NPV = 0 do not contribute to economic value and are usually rejected.

Note that the NPV is an absolute amount of money. How is the NPV interpreted as a criterion for decisions on investment possibilities? In short, what does the NPV mean in this context? For a project with NPV = 0, it means that the given cash flow of the project, discounted at the cost of capital, is exactly enough to redeem or recover the total investment amount in addition to the financing cost. A positive NPV (NPV > 0) means that the initial investment amount and the financing cost (as well as additional value) are obtained from the cash flows of the project.

A negative NPV (NPV < 0) implies that the cash flows of a project, discounted at a predetermined discount rate or cost of capital, are insufficient to redeem the initial investment and the financing cost involved.

The net present value method is an excellent capital budgeting technique as the result, the NPV, is the absolute amount by which the value of the undertaking is increased or decreased by the project. Other decision criteria used to evaluate the financial viability of projects include also;

- the pay-back period (n years)
- the internal rate of return (IRR)

Together with numerous other measures not discussed here, such as the profitability index and the equivalent annual cost.

However, note that when applying the different evaluation techniques to rank projects in order of preference, conflict may arise when ranking mutually exclusive projects. The conflict is due to the inherent assumptions and differences between the various techniques. When
one of the following conditions apply, particular care should be taken when ranking projects:

- Differences in the time pattern of cash flows
- Differences in the life-span of projects
- Differences in the initial investment (capital outlay) required for the different projects

**Payback Period** The time required \((n \text{ years})\) to payback a sum of money, including the loan sum and the interest charged. Using the annuity factor equation and assuming R10M is borrowed today subject to an annual repayment of R1.5M and an inflation rate of 10%, the payback period can be found iteratively such that \((\text{NPV} = 0)\) as given in Table 8.2.

<table>
<thead>
<tr>
<th>Payback period ((n))</th>
<th>NPV</th>
</tr>
</thead>
<tbody>
<tr>
<td>4.0</td>
<td>-1.830</td>
</tr>
<tr>
<td>6.0</td>
<td>-0.645</td>
</tr>
<tr>
<td>6.5</td>
<td>-0.382</td>
</tr>
<tr>
<td>7.0</td>
<td>-0.132</td>
</tr>
<tr>
<td>7.5</td>
<td>0.107</td>
</tr>
<tr>
<td>8.0</td>
<td>0.335</td>
</tr>
<tr>
<td>7.3 (Solution)</td>
<td>0.000</td>
</tr>
</tbody>
</table>

For the two pieces of equipment evaluated in Table 8.1, by using the NPV equation to solve for \(n \text{ years}\), payback periods for equipment A and B are 4.23yrs and 4.68yrs respectively. Thus equipment A pays for itself sooner than equipment B.
**Internal Rate of Return**

This is the interest rate (or return on an investment ROI) which makes the future worth's equal and is determined from the NPV equation, in this case solving for \( i \). Using the previous example, but in this case investing R10M on a piece of equipment that generates R1,75M profit after tax each year over 10 years, the rate of return is found by setting the NPV of payments and investment equal to zero at time \( n = 0 \);

\[
IRR = \sum_{t=0}^{t=n} \left[ \frac{CF_t}{(1 + i)^t} \right] = 0
\]

or for this example;

\[
NPV = 0 = -10 + 1.75 \left[ \frac{(1 + 0.1)^{10} - 1}{i(1 + i)^{10}} \right]
\]

This gives a solution of \( i \) or ROI = 15%.

Alternatively, using the previous net cash flows in Table 8.1, a ROI can be found iteratively when the NPV = 0. In this case the ROI is 13.74% for equipment A, representing the return on the R10M invested to generate the R11.0777M PV cash flows. Equipment B has a ROI of only 11.24%. Table 8.3 summarises the calculation.

The decision criteria for the IRR as a decision tool are the following:

- All independent projects of which the IRR is higher than the cost of capital are acceptable. The project will, in addition to the recovery of the investment and financing costs, offer an additional return. This corresponds to a NPV > 0.

- Reject projects of which the IRR is lower than the cost of capital (required rate of return or opportunity rate of return). The total capital outlay and financing costs are not recovered and such a project would not be an economic proposition. This corresponds to a NPV < 0.

- Where the IRR is the same as the cost of capital, only the initial investment and associated financing costs are recovered and no additional return or value is obtained. This corresponds to a NPV = 0.
Table 8.3 DCF analysis of investment to determine DCFROI

<table>
<thead>
<tr>
<th>Year</th>
<th>Equipment A</th>
<th></th>
<th></th>
<th>Equipment B</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Net cash flow (CF) (R.M)</td>
<td>PV (R.M) at ROI</td>
<td>Net cash flow (CF) (R.M)</td>
<td>PV (R.M) at ROI</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0</td>
<td>-10.00</td>
<td>-10.0000</td>
<td>-10.00</td>
<td>-10.0000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>+2.00</td>
<td>1.7584</td>
<td>0.50</td>
<td>0.4495</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>+2.50</td>
<td>1.9325</td>
<td>1.00</td>
<td>0.8081</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>+3.50</td>
<td>2.3788</td>
<td>3.50</td>
<td>2.5426</td>
<td></td>
<td></td>
</tr>
<tr>
<td>4</td>
<td>+3.50</td>
<td>2.0914</td>
<td>5.00</td>
<td>3.2653</td>
<td></td>
<td></td>
</tr>
<tr>
<td>5</td>
<td>+3.50</td>
<td>1.8388</td>
<td>5.00</td>
<td>2.9354</td>
<td></td>
<td></td>
</tr>
<tr>
<td>TOTAL NPV</td>
<td>0.0000</td>
<td></td>
<td>TOTAL NPV</td>
<td>0.0000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>ROI</td>
<td>0.1374</td>
<td></td>
<td>ROI</td>
<td>0.1124</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Discount Rate Theoretically, the discount rate is just another form of the interest rate $i$ - only expressed as the amount that future cash flows must be discounted to be valued equally as if they were received today. The common usage of the term discount rate refers to the rate that a company seeks to obtain from its future cash inflows to balance the cash outflow necessary at the start of the project. In this sense, the discount rate applied is invariably higher than typical interest rates because;

- there is a higher risk
- it is a longer term investment
- the company is more tied to its investment to generate repayments
- if the company has limited funds, the rate must be at least as high as other rates from alternate investments.
8.1.3 Cash Flows and DCF Tabulations for Equipment Selection and Costing

A cash flow, when used to determine or compare equipment investment options accommodates:

- Capital and operating costs
- Variations in costs over time
- Variations in production rate over time
- The time value of money.

The types of investment decisions commonly encountered are:

- Equipment choice: Which of several possible options of ‘fit-for-purpose’ equipment is preferred? The choice often depends on which item is likely to provide the largest return on the investment made.

- Replacement: Should existing equipment be replaced with something which is more efficient? The future expected cash flows on this investment are the cost savings resulting from lower operating costs, or the profits from additional production, or both. These concepts are explored in a later section.

- Cost reduction: Similar to replacement, if new equipment is purchased to improve efficiency, expected cash flows result from lower operating costs.

- Lease or buy: Should a piece of equipment be bought or leased? Selection depends on whether or not the amount required to purchase the item can earn a better return than the cash savings which will result from avoiding lease payments.

In building up the cash flow, consistency in definitions is essential. DCF tabulations are normally prepared using the end of year convention. Purchases made at the beginning of a year are normally shown in the previous year - and in any one year, the operating costs and revenues are also shown at the end of the year. Tax payable is dependant on local tax regulations regarding timing of the payment and deferral, for example, depreciation claimed in year \( n \) is only payable in year \( n+1 \).

As a simple example (with several additional accounting assumptions made), assume equipment (loader and two trucks) are purchased for R16M initially (year 0) and they generate R6M revenue in the first year of production. The following year, two more trucks are
purchased for R5M and the revenue earned increases to R8M, thereafter R10M and R12M for the remaining years. Operating costs (OPEX) are R3M in the first year of production, increasing to R4M in the following years. The cash flow schedule is given in Table 8.4.

Table 8.4 Example cash flow schedule

<table>
<thead>
<tr>
<th>Year</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
</tr>
</thead>
<tbody>
<tr>
<td>CAPEX</td>
<td>16.00</td>
<td>5.00</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>OPEX</td>
<td>0.00</td>
<td>3.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td></td>
</tr>
<tr>
<td>Revenue</td>
<td>0.00</td>
<td>6.00</td>
<td>8.00</td>
<td>10.00</td>
<td>12.00</td>
<td>12.00</td>
<td>12.00</td>
<td></td>
</tr>
<tr>
<td>Unredeemed CAPEX</td>
<td>16.00</td>
<td>18.00</td>
<td>14.00</td>
<td>8.00</td>
<td>0.00</td>
<td>0.00</td>
<td>0.00</td>
<td></td>
</tr>
<tr>
<td>Profit before tax</td>
<td>0.00</td>
<td>0.00</td>
<td>0.00</td>
<td>0.00</td>
<td>8.00</td>
<td>8.00</td>
<td>8.00</td>
<td></td>
</tr>
<tr>
<td>LESS Tax @ 40%</td>
<td>0.00</td>
<td>0.00</td>
<td>0.00</td>
<td>0.00</td>
<td>3.20</td>
<td>3.20</td>
<td>3.20</td>
<td></td>
</tr>
<tr>
<td>Cash flow (Revenue-CAPEX-OPEX-Tax)</td>
<td>-16.00</td>
<td>-2.00</td>
<td>4.00</td>
<td>6.00</td>
<td>8.00</td>
<td>4.80</td>
<td>4.80</td>
<td>4.80</td>
</tr>
<tr>
<td>NPV of cash flow at i = 10%</td>
<td>-16.00</td>
<td>-1.82</td>
<td>3.31</td>
<td>4.51</td>
<td>5.46</td>
<td>2.98</td>
<td>2.71</td>
<td>2.46</td>
</tr>
<tr>
<td>Sum of cash flows</td>
<td>19.61</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

If the cash flow is discounted at 10%, the NPV is R19.61M and as such the project acceptable.

In the above simple example, CAPEX is not depreciated but is redeemed (claimed) against profits made. This is one of several protocols for dealing with depreciation.

**Depreciation** The process whereby the capitalised cost of an asset, such as a piece of equipment, is recovered in a manner determined by the local tax authority. Periods over which assets may be depreciated, and the methods of doing so, are prescribed by the tax authority. This a periodic cost allocation which may be offset against income to reduce taxable income and thereby reduce tax paid. It is not itself a cash outflow. For example, an asset costing R5M and depreciated in equal amounts over seven years will have the cash flow shown in Table 8.5.

Taxable earnings are equal to earnings less depreciation, and this reduction in tax payable is referred to as a tax benefit or tax shield. The reduction in tax payable and consequent increase in net earnings after tax to influences the NPV and ROI/IRR.

Table 8.5 Example of cashflow schedule with depreciation included

<table>
<thead>
<tr>
<th>Year</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
</tr>
</thead>
<tbody>
<tr>
<td>CAPEX</td>
<td>5.00</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Depreciation amount</td>
<td>0.00</td>
<td>4.29</td>
<td>3.57</td>
<td>2.86</td>
<td>2.14</td>
<td>1.43</td>
<td>0.71</td>
<td>0.00</td>
</tr>
<tr>
<td>Residual value</td>
<td>0.00</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Revenue</td>
<td>0.00</td>
<td>8.00</td>
<td>8.00</td>
<td>10.00</td>
<td>12.00</td>
<td>12.00</td>
<td>12.00</td>
<td></td>
</tr>
<tr>
<td>LESS operating costs (OPEX)</td>
<td>0.00</td>
<td>3.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td>4.00</td>
<td></td>
</tr>
<tr>
<td>Operating income</td>
<td>0.00</td>
<td>5.00</td>
<td>4.00</td>
<td>6.00</td>
<td>8.00</td>
<td>8.00</td>
<td>8.00</td>
<td></td>
</tr>
<tr>
<td>LESS depreciation claimable</td>
<td>0.00</td>
<td>4.29</td>
<td>3.57</td>
<td>2.86</td>
<td>2.14</td>
<td>1.43</td>
<td>0.71</td>
<td>0.00</td>
</tr>
<tr>
<td>Profit before tax</td>
<td>0.00</td>
<td>0.71</td>
<td>0.43</td>
<td>3.14</td>
<td>5.86</td>
<td>6.57</td>
<td>7.29</td>
<td>8.00</td>
</tr>
<tr>
<td>LESS tax @40%</td>
<td>0.00</td>
<td>0.29</td>
<td>0.17</td>
<td>1.28</td>
<td>2.34</td>
<td>2.63</td>
<td>2.91</td>
<td>3.20</td>
</tr>
<tr>
<td>Cash flow (Revenue-CAPEX-OPEX-Tax)</td>
<td>-5.00</td>
<td>-0.43</td>
<td>0.26</td>
<td>1.89</td>
<td>3.51</td>
<td>3.94</td>
<td>4.37</td>
<td>4.80</td>
</tr>
<tr>
<td>NPV of cash flow at i = 10%</td>
<td>-5.00</td>
<td>0.39</td>
<td>0.21</td>
<td>1.42</td>
<td>2.40</td>
<td>2.45</td>
<td>2.47</td>
<td>2.46</td>
</tr>
<tr>
<td>Sum of cash flows</td>
<td>11.80</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
At the end of its useful life, an asset may still possess a residual or salvage value. The difference between initial cost and salvage value is the amount which is allowed for depreciation over the estimated useful life. If the asset is disposed of before the end of its depreciation (useful) life, then the income from that sale at their written-down value is regarded as a revenue in the year of disposal.

There are several permissible methods of depreciation, the most common of which are:

- **Straight Line Depreciation** - this is the most common method in which the difference between initial and residual values is written off in equal amounts over the useful life.

- **Accelerated Depreciation Methods** - to improve cash flow in the early years of a project, and therefore improve the rate of return, it is advantageous to be able to increase the tax shield. This is achieved by accelerating the depreciation rate such that the value of the asset depreciates more early in its life rather than later.

The only difference caused to cash flows by the different methods is in timing and not in total sum. In an inflationary period, the timing can make a large difference to the NPV of the cost of owning the asset.

### 8.1.4 Discounted Average Cost Method

When using a DCF tabulation technique as a basis for equipment selection from a financial perspective, whilst all of the above factors are correctly represented in the DCF analysis, some applications, especially in equipment selection or comparison, cannot be directly analysed using this method. This is because, all of the cash flow is outflow, and the conventional DCF analysis will not work where no revenues are generated.

For this type of problem, a variation on the conventional DCF analysis, termed the Discounted Average Cost Method can be used. It is based on DCF analysis, with the assumptions that;

- The equipment selection activity is a cost-centre in its own right
- The revenue is in fact the cost (to pay for) a unit quantity of production (be it drilled meters, BCM, tons coal, etc.). The cost which, when applied to the production, yields a cash flow giving an NPV = 0 when discounted at the required ROI is the
discounted average cost of the production. This cost is then used as a basis to compare equipment options, etc., the lowest cost being selected as the optimal choice.

As with a conventional DCF, the preparation of the DCF for the above situation requires determination of:

- Capital costs (ownership)
- Operating costs

The details associated with the description and determination of these costs will be presented in the next sections of the Module, but at this stage, assumed costs will be used to illustrate the technique.

Tables 8.6 and 8.7 illustrate the application of the technique to two load and haul fleet options being considered for a five year earthmoving contract. In this example, depreciation is set at a percentage of the book value of the assets and the equipment sold at their residual values on completion of the contract.

The cost per ton (R3.40 and R2.90 in Tables 8.6 and 8.7 respectively) which dictates the revenue earned based on production tonnages per year, is determined by trial and error such that the sum of NPVs is zero at the selected discount rate for the project.
Table 8.6 Example discounted average cost method of analysis – fleet option (1)

<table>
<thead>
<tr>
<th>Basic data</th>
<th>Truck</th>
<th>Loader</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Capital Cost</td>
<td>12,000.000.00</td>
<td>30,000.000.00</td>
</tr>
<tr>
<td>Units purchased</td>
<td>4</td>
<td>1</td>
</tr>
<tr>
<td>Residual Value Fraction of Purchase Price</td>
<td>0.25</td>
<td>0.30</td>
</tr>
<tr>
<td>Depreciation Rate</td>
<td>0.15</td>
<td>0.15</td>
</tr>
<tr>
<td>Depreciation Method</td>
<td>Straight</td>
<td></td>
</tr>
<tr>
<td>Marginal Tax Rate</td>
<td>0.30</td>
<td></td>
</tr>
<tr>
<td>Required Return on Investment</td>
<td>0.10</td>
<td></td>
</tr>
<tr>
<td>Operating Shifts per Year</td>
<td>7.15</td>
<td></td>
</tr>
<tr>
<td>Schedules Operating Time per Shift</td>
<td>7.6</td>
<td></td>
</tr>
<tr>
<td>Operating Cost (per Operating Hour)</td>
<td>270.00</td>
<td>305.00</td>
</tr>
<tr>
<td>Availability in Scheduled Shift Time</td>
<td>0.8</td>
<td>0.3</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Year</th>
<th>Production ($)</th>
<th>Capital Costs</th>
<th>Operating Costs</th>
<th>DCF Analysis</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>9,000,000.00</td>
<td>1,882,361.00</td>
<td>27,285,160.00</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>9,000,000.00</td>
<td>1,882,361.00</td>
<td>27,285,160.00</td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>9,000,000.00</td>
<td>1,882,361.00</td>
<td>27,285,160.00</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>9,000,000.00</td>
<td>1,882,361.00</td>
<td>27,285,160.00</td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>9,000,000.00</td>
<td>1,882,361.00</td>
<td>27,285,160.00</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Operating Costs</th>
<th>Lead cost per year</th>
<th>Truck fleet cost per year</th>
<th>Total operating costs per year</th>
<th>DCF Analysis</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1,882,361.00</td>
<td>4,225,478.40</td>
<td>6,107,839.40</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>1,882,361.00</td>
<td>4,225,478.40</td>
<td>6,107,839.40</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>1,882,361.00</td>
<td>4,225,478.40</td>
<td>6,107,839.40</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>1,882,361.00</td>
<td>4,225,478.40</td>
<td>6,107,839.40</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>1,882,361.00</td>
<td>4,225,478.40</td>
<td>6,107,839.40</td>
<td>0.000</td>
</tr>
<tr>
<td>Basic data</td>
<td>Truck</td>
<td>Loader</td>
<td></td>
<td></td>
</tr>
<tr>
<td>------------------------------------------------</td>
<td>-------</td>
<td>--------</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Initial Capital Cost</td>
<td>21,909,000.00</td>
<td>28,909,000.00</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Units purchased</td>
<td>2</td>
<td>1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Residual Value Fraction of Purchase Price</td>
<td>0.25</td>
<td>0.30</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Depreciation Rate</td>
<td>0.15</td>
<td>0.15</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Depreciation Method</td>
<td>Straight</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Marginal Tax Rate</td>
<td>0.30</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Required Return on Investment</td>
<td>0.10</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Operating Shifts per Year</td>
<td>7.15</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Scheduled Operating Time per Shift</td>
<td>7.6</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Operating Cost (per Operating Hour)</td>
<td>315.00</td>
<td>426.00</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Availability in Scheduled Shift Time</td>
<td>0.89</td>
<td>0.84</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Year</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Production ($)</td>
<td>0</td>
<td>9,000,000.00</td>
<td>9,000,000.00</td>
<td>9,000,000.00</td>
<td>9,000,000.00</td>
<td>9,000,000.00</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Capital Costs</th>
</tr>
</thead>
<tbody>
<tr>
<td>Residual Value</td>
</tr>
<tr>
<td>Value for depreciation</td>
</tr>
<tr>
<td>Depreciation amount</td>
</tr>
<tr>
<td>Residual value</td>
</tr>
<tr>
<td>Value for depreciation</td>
</tr>
<tr>
<td>Depreciation amount</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Operating Costs</th>
</tr>
</thead>
<tbody>
<tr>
<td>Leader cost per year</td>
</tr>
<tr>
<td>Truck fleet cost per year</td>
</tr>
<tr>
<td>Total operating costs per year</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>DCF Analysis</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cost per ton (Revenue)</td>
</tr>
<tr>
<td>LESS operating costs</td>
</tr>
<tr>
<td>Operating income</td>
</tr>
<tr>
<td>LESS depreciation claimable</td>
</tr>
<tr>
<td>Profit before tax</td>
</tr>
<tr>
<td>LESS tax</td>
</tr>
<tr>
<td>Cashflow (Revenue less costs)</td>
</tr>
<tr>
<td>NPV of cash flow at ROI</td>
</tr>
<tr>
<td>Sum of cash flows</td>
</tr>
</tbody>
</table>
8.2 Equipment Replacement Decisions

One of the primary considerations in equipment selection and application concerns whether or not a particular piece of equipment should be repaired or replaced if the maintenance or repair costs become excessive. If a replacement decision is made the next question to be answered is what should be purchased to give the best return, a decision which requires certain techniques to evaluate, particularly if replacement options have different economic lives. Finally, once a replacement decision is made, it is necessary to identify the financial benefits of replacement equipment.

To gather these costs successfully an effective management accounting system is required from which all costs related to both existing and proposed equipment can be obtained. These costs must include purchase price, residual value, running, maintenance and repair costs. In addition, the actual and expected availability of existing and proposed equipment should be recorded, together with costs of lost production or for maintaining standby units if appropriate. It is sound practice to cross reference sources of cost data to highlight any discrepancies. Problems are often encountered where mines have poor accounting systems which show very low or high unit costs for equipment because everything is allocated to “other” cost categories and therefore providing misleading information.

8.2.1 Analysis of Replacement

Often there is reluctance to, or a budgetary constraint on, replacing a piece of equipment and it is kept in operation by a continuous process of repair, but there is in fact an optimum economic life associated with equipment. Since different lifetimes must be compared, the most appropriate means of comparison utilises the equivalent annual cost (EAC) concept. Based on financial considerations only, Table 8.8 shows that the equipment in question should be replaced when the EAC reaches a minimum, i.e. after 3 years of operation.

An alternative analysis method, based on the discounted payback period, can be used if the savings the new equipment generates are known. If the savings per period are equal, the investment represents the present value of an annuity at the prescribed discount rate. For the investment to be attractive, the annuity must be of a minimum duration, which is the minimum acceptable life of the asset.
Table 8.8 Equipment equivalent annual costs

<table>
<thead>
<tr>
<th>Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Capital cost</td>
<td>40,000.00</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Capital recovery factor @ i = 10%</td>
<td>1.120</td>
<td>0.592</td>
<td>0.416</td>
<td>0.329</td>
</tr>
<tr>
<td></td>
<td>EAC of capital</td>
<td>44,800.00</td>
<td>23,667.92</td>
<td>16,653.96</td>
<td>13,169.38</td>
</tr>
<tr>
<td></td>
<td>Maintenance and overhaul costs</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Maintenance per year</td>
<td>4,000.00</td>
<td>4,500.00</td>
<td>5,000.00</td>
<td>6,000.00</td>
</tr>
<tr>
<td></td>
<td>Annual overhaul</td>
<td>2,000.00</td>
<td>2,100.00</td>
<td>2,300.00</td>
<td>2,600.00</td>
</tr>
<tr>
<td></td>
<td>Total maintenance and overhaul costs</td>
<td>4,000.00</td>
<td>6,500.00</td>
<td>7,100.00</td>
<td>8,300.00</td>
</tr>
<tr>
<td></td>
<td>Annuity factor @ i = 10%</td>
<td>0.893</td>
<td>1.690</td>
<td>2.402</td>
<td>3.037</td>
</tr>
<tr>
<td></td>
<td>EAC of maintenance and overhaul</td>
<td>3,571.43</td>
<td>10,985.33</td>
<td>17,053.00</td>
<td>25,210.00</td>
</tr>
<tr>
<td></td>
<td>ADD EAC of capital</td>
<td>44,800.00</td>
<td>23,667.92</td>
<td>16,653.96</td>
<td>13,169.38</td>
</tr>
<tr>
<td>TOTAL EAC</td>
<td>48,371.43</td>
<td>34,653.26</td>
<td>33,706.96</td>
<td>38,379.38</td>
<td>49,307.02</td>
</tr>
</tbody>
</table>

For example, if a machine costing R1,200,000 effects savings of R150,000 per year, the minimum payback period (PBP) at 10% cost of capital is given as the corresponding time value of the annuity factor determined as:

\[
Annuity\ factor = \frac{1200000}{150000} = 8
\]

An annuity factor of 8, at 10% cost of capital, is equivalent to a PBP of 16.9 years, given by:

\[
\begin{align*}
    n &= \left[ -\frac{\ln(1 - Ai)}{\ln(1 + i)} \right] = \left[ \frac{\ln(1 - 8 \times 0.1)}{\ln(1 + 0.1)} \right] = 16.9
\end{align*}
\]

8.2.2 Benefits of Replacement

To estimate the benefits of equipment replacement, a DCF analysis is required. Using the DCF method, with appropriate assumptions – in this example regarding equipment life (7 years), depreciation (50% of the residual value each year), tax (40%, no depreciation carry-over), incomes including residuals from the previous and new equipment purchases) and cost savings (from new equipment purchase), the NPV of the replacement and the IRR can be found, as given below in Table 8.9 for this example. In this case, the replacement has an IRR of 25% and an NPV of R1,77M.
### Table 8.9 Analysis of benefits of replacement by DCF

<table>
<thead>
<tr>
<th>Year</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
</tr>
</thead>
<tbody>
<tr>
<td>CAPEX (R.M)</td>
<td>1.5000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Depreciation amount</td>
<td>0.0000</td>
<td>0.3000</td>
<td>0.2400</td>
<td>0.1920</td>
<td>0.1536</td>
<td>0.1229</td>
<td>0.0983</td>
<td>0.0786</td>
</tr>
<tr>
<td>Residual value</td>
<td>0.0025</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>0.3146</td>
</tr>
<tr>
<td>Operating income (savings)</td>
<td>0.0000</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.5400</td>
</tr>
<tr>
<td>LESS depreciation claimable</td>
<td>0.0000</td>
<td>0.3000</td>
<td>0.2400</td>
<td>0.1920</td>
<td>0.1536</td>
<td>0.1229</td>
<td>0.0983</td>
<td>0.0786</td>
</tr>
<tr>
<td>Profit before tax</td>
<td>0.0000</td>
<td>0.2400</td>
<td>0.3000</td>
<td>0.3480</td>
<td>0.3864</td>
<td>0.4171</td>
<td>0.4417</td>
<td>0.4614</td>
</tr>
<tr>
<td>LESS tax @40%</td>
<td>0.0000</td>
<td>0.0000</td>
<td>0.0000</td>
<td>0.1392</td>
<td>0.1546</td>
<td>0.1668</td>
<td>0.1767</td>
<td>0.1845</td>
</tr>
<tr>
<td>Cash flow</td>
<td>-1.4975</td>
<td>0.5400</td>
<td>0.5400</td>
<td>0.4008</td>
<td>0.3854</td>
<td>0.3732</td>
<td>0.3633</td>
<td>0.3600</td>
</tr>
<tr>
<td>NPV of cash flow</td>
<td>-1.4975</td>
<td>0.4320</td>
<td>0.3456</td>
<td>0.2002</td>
<td>0.1575</td>
<td>0.1229</td>
<td>0.0952</td>
<td>0.0465</td>
</tr>
<tr>
<td>Sum of cash flows</td>
<td>0.0000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### 8.3 Estimating Capital and Operating Costs

Capital and operating costs are fundamental to any economic analysis of equipment, mine planning or feasibility studies, budgeting and data for analysis of different mining sub-system options. These costs are, however, often difficult to estimate reliably. Sources of data are typically:

- The company’s own operations
- Operating mines with a history of using similar plant
- Previous studies
- Equipment manufacturers or suppliers
- Consultants, Government and industry authorities
- Contractor quotations
- Rules and formulae. Text books and handbooks have some formulae but these are often too generalised and are more geared to the American market. For these to be used successfully, the local cost factors are required – and these are not simply an exchange rate correction, but must also consider relative cost of parts, labours, fuel, etc.

When attempting to use or corroborate costs from several sources listed above, there are often inconsistencies which make the selection of a representative cost figure somewhat qualitative. These discrepancies arise due to variable job condition factors, quality of maintenance, location, mine accounting practices, or abnormally high or low costs at a particular mine which are not fully reported or benchmarked across the industry. This results in the necessity for...
using a systematic approach to cost estimation. The advantages and importance of determining costs from first principles and then cross-checking them are:

- Costs can be developed for any mining equipment, not just equipment already in use on the mine site
- There is consistency between costs and estimation assumptions used and this consistency can be checked, or the assumptions modified, to generate a rough indication of cost sensitivity or assumption risks.

A cost data base, the components of which are illustrated in Figure 8.2, is the starting point for an economic analysis. Figure 8.3 shows how the relevant elements in the cost base feed into the economic analysis model of a mine, a significant component of which is the equipment selection and costing aspect.

Figure 8.2 Cost base for cash flow development, some elements pertaining to equipment costing.

8.3.1 Estimating Cost of Ownership

To protect equipment investment and be able to replace or recoup equipment value, the mine must recover over the machine’s useful life
an amount equal to the loss in resale value plus the other costs of owning the equipment including interest, insurance, freight and taxes.

If mine can estimate the loss of value on resale in advance, the original equipment investment is recovered by establishing depreciation schedules according to the various uses of the equipment. Proper financial and tax assistance is a necessity when establishing depreciation schedules and allowances. Considering economic conditions and the trend toward larger, more expensive equipment, many mines choose to keep large equipment working well after they have been fully depreciated for tax purposes. On the other hand, tax incentives in many areas may favour trading a machine well before it approaches the limits of its operating life.

Figure 8.3 Flow chart of general mine costing procedure, showing role of equipment selection and costing

Accordingly, it is imperative that careful consideration be given to the selection of depreciation periods, and that for owning and operating cost calculations they be based on “operating life” rather than “accounting life”. Accounting life is used for tax purposes and represents how quickly the equipment can be written off for tax purposes. The operating life is the actual life the mine expects the equipment to operate at an availability satisfactory to meet production targets and at an economic cost.
The starting point is therefore to establish a realistic operating life for a piece of equipment. Table 8.10 gives typical operating life for various equipment based on three categories of job condition;

- **Moderate conditions** - medium throttle applications, light utility work, soft or free-flowing materials, shallow depth digging, no overloading, good underfoot conditions, excellent roads, low grades, low load factors.

- **Average conditions** - intermittent full throttle applications, medium impact conditions, continuous digging in well-blasted rock, fairly tight benches, loading to capacity (average load close to recommended), poor underfoot conditions with some maneuvering, mostly good roads and moderate load factors.

- **Severe conditions** - high throttle applications, heavy rock ripping, continuous high impact conditions, poorly blasted rock or hard free-digging, poor uneven, wet floors with machine sliding on undercarriage, overloading, poor roads and steep grades, high load factor.

The above descriptions are generic and apply to any of the equipment listed in Table 8.10.

Maintenance practices are not considered in Table 8.10 but play an important part in determining economic machine life. For example, operating conditions may suggest a 12,000 hour operating life for a machine, but poor maintenance could make it uneconomical to retain the unit beyond 10,000 hours. Good, regular maintenance often can extend economical machine life. Therefore, a knowledge of the intended use, operating conditions and maintenance practices, plus any special factors, is essential in establishing expected machine life for depreciation purposes.

Ownership costs can be established using several methods. The first technique involves calculating the sum of the straight line depreciation and a percentage of the annual average investment to cover interest taxes, insurance and other ownership cost items. Table 8.11 shows a typical example calculation.

An alternative approach is to treat the equipment as if it were being leased. Lease rates include all of the factors in Table 8.11 but more correctly account for the higher interest component of the cost earlier in the equipment life. This allows a capital recovery factor to be calculated when the capital PV is spread over the period of ownership years and produces a series of equal values occurring at the end of each year for the period specified. Table 8.12 shows the calculation for the equivalent annual cost (EAC) approach.
Table 8.10  Typical equipment operating life

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Expected Operating Life (Equipment clock hours)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Severe</td>
</tr>
<tr>
<td>Bowl-scraper</td>
<td></td>
</tr>
<tr>
<td></td>
<td>8,000</td>
</tr>
<tr>
<td>Dozer</td>
<td></td>
</tr>
<tr>
<td>Track</td>
<td>10,000</td>
</tr>
<tr>
<td>Wheel</td>
<td>8,000</td>
</tr>
<tr>
<td>Grader</td>
<td></td>
</tr>
<tr>
<td></td>
<td>12,000</td>
</tr>
<tr>
<td>FEL</td>
<td></td>
</tr>
<tr>
<td></td>
<td>10,000</td>
</tr>
<tr>
<td>Hydraulic shovel</td>
<td></td>
</tr>
<tr>
<td></td>
<td>20,000</td>
</tr>
<tr>
<td>Rope shovels</td>
<td></td>
</tr>
<tr>
<td></td>
<td>60,000</td>
</tr>
<tr>
<td>Dragline</td>
<td></td>
</tr>
<tr>
<td></td>
<td>60,000</td>
</tr>
<tr>
<td>Haul truck</td>
<td></td>
</tr>
<tr>
<td>BD</td>
<td>30,000</td>
</tr>
<tr>
<td>RD-Small</td>
<td>15,000</td>
</tr>
<tr>
<td>RD-Large</td>
<td>20,000</td>
</tr>
<tr>
<td>Drills</td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>16,000</td>
</tr>
<tr>
<td>Large</td>
<td>60,000</td>
</tr>
<tr>
<td>Continuous mining equipment</td>
<td></td>
</tr>
<tr>
<td>In-pit-crushers, spreaders, reclaimers, BWE</td>
<td>60,000</td>
</tr>
</tbody>
</table>
### Table 8.11  Determination of cost of ownership - annual average investment method

<table>
<thead>
<tr>
<th>Equipment type</th>
<th>FEL</th>
</tr>
</thead>
<tbody>
<tr>
<td>Estimated period of ownership (yrs)</td>
<td>10</td>
</tr>
<tr>
<td>Estimated hours/year</td>
<td>5000</td>
</tr>
<tr>
<td>Ownership hours</td>
<td>50000</td>
</tr>
<tr>
<td>Residual value fraction of purchase price</td>
<td>10</td>
</tr>
<tr>
<td>Cost of capital (%)</td>
<td>12</td>
</tr>
<tr>
<td>Insurance (%)</td>
<td>2.5</td>
</tr>
<tr>
<td>Ownership tax (%)</td>
<td>0</td>
</tr>
</tbody>
</table>

**COSTS**

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Delivered price</td>
<td>7,300,000.00</td>
</tr>
<tr>
<td>LESS tyre replacement cost</td>
<td>280,000.00</td>
</tr>
<tr>
<td>Delivered price less tyres</td>
<td>7,020,000.00</td>
</tr>
<tr>
<td>LESS residual value at replacement</td>
<td>730,000.00</td>
</tr>
<tr>
<td>Value to be recovered</td>
<td>6,290,000.00</td>
</tr>
<tr>
<td>Cost/hour</td>
<td>125.80</td>
</tr>
<tr>
<td>ADD interest charges /hour</td>
<td>92.66</td>
</tr>
<tr>
<td>ADD insurance charges /hour</td>
<td>19.31</td>
</tr>
<tr>
<td>ADD ownership tax /hour</td>
<td>0.00</td>
</tr>
<tr>
<td>TOTAL HOURLY OWNERSHIP COST</td>
<td>237.77</td>
</tr>
</tbody>
</table>

### Table 8.12  Determination of cost of ownership - equivalent annual cost (EAC) method

<table>
<thead>
<tr>
<th>Equipment type</th>
<th>FEL</th>
</tr>
</thead>
<tbody>
<tr>
<td>Estimated period of ownership (yrs)</td>
<td>10</td>
</tr>
<tr>
<td>Estimated hours/year</td>
<td>5000</td>
</tr>
<tr>
<td>Ownership hours</td>
<td>50000</td>
</tr>
<tr>
<td>Residual value fraction of purchase price</td>
<td>10</td>
</tr>
<tr>
<td>Cost of capital (%)</td>
<td>12</td>
</tr>
<tr>
<td>Insurance (%)</td>
<td>2.5</td>
</tr>
<tr>
<td>Ownership tax (%)</td>
<td>0</td>
</tr>
</tbody>
</table>

**COSTS**

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Delivered price</td>
<td>7,300,000.00</td>
</tr>
<tr>
<td>LESS tyre replacement cost</td>
<td>280,000.00</td>
</tr>
<tr>
<td>Delivered price less tyres</td>
<td>7,020,000.00</td>
</tr>
<tr>
<td>LESS residual value at replacement</td>
<td>730,000.00</td>
</tr>
<tr>
<td>Value to be recovered</td>
<td>6,290,000.00</td>
</tr>
<tr>
<td>EAC factor @ 12% over 10 yrs</td>
<td>0.18</td>
</tr>
<tr>
<td>EAC of equipment per year</td>
<td>1,113,330.00</td>
</tr>
<tr>
<td>Cost/hour</td>
<td>222.67</td>
</tr>
<tr>
<td>ADD insurance charges /hour</td>
<td>19.31</td>
</tr>
<tr>
<td>ADD ownership tax /hour</td>
<td>0.00</td>
</tr>
<tr>
<td>TOTAL HOURLY OWNERSHIP COST</td>
<td>241.97</td>
</tr>
</tbody>
</table>
If tax is considered, together with alternative depreciation schedules, a discounted average cost can be determined a cash flow drawn-up to include the timing of costs and assuming insurance (and ownership tax, if applicable) are the only operating expenditures. The required ROI can also be included in the analysis. Table 8.13 shows a typical estimation of ownership hourly costs using this technique.

Table 8.13 Determination of cost of ownership – discounted average cost method (years 0-3 only shown)

<table>
<thead>
<tr>
<th>Basic data</th>
<th>FEL</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Capital Cost</td>
<td>7,020,000.00</td>
</tr>
<tr>
<td>Units purchased</td>
<td>1</td>
</tr>
<tr>
<td>Residual Value Fraction of Purchase Price</td>
<td>10.00</td>
</tr>
<tr>
<td>Depreciation Method</td>
<td>Straight</td>
</tr>
<tr>
<td>Marginal Tax Rate</td>
<td>0.30</td>
</tr>
<tr>
<td>Required Return on Investment</td>
<td>0.10</td>
</tr>
<tr>
<td>Insurance (%)</td>
<td>2.50</td>
</tr>
<tr>
<td>Estimated period of ownership (yrs)</td>
<td>10</td>
</tr>
<tr>
<td>Estimated hours/year</td>
<td>5000</td>
</tr>
<tr>
<td>Ownership hours</td>
<td>50000</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Year</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Capital Costs</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Residual value</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Value for depreciation</td>
<td>7,020,000.00</td>
<td>7,020,000.00</td>
<td>5,573,880.00</td>
<td>4,425,660.72</td>
</tr>
<tr>
<td>Depreciation amount</td>
<td>1,446,120.00</td>
<td>1,148,219.28</td>
<td>911,686.11</td>
<td></td>
</tr>
<tr>
<td>Operating Costs</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Yearly cost</td>
<td>-1,426,000.00</td>
<td>-1,426,000.00</td>
<td>-1,426,000.00</td>
<td>-1,426,000.00</td>
</tr>
<tr>
<td>Hourly cost</td>
<td>285.20</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>DCF Analysis</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cash flow</td>
<td>-7,020,000.00</td>
<td>-4,147,880.00</td>
<td>-2,999,660.72</td>
<td>-2,087,974.61</td>
</tr>
<tr>
<td>NPV of cash flow at ROI</td>
<td>-7,020,000.00</td>
<td>-3,776,800.00</td>
<td>-2,479,058.45</td>
<td>-1,568,726.23</td>
</tr>
<tr>
<td>Sum of cash flows</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

8.3.2 Estimating Cost of Operation

The recommended approach described here is to build up operating cost estimates from first principles, then cross check the costs thus derived with actual mine statistics and whatever “rules of thumb” are available. The approach has been generalised to enable the basic principles to be applied to any piece of mining equipment.

The steps involved in deriving operating costs are;

- Split the machine into defined cost elements such as;
  - power or fuel
  - lubrication and filters
  - tyres or crawler and undercarriage
  - repair parts (maintenance supplies)
wear parts (operating supplies), such as ground engaging tools (GET), bits, etc.

- major overhauls
- operating labour
- maintenance labour

- Subdivide each cost element into component parts.
- Assign a life or utilisation to each component part and calculate the hourly cost.
- Total all the components to achieve the total hourly operating cost.

Deriving costs from first principles is certainly the most reliable method. However, determining the costs and life for all components in a machine is quite time consuming and it is often difficult to get accurate information. To overcome this problem, a set of factors or formulae can be used to describe, in general terms, the conditions under which a piece of equipment is operating. Overall job conditions can be classified into the three main categories described earlier, but attention should be given to the critical role of;

- Material characteristics such as density, swell, abrasiveness, hardness
- Water and dust
- Job conditions - especially operator skills
- Labour factors including management, maintenance philosophy and availability of spare parts
- Utilisation factors such as the annual operating hours, average engine load factors

on the final estimate of operating costs.

**FUEL OR POWER COSTS**

Fuel consumption can be closely measured in the field. However, if no opportunity exists to do this, consumption can be predicted when the machine application is known. Consumption is based on engine size (kW), consumption in litres per kW and load factor. Several
simulation packages can also determine fuel consumptions, based on engine-torque and fuel consumption maps.

Equipment application determines engine load factor which in turn controls engine fuel consumption. An engine continuously producing full rated kilowatts is operating at a load factor of 1.0. Earthmoving machines may reach a load factor of 1.0 intermittently, but seldom operate at this level for extended periods of time. Periods spent at idle, doze and push or travel in reverse, haul units traveling empty, close maneuvering at part throttle and operating downhill are examples of conditions which reduce load factors.

To determine typical fuel consumption, a load factor multiplier is used to estimate the increase in fuel consumption from moderate conditions. Table 8.14 summarises typical equipment average load factor multipliers and Figure 8.4 shows the range of fuel consumptions (moderate conditions) for the equipment listed.

Table 8.14 Equipment average load factors multipliers for various job conditions, based on moderate conditions = 1.00

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Average</th>
<th>Severe</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bowl scraper</td>
<td>1.3</td>
<td>1.8</td>
</tr>
<tr>
<td>Dozer</td>
<td>1.35</td>
<td>1.75</td>
</tr>
<tr>
<td>Grader</td>
<td>1.2</td>
<td>1.7</td>
</tr>
<tr>
<td>FEL</td>
<td>1.3</td>
<td>1.6</td>
</tr>
<tr>
<td>Hydraulic shovel</td>
<td>1.1</td>
<td>1.25</td>
</tr>
<tr>
<td>Haul truck</td>
<td>1.5</td>
<td>2.0</td>
</tr>
<tr>
<td>Drill</td>
<td>1.2</td>
<td>1.4</td>
</tr>
</tbody>
</table>

Using Table 8.14 and Figure 8.4, for a haul truck with a 1500kW engine, operating under severe conditions, the hourly fuel consumption is thus 75 x 2 = 150 litres per hour.

For equipment using electric power, consumption is subdivided into an “energy component” as well as a “demand” component which reflects the required installed capacity of the power generation facility or transmission system. This demand charge comes about because of the cyclical loads of most mining machines and the electricity
authority must be able to supply high power for short periods of time. Equipment suppliers can supply this information.

![Diagram of fuel consumption for various equipments.](image)

**Figure 8.4** Estimates of fuel consumption for various equipments. To be used in conjunction with Table 8.14

For quick estimates, the average power for a shovel is 0.6kW per cubic metre per hour and 1.5kW per cubic metre per hour for a dragline. For a large mine with a large number of electrical items of plant the demand charge can be assumed to be similar to the energy charge. If, however there is only one item of plant this is not reasonable as the diversification is much less and the effect of peaks are much more pronounced. Demand is typically 2-2.5 the average power.

**LUBRICATION AND FILTER COSTS**

If no detailed lubrication charges are available they can be calculated as a percentage of the hourly fuel cost. These proportions range from
15% for equipment with a relatively low proportion of hydraulic components up to 30-40% for equipment with a high proportion of hydraulic components (such as a hydraulic excavator). Adjustments may be made to these figures depending on how severe the duty is, heavy dust, deep mud or water typically increase quantities by 25%. For filters, the proportion ranges from 10-30% depending on size and hydraulic complexity of the equipment.

Alternatively the consumption rate can be expressed as either litres/hour or kg/hour which can be obtained from manufacturers or operational records. These are then multiplied by their appropriate unit cost. This is obviously a more accurate method and is possibly the only method for equipment such as draglines which consume substantial quantities of lubricants but no fuel oil.

Table 8.15 presents generalised data for lubricant costing.

**TYRES OR CRAWLER AND UNDERCARRIAGE**

Since tyres are considered a wear item when estimating owning and operating costs, total tyre replacement cost is deducted from machine delivered price to arrive at a net figure for depreciation purposes. Outlay for tyres is then included as an item in operating costs (as shown in Table 8.11).

Total tyre costs are obtained by multiplying the cost of each tyre by the number of tyres and dividing by the hourly life. Tyre manufacturers give guidelines for calculating hourly life, based on a series of reduction factors for base average life. For a coal strip mine, a good average would be at least 5000 hours per tyre.

Base average tyre life is given in Table 8.16, to which various tyre life adjustment factors are applied according to specific conditions, as given in Table 8.17. Using Base Hours (or kilometers), multiply by the appropriate factor for each condition to obtain approximate estimated hours (or kilometers) as the final product.

As an example, an off-highway truck equipped with E-4 drive tyres running on a well maintained haul road having easy curves and minimum grades and receiving “average” tyre maintenance attention but being 20% overloaded will have a life of some 2114 hours, given by the factor combination:

\[
3510 \times (0.981 \times 0.872 \times 0.981 \times 0.872 \times 0.981 \times 0.981 \times 0.981) = 2114 \text{ hours}
\]
Table 8.15 Generalised data for lubricant costing

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Lubricants litres per hour</th>
<th>Hydraulics litres per hour</th>
<th>Filters per hour</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bowl scraper</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.15</td>
<td>0.05</td>
<td>0.05</td>
</tr>
<tr>
<td>Large</td>
<td>1.00</td>
<td>0.1</td>
<td>0.1</td>
</tr>
<tr>
<td>Dozer</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.25</td>
<td>0.025</td>
<td>0.05</td>
</tr>
<tr>
<td>Large</td>
<td>0.75</td>
<td>0.1</td>
<td>0.1</td>
</tr>
<tr>
<td>Grader</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.2</td>
<td>0.05</td>
<td>0.05</td>
</tr>
<tr>
<td>Large</td>
<td>0.8</td>
<td>0.125</td>
<td>0.1</td>
</tr>
<tr>
<td>FEL</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.15</td>
<td>0.05</td>
<td>0.05</td>
</tr>
<tr>
<td>Large</td>
<td>1.75</td>
<td>0.45</td>
<td>0.2</td>
</tr>
<tr>
<td>Hydraulic shovel</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.15</td>
<td>0.2</td>
<td>0.15</td>
</tr>
<tr>
<td>Large</td>
<td>1.25</td>
<td>0.85</td>
<td>0.2</td>
</tr>
<tr>
<td>Haul truck</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.5</td>
<td>0.25</td>
<td>0.1</td>
</tr>
<tr>
<td>Large</td>
<td>1.0</td>
<td>0.6</td>
<td>0.25</td>
</tr>
<tr>
<td>Drill</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Small</td>
<td>0.2</td>
<td>0.075</td>
<td>0.05</td>
</tr>
<tr>
<td>Large</td>
<td>0.35</td>
<td>0.15</td>
<td>0.075</td>
</tr>
</tbody>
</table>
Table 8.16  Base average tyre life

<table>
<thead>
<tr>
<th>Type of Tyre</th>
<th>Base Average Life</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Hours</td>
</tr>
<tr>
<td>E-3 Std. Bias Tread</td>
<td>2510</td>
</tr>
<tr>
<td>E-4 Bias Extra Tread Depth</td>
<td>3510</td>
</tr>
<tr>
<td>E-4 Radial Extra Tread Depth</td>
<td>4200</td>
</tr>
</tbody>
</table>

For track-type machines, there is often little correlation between undercarriage costs and basic machine costs. This is because the undercarriage can be employed in an extremely abrasive, high-wear environment while the basic machine may be in an essentially easy application, and vice-versa. For that reason, the hourly cost of undercarriage is often calculated separately as a wear item rather than being included in the repair and maintenance cost breakdown for the basic machine. Note that for dozers working in coal stockpile handling applications, the undercarriage costs are nominal.

Three primary conditions affect probable life-expectancy of track-type undercarriage;

- Impact. The most measurable effect of impact is structural, i.e. bending, chipping, cracking, spalling, roll-over, etc., and problems with hardware and pin and bushing retention.

- Abrasiveness. The tendency of the underfoot materials to grind away the wear surfaces of track components.

- Environmental, operational and maintenance. Represents the combined effect on component life of the many intangible considerations on a given job. To assist in arriving at an appropriate value for the effect, maintenance will represent about 50% of its effect, environment and terrain 30%, and operator practices 20%.

To estimate the hourly undercarriage cost, use a manufacturer's basic cost and multiply it by the sum of the condition factors given in Table 8.18.
Table 8.17 Tyre service life reduction factors

<table>
<thead>
<tr>
<th>Factor</th>
<th>Condition</th>
<th>Tyre Service Life Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Maintenance</strong></td>
<td>Excellent</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>Average</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Poor</td>
<td>0.763</td>
</tr>
<tr>
<td><strong>Speeds (Maximum)</strong></td>
<td>&gt;15km/h</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>&gt;30 km/h</td>
<td>0.872</td>
</tr>
<tr>
<td></td>
<td>&gt;50 km/h</td>
<td>0.763</td>
</tr>
<tr>
<td><strong>Surface Conditions</strong></td>
<td>Soft Earth - No Rock</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>Soft Earth - Some Rock</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Well Maintained - Gravel Road</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Poor Maintained - Gravel Road</td>
<td>0.763</td>
</tr>
<tr>
<td></td>
<td>Blasted - Sharp Rocks</td>
<td>0.654</td>
</tr>
<tr>
<td><strong>Wheel Positions</strong></td>
<td>Trailing</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>Front</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Driver (Rear Dump)</td>
<td>0.872</td>
</tr>
<tr>
<td></td>
<td>(Bottom Dump)</td>
<td>0.763</td>
</tr>
<tr>
<td></td>
<td>(Self Propelled Scraper)</td>
<td>0.654</td>
</tr>
<tr>
<td><strong>Loading</strong></td>
<td>Recommended Loading</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>20% Overload</td>
<td>0.872</td>
</tr>
<tr>
<td></td>
<td>40% Overload</td>
<td>0.545</td>
</tr>
<tr>
<td><strong>Curves</strong></td>
<td>None</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>Medium</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Severe</td>
<td>0.872</td>
</tr>
<tr>
<td>Grades (Drive Tyres Only)</td>
<td>Level</td>
<td>1.090</td>
</tr>
<tr>
<td>---------------------------</td>
<td>-------</td>
<td>-------</td>
</tr>
<tr>
<td></td>
<td>5% Max.</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>15% Max.</td>
<td>0.763</td>
</tr>
<tr>
<td>Other Miscellaneous Combinations (See note below)</td>
<td>None</td>
<td>1.090</td>
</tr>
<tr>
<td></td>
<td>Medium</td>
<td>0.981</td>
</tr>
<tr>
<td></td>
<td>Severe</td>
<td>0.872</td>
</tr>
</tbody>
</table>

Miscellaneous condition used when overloading is present in combination with one or more of the primary conditions of maintenance, speeds, surface conditions and curves. The combination of severe levels in these conditions, together with an overload, will create a new and more serious condition which will contribute to early tyre failure to a larger extent than will the individual factors of each condition.

Table 8.18 Conditions multipliers for undercarriage costing

<table>
<thead>
<tr>
<th>Condition</th>
<th>Impact</th>
<th>Abrasiveness</th>
<th>EOM</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>0.3</td>
<td>0.4</td>
<td>1.0</td>
</tr>
<tr>
<td>Moderate</td>
<td>0.2</td>
<td>0.2</td>
<td>0.5</td>
</tr>
<tr>
<td>Low</td>
<td>0.1</td>
<td>0.1</td>
<td>0.2</td>
</tr>
</tbody>
</table>

REPAIR PARTS (MAINTENANCE SUPPLIES)

Repair parts are normally the largest single item in operating costs and include all parts chargeable to the machine. When broad averages are considered, hourly repair costs for a single machine normally follow a smooth, upward curve. Since this hourly repair cost curve starts low and gradually rises over time, hourly operating costs must be adjusted upward as the unit ages. Since repair costs are low initially and rise gradually, averaging them produces extra funds at first which are reserved to cover future higher costs.

The cost of repair parts is difficult to estimate and in any specific application, actual cost experience on similar work provides the best basis for establishing the hourly repair reserve. Equipment applications, operating conditions and maintenance management determine repair costs.
Where no historical data is available, two generic approaches are used:

The first approach is a general formula which is appropriate for large equipment such as shovels, draglines, and crushing - conveying systems. This method recognises that generally, the more expensive the asset, the greater will be the sum of its repair component costs. Thus the capital cost of the equipment is multiplied by a percentage repair cost and divided by the number of operating hours per year. Typical values for the repair percentage range from 3% (for conveyors) to 10% (for draglines, BWEs, etc.).

The second method is based on the assumption that any piece of equipment is a collection of spare parts. Although part life varies, assuming a standard operating life of 10,000 hours, it is possible to calculate the total costs of parts expected to be purchased over this life, and therefore the hourly cost.

Since the repair cost is related to the equipment capital cost, the repair parts cost is calculated by multiplying the initial capital cost by a repair factor and then dividing this by the standard operating life to get an hourly rate. This is adjusted if the number of hours is greater than the standard operating life and then further adjusted for job conditions. Table 8.19 gives examples of typical repair factor standard life and extended life multipliers which can be used to determine the appropriate maintenance supply hourly cost.

In this case, a haul truck costing R 12 000 000 running under severe conditions, with a planned 15 000 hour life, will have a repair cost given by;

\[
12 \ 000 \ 000 \times 0.2 \times 1.2 \times 1.05/10 \ 000 = R302.40 \text{ per hour}
\]

**WEAR PARTS**

Wear parts are often referred to as operating supplies or ground engaging tools (GET). They include high-wear items such as bucket teeth, ripper boots, drill bits, cutting edges, wear plates and liners, etc. (and also welding costs on booms and sticks). These are usually separately itemised as they are directly related to the ground conditions. An approximate method is to take a factor of the capital cost which is the same logic such as for maintenance supplies. Table 8.20 summarises the various wear parts factors.
Table 8.19 Repair cost estimate factors

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Repair factor</th>
<th>Job conditions</th>
<th>Extended life factor (khours)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Severe Average Moderate</td>
<td>10</td>
</tr>
<tr>
<td>Bowl scraper</td>
<td>0.20</td>
<td>1.4 1.0 0.8</td>
<td>1.00 1.10 1.25 1.35</td>
</tr>
<tr>
<td>Dozer</td>
<td>0.25</td>
<td>1.1 1.0 0.8</td>
<td>1.00 1.10 1.30 1.40</td>
</tr>
<tr>
<td>Grader</td>
<td>0.20</td>
<td>1.3 1.0 0.8</td>
<td>1.00 1.10 1.20 1.30</td>
</tr>
<tr>
<td>FEL</td>
<td>0.25</td>
<td>1.4 1.0 0.8</td>
<td>1.00 1.10 1.30 1.45</td>
</tr>
<tr>
<td>Hydraulic shovel</td>
<td>0.15</td>
<td>1.5 1.0 0.8</td>
<td>1.00 1.05 1.10 1.35</td>
</tr>
<tr>
<td>Haul truck</td>
<td>0.20</td>
<td>1.2 1.0 0.5</td>
<td>1.00 1.15 1.30 1.45</td>
</tr>
<tr>
<td>Drill</td>
<td>0.10</td>
<td>1.3 1.0 0.4</td>
<td>1.00 1.05 1.10 1.15</td>
</tr>
</tbody>
</table>

**MAJOR OVERHAULS**
Major overhauls, as distinct from routine maintenance or repair supplies, covers the cost of major component exchange or rebuild. This can be estimated as a percentage of initial capital cost or as a build up of components and their life. For example, a truck could be subdivided into engine, transmission, body, frame, electrical and so on. The cost of each of these major components (or the cost of rebuilding them) can then be estimated with the estimated life. This gives a standard cost per hour even though the actual expenditure may only occur when the damage or rebuild is implemented. A typical example of overhauls is 15% of capital cost every 10,000 hours.

**OPERATING LABOUR**
In allocating the cost of operating labour to a piece of equipment, a number of factors need to be established to determine the labour cost;

*The availability of equipment.* Unavailable mobile equipment is normally not manned, whereas large fixed or semi-mobile production
equipment is manned even when it is unavailable. For example, dragline and drills are typically manned continually even when they are on maintenance or broken down. Conversely, if there are 10 trucks in the fleet and the expected availability is 80% then normally only 8 trucks are manned. This is referred to as the “operator ratio” - the number of persons per operating shift required to man up the machines. As such, this ratio can take into account:

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Wear parts factor (Percentage of Rand million capital cost) for job conditions</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Severe</td>
</tr>
<tr>
<td>Bowl scraper</td>
<td>5</td>
</tr>
<tr>
<td>Dozer</td>
<td>8</td>
</tr>
<tr>
<td>Grader</td>
<td>4</td>
</tr>
<tr>
<td>FEL</td>
<td>9</td>
</tr>
<tr>
<td>Hydraulic shovel</td>
<td>9</td>
</tr>
<tr>
<td>Haul truck</td>
<td>4</td>
</tr>
<tr>
<td>Drill</td>
<td>3</td>
</tr>
</tbody>
</table>

**The shift roster.** To determine the operating labour requirement per shift.

**Ancillary personnel.** Equipment support personnel also needs to be incorporated. Typically, a person who is on the machine at all times even though there is only one operator required to physically operate the machine. For example, a dragline requires an operator and an electrical and/or mechanical (oiler) assistant.
**Level of absenteeism.** Although not a direct extra staff appointment is required to cover absenteeism, the level of absenteeism should be incorporated into the cost of operating labour. It can also incorporate scheduled leave days.

As an example, if a rope shovel needs two operators costing R50 000 per year, working three shifts per day, for 18 hours over 357 days per year. If the shovel is manned on service days and absenteeism runs at 20% then, the hourly cost can be calculated as:

\[3 \times 2 \times R50,000 \times 1.20/18 \times 357 = R56 \text{ per hour}\]

The cost of labour should be on a total cost of employment basis, including pay rates, bonuses plus all additional costs such as pensions, medical, insurance, etc.

**MAINTENANCE LABOUR**

The simplest method is to determine the maintenance ratio for the particular piece of equipment, which is the ratio of workshop-man hours required per machine operating hour.

These ratios can be determined by back calculation from the maintenance repair costs per machine per operating hour. The ratio changes with the duty of the machine and obviously job conditions Table 8.21 shows various maintenance ratios for surface mining equipment operating under average site conditions, with an indication of the typical parts and labour ratios.

Finally, Table 8.22 presents a summary of the hourly operating cost estimation technique described in this section. It should be noted, however, that cost estimation is not an exact science and the data presented should not be used as a final estimate, but rather as a starting point which can be expanded and modified to suit a specific application.
Table 8.21  Maintenance ratio factors (per operating hour) and parts and labour ratios

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Maintenance ratio</th>
<th>Cost distribution (%)</th>
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<tbody>
<tr>
<td></td>
<td></td>
<td>Parts</td>
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<tr>
<td>Bowl scraper</td>
<td>1.4</td>
<td>55</td>
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<tr>
<td>Dozer</td>
<td>0.5</td>
<td>70</td>
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<tr>
<td>Grader</td>
<td>0.3</td>
<td>65</td>
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<tr>
<td>FEL</td>
<td>0.8</td>
<td>60</td>
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<tr>
<td>Hydraulic shovel</td>
<td>0.7</td>
<td>70</td>
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<tr>
<td>Haul truck</td>
<td>0.7</td>
<td>70</td>
</tr>
<tr>
<td>Drill</td>
<td>0.4</td>
<td>60</td>
</tr>
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</table>
Table 8.22  Summary of operating cost estimation technique

<table>
<thead>
<tr>
<th>Basic data</th>
<th>Haul truck</th>
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<tbody>
<tr>
<td>Capital cost</td>
<td>12,000,000.00</td>
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<tr>
<td>Estimated life (hrs)</td>
<td>15000</td>
</tr>
<tr>
<td>Job condition</td>
<td>Average</td>
</tr>
<tr>
<td>Load factor</td>
<td>1.5</td>
</tr>
<tr>
<td>Engine kW</td>
<td>1500</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Operating cost data</th>
<th>Consumption or factor</th>
<th>Unit cost</th>
<th>Hourly cost</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Fuel (litres/hour)</strong></td>
<td>92.000</td>
<td>3.18</td>
<td>292.56</td>
</tr>
<tr>
<td>Lubrication (litres/hour)</td>
<td>1.000</td>
<td>7.50</td>
<td>7.50</td>
</tr>
<tr>
<td>Hydraulics (litres/hour)</td>
<td>0.600</td>
<td>4.75</td>
<td>2.85</td>
</tr>
<tr>
<td>Filter (/hour)</td>
<td>0.250</td>
<td>128.00</td>
<td>32.00</td>
</tr>
</tbody>
</table>

| Tyres                  |                       |           |             |
| Base average life      | 4200.000              |           |             |
| Service life factor    | 0.780                 |           |             |
| Tyres used             | 6.000                 | 87,500.00 | 160.26      |

| Undercarriage          |                       |           |             |
| Base cost              | 0.00                  |           |             |
| Condition multiplier   | 0.000                 | 0.00      |             |

| Repair parts           |                       |           |             |
| Extended life factor   | 1.150                 |           |             |
| Job condition factor   | 1.000                 |           |             |
| Repair factor          | 0.200                 |           |             |
| Parts costs            |                       | 276.00    |             |

| Wear parts             |                       |           |             |
| Wear parts factor      | 3.000                 |           |             |
| Wear parts cost        |                       | 36.00     |             |

| Major overhaul         |                       |           |             |
| Interval (hours)       | 10000.000             |           |             |
| Cost factor            | 0.150                 |           |             |
| Major overhaul cost    |                       | 180.00    |             |

| Labour                 |                       |           |             |
| Operating              | 1.000                 | 56.00     | 56.00       |
| Maintenance            | 0.210                 | 48.00     | 10.08       |

**TOTAL OPERATING COST** 1,053.25
MINING ENVIRONMENT, REHABILITATION AND CLOSURE

<table>
<thead>
<tr>
<th>Learning outcomes</th>
<th>Knowledge and understanding of</th>
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<tbody>
<tr>
<td></td>
<td>Surface mining and unit operation effects in the environment</td>
</tr>
<tr>
<td></td>
<td>Typical impacts and mitigation strategies applied</td>
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<tr>
<td></td>
<td>The process of IEM within mine planning</td>
</tr>
<tr>
<td></td>
<td>The general requirements of an EMP and EMPR</td>
</tr>
<tr>
<td></td>
<td>Water management, impacts and remediation</td>
</tr>
<tr>
<td></td>
<td>Content of water management plans</td>
</tr>
<tr>
<td></td>
<td>Rehabilitation planning and sequence of activities</td>
</tr>
<tr>
<td></td>
<td>Categories of pot-rehabilitation land use</td>
</tr>
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<td></td>
<td>Spoil leveling options and treatments</td>
</tr>
<tr>
<td></td>
<td>Topsoil placement and reconstruction techniques</td>
</tr>
<tr>
<td></td>
<td>Methods of establishing vegetative cover</td>
</tr>
<tr>
<td></td>
<td>Factors to consider in costing rehabilitation</td>
</tr>
<tr>
<td></td>
<td>Closure planning process and design requirements</td>
</tr>
</tbody>
</table>

Apply, calculate or predict
- The cost of rehabilitating mined land
- Dozer productivity estimations

Evaluate or design
- Situational analysis of rehabilitation requirements, based in impacts assessment for a surface strip coal mine
9.1 Environmental Protection

An environment that is not harmful to people’s health or well-being and the protection of the environment for the benefit of present and future generations is a cornerstone of a peoples’ constitutional rights. This is achieved through legislative measures that, whilst promoting economic mineral development on the one hand, prevent pollution and ecological degradation, promote conservation and secure ecologically sustainable development and use of natural resources. Mineral exploration, development, extraction and processing all have different impacts and have the potential to conflict with these essential requirements.

To ensure that the integrity of the environment remains unimpaired, it is necessary to apply an integrated approach to environmental impact management and remediation. This approach has evolved during the long history of surface mining from being a very crude, or non-existent procedure, to a highly sophisticated and successful part of surface mining operations. The main problem associated with surface strip coal mining is the large areal extent of disrupted ground surface which automatically results from mining predominantly shallow, flat lying coal seams at high volume production. Land disturbances, in terms of area affected, have grown to enormous proportions in recent years. This is due in part to the fact that as shallow and higher quality seams are fast being exhausted, attention has been directed towards mining larger tracts of lower-grade, higher cost deposits where the stripping ratio has been increasing and current operations are involved with handling more material, often from greater depths, than previously.

Historically, land damages cited against surface strip coal mining are mainly due to the destruction of surface topographies and of soil conditions that existed before mining commenced. Often, the productivity of the soil for plant growth was greatly reduced after mining. Soils that were disrupted by these operations were often chemically active and toxic, thereby becoming a source for water pollution. Strip mine spoils piles are characteristic of the mining method and, without proper attention to environmental impact management and remediation, blocks of rock occurring in the spoil made it difficult for the farm machinery to work the land. Much larger areas are also affected by unconsolidated spoil heaps and voids because these conditions affected local drainage patterns. In these areas, the processes of erosion and sedimentation were accelerated, moving spoil into water courses and streams. Finally, the highwalls of the box-cut and final void, coupled with the possibility of ground movement or collapse, represented a safety hazard where competition for land for agriculture or living space was intense.
Recognition of these problems, by both industry and government, has driven the adoption of legislation which has brought with them changes in the methods of mining, processing and beneficiation techniques. Typical of these are;

1. Topsoil removal. Topsoil is now removed from the overburden in a separate operation from the removal of other overburden horizons. Additional resources are needed to remove, segregate, store and preserve topsoil for rehabilitation.

2. Runoff control. Runoff from mines must typically meet specified limitations and Receiving Water Quality Objectives (RWQO). Often, water treatment and sediment control facilities must be constructed prior to mining, and the mining layout and configuration must be designed to facilitate the control of runoff and the prevention of pollution of surrounding waters.

3. Stream and wetlands protection. Buffer zones to protect streams and wetlands are often required to prevent intrusion of mining into protected or ecologically sensitive areas.

4. Waste disposal. Historically, mine waste was often simply dumped at the nearest convenient location. Today virtually all mine waste must meet standards for the designed and controlled disposal of the waste material following the relevant code of practice.

5. Overburden handling. Rehabilitation requirements often require consideration of overburden quality, potential pollution problems and materials types, locations within the dump and the associated handling requirements for overburden dumping. No longer can all overburden types be simply dumped in a spoil pile.

6. Processing. Discharge of water and air pollutants from processing are carefully monitored. Significant revisions can be required in these processes to ensure that the environment is protected and pollution potential minimised.

7. Revegetation. Revegetation of areas disturbed by mining was at one time left to nature. Today, it is a legislative (and as much a moral) requirement that mined areas are restored to an acceptable, self-sustaining post-mining land use.

A summary of the major environmental effects due to the various surface mining unit operations is presented in Table 9.1. Effects on soil, water, air, wildlife, and other resources must not be considered
mutually exclusive. For example, air pollution may eventually lead to water pollution and both may degrade the land. Surface mining without control may result in serious blights and hazards to public health, contaminate air and water resources, adversely affect land values, create public nuisances, and generally interfere with community development and the social fabric of the region in which it operates.

It is against this background that environmental planning has been brought to the forefront of considerations integrated into the mine planning process. The integrity of any mining project often depends as much upon the measures proposed to mitigate environmental consequences, as it does on the mining engineer’s ability to extract and produce coal at the lowest cost. Using Table 9.1 as a guide, some typical impacts of mining and the associated rehabilitation objectives on soil, water, air and vegetation are summarised in Table 9.2.

The principles of Integrated Environmental Management (IEM) are be applied to environmental management throughout the mining industry. They are often expanded to include cradle-to-grave management of environmental impacts in all phases of a mine’s life (inception, through production to closure), effective monitoring and auditing procedures, financial guarantees for total environmental rehabilitation responsibilities, controlled decommissioning and closure procedures, procedures for the determination of possible latent environmental risks after mine closure and the retention of responsibility by a mine until an exonerating or closure certificate is granted (after-care).

9.2 Integrated Environmental Management

Integrated Environmental Management (IEM) involves the integration of environmental factors into project planning, execution, and operation from inception to final closure and after-care. The objective is to develop the most cost effective, environmentally acceptable manner in which a mining project can proceed, and to gain the approval of the legislative authorities and the support of the interested and affected parties for the benefit of the developer, the economy, the people of region and the environment on which they depend. The following abbreviated definitions are used throughout;
Table 9.1 Potential mining unit operation environmental impacts

<table>
<thead>
<tr>
<th>Environmental factors</th>
<th>Exploration</th>
<th>Area dewatering, river diversions, etc.</th>
<th>Drilling</th>
<th>Blastng</th>
<th>Overburden removal</th>
<th>Haulage and transport</th>
<th>Top soil storage</th>
<th>Equipment maintenance</th>
<th>Processing</th>
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</tr>
</tbody>
</table>

Impact
- High
- Depends on site conditions
- Low
Table 9.2 Typical impacts and mitigation strategies for surface strip coal mining

<table>
<thead>
<tr>
<th>Impact</th>
<th>Mitigation and management</th>
</tr>
</thead>
<tbody>
<tr>
<td>Soil</td>
<td>To establish self-sustaining land, contoured in such a way to prevent erosion and to prevent the pollution of both surface and groundwater by contact with contaminated water or potentially contaminant spoil; creating land capable of achieving the predetermined, post-mining productivity and usage as close as possible to the pre-mining condition.</td>
</tr>
<tr>
<td>Water</td>
<td>To establish a water management system which will prevent erosion and the contamination and wastage of water resources and ensure compliance with the water quality management objectives as determined by the Department of Water Affairs for the specific catchment area.</td>
</tr>
<tr>
<td>Air</td>
<td>To prevent the dissemination of any form of airborne pollution. In this respect, rehabilitation of slimes dams and discard dumps is critical. It is important to achieve a stable and permanent cover on these dams and dumps.</td>
</tr>
<tr>
<td>Vegetation</td>
<td>To establish stable vegetation cover and habitat, capable of natural survival and propagation. To return the land to pre-mining productivity or better.</td>
</tr>
</tbody>
</table>

- EIA Environmental Impact Assessment
- EMP Environmental Management Plan or Programme
- EMPR Environmental Management Programme Report
- I & APs Interested and Affected Parties
One of the most serious deficiencies previously encountered by many mines in the planning process was to inadequately resource and allow for the time required to identify and collate baseline environmental data, as part of either the prospecting or mining authorisation process. For both, an EMPR is required, compiled according to the requirements of the Department of Minerals and Energy’s aide memoir for the preparation of EMPRs, with particular reference to the mitigation, management or counteracting of the various impacts. This aspect is considered in more detail in a following section. Using an IEM approach, integrated in this case with project planning and management, will significantly reduce problems associated with insufficient or poorly-timed environmental assessments.

IEM is a process by which all proposed actions which impact on the environment become the subject of continuous, systematic scrutiny throughout the project’s life. It is important that the environmental planning process runs in parallel with the mine planning process, since the data, decisions and results emanating from both the processes can significantly influence each other. The environment should rank equally with the technical, economic, social, political, and financial aspects of a project which have always formed part of planning. As these elements are iteratively developed, so is environmental understanding increased throughout the feasibility study phases to preparation of the final option. By the time a decision is made to proceed with the project, it should be accompanied by an acceptable, comprehensive and approved EMPR. The parallel development of the environmental management programme and the project programme is depicted in Figure 9.1. Five common phases are recognised within both programmes (1-5 in Figure 9.1), namely:

- exploration target generation
- mining project evaluation
- mine design and construction
- mine operation
- mine decommissioning and closure

The environmental planning process, as with mine planning, must begin well before the start of mining activities and continue through the entire life of the mining project. Once underway, the future viability of a mining project can largely depend upon an effective compliance strategy that combines coal production and cash-flow objectives with legislative performance goals and financial or other
guarantees required in respect of rehabilitation costs and closure. In this case, it must be borne in mind that closure may not always occur at the end of the planned life of mine, but in reality, at any point in time, and it is necessary to consider the environmental impact and rehabilitation costs associated with premature closure.

Figure 9.1 Environmental management integrated with a mining project planning methodology showing parallel links (modified after Freere, 1993)

9.2.1 Exploration Target Generation Phase

Exploration and prospecting, requires preparation of an EIA and EMPR and an acceptable rehabilitation plan. Grid drilling, aditing or trenching certainly impact on the environment, and the process should be managed as if it were a major development project. The exploration staff have a widely-defined role in addition to ore resource to reserve definition. They should continuously provide the mining and metallurgical project engineers with information on environmentally sensitive matters such as groundwater and the presence of minerals in the coal or overburden strata, which may
cause adverse impacts along the processing route. These would include pyrites, sulphides and arsenides which, in waste rock, might oxidize, leading to acid mine drainage problems.

### 9.2.2 Mining Project Evaluation Phase

Identification of the baseline environmental issues, obtained during the exploration target generation phase is augmented in this phase by input from mine design, coal processing design and other mine planning issues, in a series of iterative pre-feasibility studies. Environmental factors that were exposed during prospecting, or continue to be noted from development of the mine design are included in the pre-feasibility EIA study. In this way, no environmental issue or risk will go through to the feasibility stage without knowledge of its environmental and mine design implications.

At this stage, the pre-mining environmental and social conditions should be well defined in- and around the proposed mining area, as determined by the pre-feasibility studies. These are typically:

- interested and affected parties (I&APs)
- regional socio-economic conditions
- land ownership currently and any identified anthropological rights to the land
- land capability and current capacity
- sensitive landscapes with special scientific interest
- archaeological, cultural or historic sites of interest
- soil types
- topography
- natural vegetation and plant life
- animal life and habitats
- surface and other water courses
- air quality
- noise levels
- visual aspects
For each of the above conditions, it is necessary to identify methods and procedures that could be used to eliminate, minimise or control (mitigate) the impacts identified surrounding these issues. It is also important to flag those factors that could prevent the project from gaining the necessary mining permits, for example, rare fauna or flora, water courses, etc.

As the mining project develops into a feasibility study, prior to submission to the company’s board for approval, the full EIA is produced as a pre-cursor to the EMPR. By the time it is submitted to the Board, the EIA will have contributed to;

- the selection of the most cost-effective, environmentally acceptable approach to the project
- the acceptance of the project by the I&APs
- The data for design and for EMPs, in order to carry the project through to commissioning and the first years of operation
- the post-mining conditions after mine de-commissioning and closure

ENVIRONMENTAL MANAGEMENT PROGRAMME REPORT

The EMPR is an important document in the permitting process. In terms of Sections 9 and 10 of the Minerals Act, 1991 (Act 50 of 1991), authorisation for the project must be obtained from the Director, Mineral Development, Department of Minerals and Energy. Such mining authorisation (either a permit or a license) will only be granted on condition that (inter-alia);

- The applicant is in possession of an approved EMP
- The Director, Mineral Development, is satisfied with the manner in which the applicant intends to rehabilitate disturbances
- That the applicant has the ability and can make the necessary provision to mine optimally and safely, and to rehabilitate any disturbances

The Aide-memoir for Environmental Management Programme Reports for Proposed Prospecting and Mining (prepared by Department of Minerals and Energy in conjunction with the mining industry and the South African Agricultural Union) gives applicants
guidelines for compiling EMPRs in accordance with an established approach which is acceptable to all the regulating authorities. They specify that the EMPR should achieve the following overall objectives:

- To meet the environmental requirements and directives under the Minerals Act, No. 50 of 1991, and its regulations.
- To provide a single document that will satisfy the various authorities concerned with the regulation of the environmental impacts of mining.
- To give reasons for the need for, and the overall benefits of, the proposed project.
- To describe the relevant baseline environmental conditions at and around the proposed site.
- To describe briefly the prospecting or mining method and associated activities so that an assessment can be made of the significant impacts that the project is likely to have on the environment during and after mining.
- To describe how the negative environmental impacts will be managed and how the positive impacts will be maximised.
- To set out the environmental management criteria that will be used during the life of the project so that the stated and agreed land capability and closure objectives can be achieved and a closure certificate can be issued.
- To indicate the financial resources that will be made available to implement the EMP.

9.2.3 Mine Design and Construction Phase

Following approval, the detailed design, procurement, and construction phase begins, in parallel with the development of environmental management plans based on the EIA and EMPR. These plans are also used in the initial audit stages of design and construction, in policing contractor performance standards during project construction, as an example. Typically, specific areas would be defined for stores, soil stockpiles, construction camps, and waste dumps. Routes for vehicle movements would be defined. Protected areas such as sensitive landscapes and archaeological sites would be demarcated. Guidelines would also prescribe the early development of features such as tree lines, berms and similar long term life-of-mine features. Water management structures and
systems would be developed following a basic water management plan.

9.2.4 Operations Phase

The EMP has been established and approved at the permitting stage and as such, during operation the EMP should make provision for regular environmental audits. The first should take place within a year of commissioning and be repeated at annual intervals to review critically the effectiveness of the EMP. Often, changing technological, mining or geological conditions will demand reappraisal of optimum operating levels. Threshold limits for emissions may be tightened, new processes developed or product specifications may alter. The overall profitability of the venture may compel changes in operating parameters. The new data from the audit is compared with the EMPR and, where necessary, the environmental management plans are modified and recorded in the EMPR with the approval of the Regional Director in terms of the Minerals Act.

9.2.5 Mine Decommissioning and Closure Phase

Once a mine (or section of a mine, in the case of partial closure) reaches the end of its operating life, the decommissioning and closure phases begin. If the principles of IEM has been followed since the start of the project, closure should not present a problem. Rehabilitation should be on schedule and remaining work to closure should be minimal. The primary objective of project and environmental planning during this period is to develop strategies that will:

- ensure compliance with the approved EMPR
- minimise the impact of mine closure on the environment
- minimise future health and safety risks
- ensure sustainable economic development potential.

Decommissioning is the activity or process that begins after mineral production ceases (including processing plant production) and it ends with closure. It involves, inter alia, the removal of unwanted infrastructure, making safe dangerous excavations and surface rehabilitation with a view to minimising the adverse environmental
impacts of mining activities remaining after production. It also includes the after-care or maintenance that may be needed until closure.

Closure means that a closure certificate has been issued in terms of section 12 of the Minerals Act, 1991, and that a closure certificate provided for in section 32(2) of the Atmospheric Pollution Prevention Act, 1965, has been issued. In the case of partial closure, the environmental management issues that need to be addressed for partial closure are the same as those required for closure of the whole mine.

A closure certificate issued by the authority brings the accountability of the operator to an end. Often, however, certificates issued are only partial. For the remainder, a measure of after-care is usually necessary for which the cycle of EMP and auditing continues until a full closure certificate is issued. This is often due to significant, long-term residual impacts resulting from the mining that persist after these activities have ceased and a closure certificate has been issued. Where possible, these impacts should have been identified at least qualitatively so that they can be accommodated when the closure objectives are being defined and when the EMP is being devised. The EAI will have highlighted the major issues on which to focus and typically include, but may not be limited to;

- The potential for acid mine drainage or poor quality leachates emanating from the mine or residue deposits
- The long-term impacts on ground-water
- The long-term stability of rehabilitated ground and waste or discard dumps
- The long-term impacts arising from river diversions around the mining area, or river re-establishment

9.3 Integrated Water Management

The effective management of the water environment is an integral component of an EMP and is especially challenging for most surface-mining operations. For approval of the EMPR, the water environment must be addressed through a structured study aimed at assessing;

- the impact of hydrological aspects on the mining operation
- the impact of mining on the hydrological regimes
the probable costs associated with water supply, pit dewatering, flood protection, pollution control, and rehabilitation

how all these factors will affect the feasibility of the proposed mine; the information requirements sufficient to satisfy mine planning requirements, rehabilitation planning requirements, and permit and legal requirements.

Typical water and hydrological impacts of surface mining are given in Table 9.3 in terms of the modification caused by mining and probable impacts.

Table 9.3 Water and hydrological problems associated with surface coal mining

<table>
<thead>
<tr>
<th>Modification or mining activity</th>
<th>Cause of modification</th>
<th>Probable impact</th>
</tr>
</thead>
<tbody>
<tr>
<td>Change in the surface hydrology</td>
<td>Disruption of natural geology. Replacement of disturbed materials</td>
<td>Change in volume and rate of surface runoff. Increased sediment production. Possible high salt concentrations. Potential for acid mine drainage.</td>
</tr>
<tr>
<td>Change in the groundwater hydrology</td>
<td>Disruption of existing ground water system. Formation of a new ground water system. Possible dewatering of undisturbed areas.</td>
<td>Possible elimination of near-surface water tables. Disruption to groundwater flows. Potential pollution of groundwater flows. Potential for acid mine drainage. Modification of ground-water recharge potential.</td>
</tr>
<tr>
<td>Decrease in land productivity</td>
<td>Rehabilitation practice and change in hydrology</td>
<td>Increase in land slopes. Reduced topsoil depth. High compaction at the surface. Change in water holding capacity. Possible high salt concentrations.</td>
</tr>
</tbody>
</table>
Of the probable impacts described in Table 9.3, acid mine drainage is in all probability the main water pollution concern at any coal mining operation. At most surface strip coal mines in the Witbank coalfields this is not a major problem since the majority of spoil materials have a low pollution potential and a slight neutralization capacity. Disposal of washing plant discards is, however, associated with a very high pollution potential. When oxidised, these materials have high levels of sulphate, iron, lead, boron, arsenic and fluoride. For some seams and coalfields, the local washing plant discards and low grade coal (middlings) might constitute 10%-50% of the material removed from the pit.

A water management plan should focus on reducing the impact on the water environment to the point where water-quality objectives are satisfied. In order to meet the stated water-quality objectives, the Department of Water Affairs and Forestry may impose a limit on the mass loads of pollutants to be discharged by a surface mine. Sulphate is generally regarded as the pollutant that best reflects the impact from coal-mining operations, and surface strip coal mines usually apply for a waste-load allocation from the Department of Water Affairs and Forestry before discharging polluted water into the natural environment. The water management plan for a mine must therefore be developed to satisfy both the aforementioned criteria, i.e. compliance with both the water-quality objectives and the waste-load allocations.

The development of an environmental water management plan should focus on the following aims;

- to reduce the deterioration of water quality within the pit (in situ water management)
- to ensure that the water to be discharged complies with the water-quality requirements
- to minimise the discharges of pollutants to the natural environment.

### 9.3.1 Development of a Water Management Plan

A hydrological pre-feasibility study should provide sufficient information to define both the scope and need for a water management plan, based on the following aspects;

- **Data requirements.** The first step in determining the probable hydrological consequences of a surface mining operation is a
thorough examination of available data for the region. Field investigations should then be initiated to obtain, where possible, specific data that are not available. Basic data requirements for the groundwater and surface water investigations are:

- **General information** - geological and topographic maps; air photos; physical layout of the mine showing pits, plant facilities, haul roads, stockpiles and waste dumps; water resources map showing existing wells, surface impoundments and river systems; and water supply requirements.

- **Groundwater investigations** - piezometric pressures and groundwater flow directions; aquifer definition and hydraulic characteristics; modifications to groundwater flow by geological structures (dykes, faults, etc.); groundwater chemistry; groundwater recharge; and physical, hydraulic and chemical characteristics of the spoil and topsoil materials.

- **Surface water investigations** - general climatic conditions; long-term rainfall records; long-term evaporation records; streamflow records (if any); groundcover characteristics; infiltration/runoff characteristics of undisturbed soil profiles; infiltration/runoff characteristics of rehabilitated areas; and depth, duration and frequency of rainfall.

- **Legal aspects.** Legislation and permit requirements should be established. The main aspects to be considered are:
  - abstraction and use of public water in the mine lease area
  - control of surface water entering the mine lease area
  - disposal of polluted water produced by the mining operations
  - requirements pertaining to the quantity and quality of water discharging from the mine lease area
  - requirements related to the rehabilitation of the mine lease area, and air pollution considerations.

- **Water resources study.** The approach towards water resources evaluation will depend on the anticipated supply potential of the mine lease area, and the availability of water from existing sources. Where extensive pit dewatering might be necessary, utilisation (or disposal) of this water can have
important economic implications. Usually an investigation into one or more of the following aspects will be required;

- water availability in the region
- groundwater surface water and supply from the mine lease area and conjunctive use of surface and groundwater.

- **Flood protection study.** A flood study should be conducted to evaluate the flood potential and to design and cost the necessary flood protection works. During the life of an operation it will be necessary to continually change the location and size of the flood diversions. In general, each flood diversion system will only be designed to protect an area for part of the life of the operation. A drainage plan should therefore be developed for different phases in the life of the mine. Information obtained from the study would be used to size and cost the following;

  - flood diversions or berms around the pit, plant facilities, stockpiles, waste dumps, haul roads, and rehabilitated areas
  - pit dewatering reticulation systems
  - sediment control facilities (if any)
  - surface runoff storage reservoirs (if any)
  - river diversions (if any).

- **Water quality investigation.** A baseline water quality investigation should be conducted so that the effect of any potential changes in quality, due to mining, can be evaluated and an assessment made of possible treatment alternatives.

- **Mine water balance.** A water balance is normally required to assess the quantity of water required for the operation of the mine facilities. The water balance forms an integral part of any water management plan. An appraisal should be made of water supply requirements, water supply sources, storage requirements, and disposal and reuse of any polluted water. Polluted water sources which could be considered for re-use include water pumped from the pit, effluents from plant facilities, and runoff from stockpiles and waste dumps. An evaluation should be made of the supply potential and costs associated with treating and utilising these sources. Objectives of water resources investigations would be to;
- develop a water reticulation flow chart which identifies the quantity, quality and location of all demands, and the location and supply potential of each supply source
- storage facility requirements
- sizes and costs associated with each supply alternative
- a cost benefit analysis which would be used to establish a water supply policy for the mine.

### 9.3.2 Minimization of Pollutant Discharges

Any mine effluent discharges have to comply with the most restrictive of the effluent standards and the RWQOs. The options for the control of water-quality deterioration within the pit concentrates on the reduction of acid production, with the concomitant reduction in sulphate generation. Specific methods relating to the treatment of spoils are discussed in the following section, including application of buffering agents, fly-ash or sewage sludge.

Once the water level in a surface coal mine reaches the weathered zone, the water begins to seep towards the perched aquifer and may eventually reach a natural stream. Contamination of the perched aquifer can be reduced if the water level in the pit is kept below the weathered zone, but seepage may still occur towards a deep fractured aquifer, or at the contact between the coal seam and the overlying strata.

If the water quality is acceptable, and if a permit has been granted for the discharge of a waste load, excess water from the pit can be discharged into a stream or river. Unfortunately, this seldom happens; either the water quality is unacceptable, or the waste-load allocation does not allow for the discharge of the full pollution load. Under these conditions, various management options can be implemented to allow the discharge of excess water from the pit.

### ACTIVE TREATMENT SYSTEMS

Various chemicals, such as hydrated lime, soda ash, caustic soda, and ammonia, have been used to neutralize acid mine drainage. Of these chemicals, lime is most frequently used. Neutralization of acid mine drainage may increase the pH of the water, but chemicals are also added that increase the salinity of the drainage.
Saline mine drainage can be desalinated to acceptable salt concentrations by processes such as reverse osmosis, but which produces a brine that still requires treatment and/or disposal.

**PASSIVE TREATMENT SYSTEMS**

Passive treatment systems usually comprise a combination of anoxic limestone drains (ALDs), followed by a series of aerobic and anaerobic wetlands. The aerobic wetlands are used for the oxidation of metals, while sulphide precipitation and alkalinity production occur in the anaerobic wetlands. Anaerobic wetlands require a readily available source of carbon, such as spent mushroom compost and sewage sludge, for effective sulphate reduction.

If some of the aforementioned water-management options have been implemented and the quality of the drainage is still too poor to meet the stream water-quality objectives, the mine drainage can be used consumptively on the mine (for the suppression of dust on haul roads and in coal beneficiation plants), evaporated in constructed impoundments, or disposed in mined-out underground workings. Underground disposal is usually regarded as a short-term management option since the underground storage capacity is limited. There is also the risk associated with underground disposal that the drainage may seep out of the workings.

Excess water is frequently evaporated. If a permit is not granted for the discharge of excess polluted drainage from the mine, evaporation dams may have to be constructed to contain all the polluted water produced by the pit. The sludge remaining from evaporation dams would also still require treatment or disposal too. It must be remembered that waste loads can only be discharged if a permit has been granted by the Department of Water Affairs and Forestry.

### 9.4 Rehabilitation Planning and Costing

In this section, the concept, activities and costing associated with rehabilitation as a distinct component of the EMP will be discussed. Rehabilitation (or reclamation) in this context refers to the activities associated with returning the lands disturbed by strip mining to their former status. In particular, this section addresses methods of excavation rehabilitation, i.e. the working of the strip void, ramp roads and spoils to restore the former surface. In terms of the EMP described previously, some, if not all of this detail would be required in sections of the document dealing with;
- Detailed description of the proposed project
  - Operational phase
- Environmental impact assessment
  - Operational phase
  - Decommissioning phase
  - Residual impacts after closure
- Environmental management programme
  - Operational phase
  - Decommissioning phase and closure
  - Proposed timetable, duration and sequence
- Financial provision

Although spoil backfilling (or dumping with draglines in strip mining), grading or contouring, topsoil replacement, and revegetation are the major components of contemporaneous rehabilitation, a detailed list of all rehabilitation activities associated with the surface mining of coal is given in Table 9.4, which indicates there are a number of additional requirements to those mentioned above.

The isolated or sporadic rehabilitation activities do not lend themselves to simple characterisation. They may include such diverse operations as final mine closure, rehabilitation of box-cut spoil, or ramps. Therefore, it is necessary to describe these activities on an individual basis.

If rehabilitation is only contemplated at the end of a mine’s life, the costs would be prohibitive. Clearly, for rehabilitation to run concurrently with mining and for the whole operation to cause only the absolute minimum of disturbance to the environment as a whole, sound planning is required.

A successful rehabilitation plan can be derived from a land capability study alone and the land restored to its former productivity after mining. This does not however, take into account all the other influences of a large surface coal mine on the surrounding ecosystem. This is why an IEM approach is recommended, to minimise effects of mining on the environment through a structured assessment and to examine the effect of mining decisions on the environment as a whole.
Table 9.4 Time sequence of reclamation activities for surface strip coal mining

| During site preparation | 1. Install water pollution control measures (river diversion, sediment traps, drainage and basins, etc.).  
| 2. Clear and grub.  
| 3. Stabilise areas around temporary facilities. |
| During overburden removal | 1. Divert water away from and around active mining areas.  
| 2. Remove topsoil and store or replace directly.  
| 3. Mine, spoil and selectively place overburden strata if possible and/or necessary. |
| During coal removal | 1. Remove all coal insofar as possible.  
| 2. For the purpose of controlling post mining groundwater flows, prevent damage to the strata immediately below the coal seam. |
| Shortly after coal removal (Topsoil placement in dry season) | 1. Rough grade and contour, taking these factors into consideration;  
Specific time limit; tied to strip turnover  
Slope steepness and final land use  
Length of uninterrupted slope  
Compaction  
Reconstruction of underground and surface drainage patterns  
2. If necessary, work spoils to;  
Remove boulders  
Blind  
Treat (fertilisers, limestone, fly ash, sewage sludge, etc.)  
Fine-grade and spread topsoil taking into account;  
Land use, thickness of topsoil  
Retention and mulching  
Treat (fertilisers, limestone, fly ash, sewage sludge, etc.) |
| Immediately prior to first planting season | 1. If necessary, manipulate the topsoil mechanically -ripping, furrowing, deep-chiseling or harrowing, or constructing dozer basins.  
2. Mulching and seedbed preparation  
3. Seed and revegetate, considering the time and methods of seeding, choice of grasses and legumes in line with final post-mining land use. |

To fulfill the legal requirements, the submission of an EMP requires considerable knowledge concerning the contents of such plans and the interaction of such with the overall mining plan. The rehabilitation component of the EMP cannot be compiled in isolation, but must
draw on specific aspects of the overall EMP and mine plan to ensure that:

- The rehabilitation plan equates to the mining plan and EMP as a whole
- The true cost of rehabilitation is rigorously assessed for the various mining options analysed
- The final rehabilitation plan equates to the lowest overall mining cost that satisfies the legal and moral constraints of the mining company.

Using the planning approach described in Figure 9.1, successively greater detail is added to the rehabilitation component of the EMP as the whole mine plan proceeds from the broadbrush pre-feasibility to the detailed feasibility, operating and closure phases of planning.

For the initial broadbrush pre-feasibility study, a broad view should be taken of the natural environment in which the mine is to be situated. Consideration is given to the question of whether rehabilitation can be performed within the mining cycle and economically. This broad approach, if applied to mining in the Witbank coalfields region of Mpumulanga Province, would for instance require a soil and water inventory and study, whilst the fauna and flora, since farming has already disturbed them from their natural state, would warrant a less detailed study. However, when mining in a more ecologically undisturbed or sensitive area, more attention would have to be given to the component interactions and socio-economic influences. The amount of pre-planning capital required can thus be relatively easily estimated and included in the pre-feasibility study.

The next step should be an evaluation of the proposed mining plan to determine equipment, capital and operating costs and how they will vary as mining proceeds. The whole project can then be re-evaluated in the light of these costs as long as rehabilitation is treated in the same way as any other production parameter.

To add successively greater levels of detail to the mining plan, the environment of the mine is defined in progressively more detail. Pre-mining land use, the basis on which a rehabilitation standard is derived, can be broadly divided into four categories;

- **Arable.** Arable land has soil that is readily permeable to the roots of common cultivated plants throughout to a depth of 0.75 meters from the surface, a soil pH value between 4.0 and 8.4, electrical conductivity less than 400 mS/m, permeability of
at least 1.5mm per hour, less than 10% rocks, slope erodibility factor of less than 2.0, and a climate regime which permits economic attainment of yields of agronomic or horticultural crops, that are at least equal to the current national average of those crops. The post mining requirement is that soil depth exceeds 0.6 meter, soil material must not be saline or sodic, and the graded slope is 1:14 or flatter, based on the average erodibility factor of pre-mining arable soils.

- **Grazing.** Grazing land has soil permeable to the roots of native plants, is more than 0.25 meter thick, contains less than 50% by volume of rocks larger than 100mm in diameter, and is capable of supporting a stand of native or introduced grass species, utilizable by domesticated livestock. Post-mining requirements are that grazing land is sloped to a maximum slope of 1:3 on box-cut outslopes, final void and haul ramp sides. All other grazing land is at 1:10 or flatter, and has a minimum soil cover of 0.25 meter of suitable soil.

- **Wilderness.** Wilderness land has little or no agricultural capability by virtue of being too arid, too saline, too steep, or too stony to support plants of economic value. Its uses lie in the fields of recreation and wildlife conservation, and includes watercourses, submerged land, built-up land, and excavations. Post-mining requirement is a soil cover of less than 0.25 meter, but more than 0.15 meter of suitable soil.

- **Wetland.** Wetland is made up of vleis, swamps, marshes, peatbogs and the like. There is usually a water table present at shallow depth and wetland has a horizon that is gleyed throughout more than 50 percent of its volume. The post-mining requirement is that prior wetland should be rehabilitated to a wilderness or wetland standard, dependant on the prevailing contours.

Pre-mining land use provides the only objective basis for the establishment of the post mining capability since no cognizance is taken of past management practices, thus it forms the basis of the rehabilitation plan within the EMP. The overall goal of the rehabilitation plan is to recreate the same proportion of land classes after mining as before mining. There are some problems arising from this overall objective, namely;

- The final void will be classed as wilderness, but the total (void) area may be greater than the pre mining wilderness area
- Wetlands in low lying areas will not be immediately established

- Box-cut spoils may not be suitable for classification as arable, depending on the final (stable) spoil slope angles selected

- Ramp roads may only be rehabilitated at closure and if classified as arable land, only at considerable cost.

To compensate for these variations, the planning process is relatively flexible in terms of whatever reasonable spatial distribution of land is selected and thus can compensate to a degree for the above problems, as long as the category distribution total or proportions remain similar and minimum economic parcels of land categories (arable and grazing especially) exist.

Since each alternative plan will represent different amounts of earthmoving, grading, topsoiling, and planting, post-mining topography is one of the most important factors to be considered. Post-mining topography is approximated by applying mine-plan data, appropriate swell factors, and box-cut and final void locations to the pre-mining topography. Another key factor is the location and thickness of high-capability soils. This is particularly important in determining the location of agricultural lands, as is the location of property boundaries and limitations on certain equipment. As an example, the grade limitation or size of mechanical farming equipment may influence the location of arable land or the size of parcels of arable land. Pre-mining land use, however, may play the most important role in determining postmining land-use plans. Surface mine planners often endeavor to balance, as nearly as possible, pre-mining and post-mining areas devoted to various land uses. Although the distribution of land among various uses may not change significantly, planning skills are required in designing a site plan that spatially orients the land uses in an efficient and aesthetically pleasing manner.

When detailing the proposed rehabilitation method, this section of the EMP submission outlines how the mine proposes to achieve its rehabilitation goal and generally includes the following;

- Brief description of the mining method

- Schedule for topsoil stripping and defined horizon depths

- Topsoil immediate use (run onto spoils, stock pile or stock pile for final rehabilitation)
- Arable land slopes, (not exceeding 1:14 and with a minimum soil depth of 0,6m)
- Grazing land slopes, (not exceeding 1:10 and with a minimum soil depth of 0,25m)
- Wilderness land slopes (final void, box cut sides and in ramps), with a maximum slope of 1:3 and soil depth of 0,15m
- Surface water storage areas, showing drainage pattern into pan and compaction and sealing of pan (to prevent leakage).
- Pit width and estimated graded land form behind pit, usually a series of gently sloping parallel ridges except in the ramp regions.
- Tillage, fertilisation, liming and cropping schedules together with species selection and planting times. Details of after-care must also be included.
- The time between mining and the rehabilitation of each mining strip must be specified. Dozing can take place usually within two rows of spoil behind the pit throughout the year, but topsoil can only be placed in the dry season (to prevent compaction) and planting only in the growing season.
- The plan should specify what form of maintenance will be applied, for how long and what provision will be made in case of failure. Three years aftercare is a minimum, but longer may be required to establish the full humus content of the A horizon.

The auditing process described earlier is based on a programme of monitoring and control throughout the mining and rehabilitation stages and, in addition to the legal requirements of auditing and compliance, also provides a useful tool in assessing performance relative to initial planned costs. Some of the areas where monitoring and control pay the most dividends are:

- Topsoil stripping - from the soil horizons depths plan, add data to mining block plan to schedule stripping operations. With this schedule, optimise the distance from stripping to stock pile or direct replacement on levelled spoils.
- Selection of areas according to land capability class - arable land should be placed as close as possible to ramps and haul roads. Since this land receives most top soil, this minimises haul distances.
• Final contour plans - make use of aerial photography to plan
the levelling of spoils. The cost of the method is offset by the
savings made in the dozing costs.

• Vegetation and soil - should be monitored at the end of each
growing season to ensure that rehabilitation is successful.
Usually the organic matter improvement in the A horizon is
measured together with the vegetative cover at ground level.
Modifications to the EMP can be made (subject to regulatory
approval) if the current plan is not successful.

Finally, the decommissioning and closure phase detail of the EMP
usually includes the following items that impact specifically on
rehabilitation;

• Ramp rehabilitation - since ramps will be used up to the end of
the life of mine to produce coal out of the pit, they will pose a
rehabilitation problem. The most efficient approach to deal
with them is to leave spoils adjacent to each and push into the
ramp when it is no longer needed. This will result in a small
trough, but such a trough will provide a drainage path for storm
water into the final void.

• Final void - a closure or partial closure certificate will not be
issued unless the highwalls are made safe. This involves
profiling the spoil side to a 1:3 slope down to the water and
preventing access to (over) the highwall. The highwall may be
considered useful in terms of wildlife and will in all likelihood be
blasted down to a similar gradient under certain circumstances.

9.4.1 Approaches to Rehabilitating Mined Land

Using the case of surface strip coal mining and contiguous
rehabilitation, the steps outlined previously, namely spoiling, spoil
contouring, topsoil placement and establishing vegetation, form the
basis of this analysis of methods.

SPOILING AND CONTOURING
The overriding objective of the spoiling and contouring is to prepare
the land to support approved postmining land use. With few
exceptions, spoiling initially, and contouring thereafter must achieve
the approximate original contours that existed before mining. This
includes the elimination of all highwalls, spoil piles, and depressions.
Stable post-mining slopes must also be achieved in accordance with the specified land use (arable, grazing, wetlands or wilderness).

Minimisation of erosion and water pollution, both on-site and off-site, is another objective. Exposed coal seams, acid- and toxic-forming materials in the overburden, and combustible materials (unexploited or sub-economic seams) exposed during mining must be covered adequately. This burial process is performed to control the impact on surface and groundwater among other effects. The overburden should be tested for neutralisation potential and if large amounts of neutralising agent are required, the overburden should be considered toxic and should be suitably buried.

High acidity in overburden is usually the result of pyrite oxidisation, in the presence of air and water, producing sulphuric acid and iron (ferrous) sulphate. This toxic combination can enter streams and ditches as runoff resulting in water pollution. High acidity will also prevent the growth of trees and plants on the rehabilitated spoil areas if not buried. Some treatment techniques include;

- **Addition of Buffering Agents.** Buffering agents such as fly ash or lime can be added during the placement of spoils. The addition of lime is not always effective and the amount of lime required may therefore be significantly higher than predicted by laboratory tests alone. Power-station fly ash generates between 54 to 171 kg of CaCO$_3$ per ton of base potential, which is approximately 5 to 16% of the base potential of lime. However, although fly ash provides a cheap source of alkalinity, it also contains significant quantities of heavy metals. If the addition of fly ash is insufficient to neutralize all the acidity produced within a pit, it can lead to the mobilization of the contained metals, many of which are toxic. An excess dosage of fly ash is therefore required to compensate for the risks associated with the use of this neutralizing agent.

- **Application of Sewage Sludge.** Sewage sludge can be added to spoils to provide a source of carbon for the bacterial reduction of sulphate and the concomitant production of alkalinity. However, surface coal mines are not normally located near large sewage-purification works, and the transportation costs of the quantities of sludge required usually make this option non-feasible.

Spoiling equipment generally dictates the spoiling methods used and the extent of contouring required. In Module six, these techniques
were characterised in detail. The following are the most common methods of placing spoils;

- Dragline direct casting – with an attendant loss in dragline coal exposure rate due to selective dumping required to form a recontoured landscape. Dragline casting of spoils is also often used with ‘hot’ spoils – where thin discarded coal seams (of sub-economic value) are stripped with the overburden and later spontaneously combust.

- Bucket wheel excavator (BWE) direct casting via cross-pit or around-the-pit conveyor

- BWE in combination with a dragline where the BWE strips a portion of the (upper, soft) overburden and spoils it on top of direct-cast dragline (blasted) spoil.

- Dragline rehandle or spoil pull-back can be done with the primary dragline or a second dragline working in tandem with the primary dragline. Often used on ramp or box-cut spoils where the push distance of dozers would result in excessive costs.

Truck and shovel haulage backfilling methods are common in terrace type operations. Due to pit configuration and limited reach of the overburden removal equipment, the material must be transported some distance before it can be disposed properly. These methods can be categorised into the following two types;

- Separate load and haul units.

- Combined load and haul units such as a bowl scraper.

Some methods of spoiling do not easily lend themselves to selective placement of overburden and, even with controlled blasting and selective stripping, is not an easy matter to bury strata of overburden in the spoils.

Regardless of the spoiling method, some contouring is required. The amount of contouring, however, is dependent upon the method of spoiling. Dragline direct casting methods produce, in general, very large spoil ridges that require extensive grading. BWE spoiling produces narrower ridges than draglines, but leveling is still required. Haulage methods of spoiling generally produce a more even surface than direct casting methods, but some leveling or contouring of the surface is still required prior to topsoil placement. This contouring is performed using large track dozers, however, too much contouring
work tends to compact the upper spoil layers while not enough can cause ‘puddling’ with associated plant rot, poor drainage and ‘mud flats’ in dry weather.

The applicability of various types of equipment to contouring and spoiling was evaluated by Skelly and Loy with respect to spoil configuration, rehandle percentage, transportation distance, and final surface contours. This evaluation is summarised in Table 9.5.

Table 9.5 Equipment selection for spoil contouring

<table>
<thead>
<tr>
<th>Spoil conditions</th>
<th>Ground conditions</th>
<th>Equipment</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Dozer</td>
</tr>
<tr>
<td>Spoil configuration</td>
<td>Heavy</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Light</td>
<td></td>
</tr>
<tr>
<td>Spoil rehandle whilst contouring</td>
<td>&gt;75%</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>50-75%</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>25-50%</td>
<td></td>
</tr>
<tr>
<td></td>
<td>&lt;25%</td>
<td></td>
</tr>
<tr>
<td>Push or transport distance</td>
<td>20-50m</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>50-100m</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>100-200m</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>200-350m</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>&gt;350m</td>
<td>X</td>
</tr>
<tr>
<td>Final contours</td>
<td>Flat</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>Flat rough</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>Undulating</td>
<td>X</td>
</tr>
<tr>
<td></td>
<td>Steep, rough</td>
<td>X</td>
</tr>
</tbody>
</table>

KEY TO COLOURS

- Should consider
- May consider
- May consider under certain circumstances
- May be considered under special situation
- Not an option for this equipment
Although the rehabilitation process, when contiguous with mining, tends to follow a set pattern of operations, the strip mining method and associated rehabilitation planning needs also to consider box-cut and final void rehabilitation. There are two main methods to handle the reclamation of box-cut spoil;

- Box-cut spoil can be completely removed, possibly being used to fill the final void. This method returns the land to its original contours and helps to eliminate the remaining highwall problem. However, the cost can be high, especially if there is a long distance between the box pit and the final void. Another problem is that the spoil is rehandled and, while it may have been possible to bury acidic and rocky material during the initial placement of the spoil, following this reworking, both the spoil area and the final cut area may require a great deal more attention. The borrow pit box cut method may help in this respect as it does allow the immediate burial of rocks, etc. below final ground elevation during the original spoiling sequence.

- The second method for the box pit spoil is to contour it in-situ to form a new ground surface blended into the surrounding terrain. While this method is certainly less expensive it is not always feasible or desirable in terms of final land use. Also, if the box-cut spoil can be used to fill the final void, the overall cost of rehabilitation may be reduced, since contouring of highwalls (drill and blast to a flatter slope angle) is expensive.

The most common method used for contouring is that of dozing the spoils to the required grade and topography. This grade depends on the category of post mining land-use (arable, grazing, wilderness or wetland). An important aspect that should be considered when levelling spoils is that the dragline leaves numerous forms of spoil piles, the most common amongst these being;

- Continuous ridges. Where seam depth results in the dragline maximum spoil height remaining fairly constant and the machine operating at very nearly its maximum dumping height. This is an ideal configuration for cost-effective contouring with dozers.

- ‘Sugar piles’. Where the seam is shallow and the dragline cannot maintain the continuous ridge due to lack of overburden. In this case, isolated cones of spoil are left as the dragline proceeds along the cut. The only way to ensure a level continuous spoil ridge in this case is to increase the swing
angle behind the machine (with the attendant loss of productivity).

- A combination of the above where ramps or pit ends are encountered and spoil height increases to accommodate the ramp or pit end wall (with ramp also).

Spoils are often classified as heavy, medium and light, depending on the length of time it takes to level a hectare of the specific type of spoil pile. Table 9.6 gives a classification of the three types of spoils according to the time and cost index of leveling one hectare of spoil piles.

Table 9.6 Spoil levelling time and cost according to conditions

<table>
<thead>
<tr>
<th>Spoil classification</th>
<th>Example</th>
<th>Time$^1$ (Hours/Hectare)</th>
<th>Cost factor$^2$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Heavy</td>
<td>Spoil piles from box cuts, ramps and end of pit</td>
<td>120</td>
<td>3.4</td>
</tr>
<tr>
<td>Medium</td>
<td>Spoil piles from thick overburden or thin overburden in discontinuous ridges</td>
<td>80</td>
<td>2.2</td>
</tr>
<tr>
<td>Light</td>
<td>Spoil pile from thin overburden in continuous ridges</td>
<td>40</td>
<td>1.0</td>
</tr>
</tbody>
</table>

Notes
1. Time estimate based on contouring to grazing land-use, 40m ridge to ridge, 50m drift and 30° angle of spoil repose
2. Cost factor increase based on light spoils, Rand per hectare

Work progresses by firstly dozing a road up the spoils to reach the crest or ridge. Once in that position the dozer proceeds to push the peaks into adjacent valleys. This process is done using several methods;

- Using angle blades that move material short distances laterally towards the outslope under continuous cutting and pushing (blade full) conditions. This technique is productive if spoil geometry allows, because the blade travels parallel with the spoil bank, moving material laterally towards the outslope. The
average push distances are therefore about half the blade width. Also, the blade moves in a continuous cutting and casting action as it moves along the ridge, which implies it always works full and reversing/manoeuvering time is minimised.

- Using narrow short blades that fill in short distances and carry large volumes. This technique is employed when push distances are short and the operator reaches the dump point when the blade is not yet full. This is typically the case with sugar pile spoils.

- Using angle blades with a method called ‘slot dozing’ which involves the creation of slots parallel to each other up to a depth of approximately 2m. Productivity is significantly increased when dozing in a slot, since the slot sides act as barriers to keep the spoils being drifted within the blade area. Once the required amount of material is moved the windrows between the slots are levelled. As the work progresses, the spoil bank height decreases and the average push distance increases, usually up to 50m depending on the cut width.

The final choice of method and blade is thus dependent on the material type to be moved and its configuration in-situ, together with the final landform requirements. These aspect are discussed later in detail.

**TOPSOIL PLACEMENT AND RECONSTRUCTION**

In the EMP, a description of the soil types to be disturbed, their fertility, erodibility and depth should be provided and the soil should be mapped according to a recognised soil classification system. The dryland production potential and the irrigation potential of these soils should also be described. A soil utilisation guide is also required, based on the soil map, to show the depths of usable soil in disturbed areas which will be utilised mining rehabilitation and with it soil stockpile positions.

Equipment was reviewed in Module six in which it was seen that scrapers are widely used to remove the topsoil, particularly where the topsoil is dumped directly on the contoured, graded spoils for rehabilitation. Haulage and replacement of soil are related to the method of soil removal. Topsoil or subsoil removed by scrapers is typically hauled and replaced by the same machine. The soil is spread evenly over the site in lifts until the desired depth is achieved.
A minimal amount of grading is required in the operation; however, the number of scraper passes necessary to satisfy soil reconstruction standards is blamed most frequently for the compaction problem.

Using truck and shovels for topsoil removal is an option finding increased favour, especially where topsoils is dumped and stored prior to topsoil spreading. Dumping with trucks may reduce topsoil compaction, but it is often also necessary to doze or grade the soil in any case, which itself can over-compact the soil.

Bulk density and soil strength are two physical properties of soil that are concerns while the soil is being handled. Compaction caused by the equipment used for topsoil placement causes bulk density and soil strength to increase which will inhibit movement of air and water through the topsoil and root growth will be compromised. Therefore, it is important to minimise compaction of the topsoil while it is being reconstructed. Practical considerations with regard to topsoil handling include the following:

- The stripping and replacement of topsoil should be carried out when the topsoil is dry, and handling should be kept to a minimum. These operations should be undertaken during the dry season
- The traffic of both the earthmoving and the agricultural equipment in the area should be controlled
- The compacted traffic routes should be deep-ripped to reduce over-compaction of the topsoil and improve permeability

The topsoiling and vegetation of the spoils have two beneficial effects, (in addition to the requirements of rehabilitation in general), specifically the reduction of rainfall recharge to the spoils, and the retardation of oxygen ingress into the spoils. This is important since rainwater is saturated with oxygen and may act as a supplier of oxygen to the deeper spoils, increasing their acid oxidation potential.

Once placed, the major problem faced is to revitalise the topsoil to the level where it can sustain plant growth, initially simple cover and, once established, a carrying capacity or yield similar in excess of that of the pre-mining condition. Where little or no high quality topsoil exists, soil treatment and upgrading becomes very important. There is no universally accepted method to accomplish revitalisation. Each situation will have a different set of variables such as local topsoil quality, quality of the underlying strata and climate which will dictate the extent of ‘amendments’ or treatments required.
The volume composition of the average topsoil is generally about 50 percent solids and 50 percent void space by volume. About 50 percent of the pore space could be filled with water under conditions where free drainage occurs. Since most reactions in soils take place at or on the surface of the minerals, the fractions having the largest surface area are the most important. The coarse fragments, on the other hand, play only a small role in physical and chemical reactions but are significant with regards to controlling the size of pores or spaces between particles. The overall requirements for healthy growth are termed 'essential elements'. In general about 90 to 95% of a plant, (dry weight) is composed of hydrogen, oxygen, and carbon. The remaining 5 to 10 percent is composed of the remaining elements. All of the sixteen essential elements are necessary for plants in order for them to complete their full life cycles. Of the 'macro-nutrient elements', nitrogen, phosphorous, calcium, potassium, magnesium and sulphur are required in large amounts by many crop plants. The balance, 'micro-nutrients', are required in much smaller or trace amounts whilst carbon is obtained from CO$_2$ in the atmosphere, hydrogen and oxygen comes from water and the remainder are obtained from the soil. Table 9.7 summarises a typical revitalization program for topsoil, typical of the Witbank coalfields area.

Table 9.7  Fertilisation rates for topsoil prior to first seeding

<table>
<thead>
<tr>
<th>Fertiliser</th>
<th>Application rate (kg/ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dolomite lime</td>
<td>4000</td>
</tr>
<tr>
<td>Superphosphate</td>
<td>1000</td>
</tr>
<tr>
<td>2.3.4 (Nitrogen, phosphorous and potassium ratio)</td>
<td>600</td>
</tr>
<tr>
<td>Limestone ammonium nitrate</td>
<td>350</td>
</tr>
</tbody>
</table>

ESTABLISHING VEGETATIVE COVER
A detailed review of methods and techniques of establishing cover are beyond the scope of this Module. However, some basic principles apply, irrespective of local conditions. Revegetation operations are planned in light of the approved post-mining land use. Many considerations such as species selection, soil treatments, and methods of planting are related specifically to the land use. Other
considerations such as soil stabilisation, water infiltration, and compaction are more independent of the end use.

Chemical treatments of the topsoil can be regarded in two phases;

- Primary treatment phase, which involves the preparation of the soil for the establishment of vegetation, as in Table 9.7.
- Maintenance treatment phase, in which the soil nutrients are supplemented above a pre-determined critical level. This would also be used as the minimum level in a soil-nutrient monitoring or audit programme of the EMP. Maintenance fertilisation rates are site-specific and based on nutrient losses through grazing, mowing and baling, ammonification, run-off and leaching.

Topsoils preparation for seeding consists of two phases;

- **Primary tillage.** Primary tillage penetrates deeply, breaking the soil into large clods with an uneven surface.
- **Secondary tillage.** To even-out the surface, secondary tillage is performed, usually with some type of harrow to produce a suitable seedbed.

Although the establishment of a vegetative cover has several obvious and accepted advantages, the challenge is to maintain the sustainability of the system in the long term. A healthy, vigorous cover assists in the following ways;

- It protects the limited (and often limiting) topsoil cover by reducing erosion both by water and wind.
- It re-establishes an agriculturally productive area in the form of artificial pastures. The merits of this ‘advantage’ are discussed later in regard to maintenance and utilization.
- It builds up the organic matter in the replaced topsoil through the decomposition of the root mass.

Seeding takes place once the moisture content of the soil has been replenished by summer rains. A seed mix consisting of Rhodes (Chloris Gayana), Smuts (Digitaria Smutsii), Lucern (Medicago sativa) and a grass species is recommended.
Sativa) and Teff (Eragrostus Teff), is often used in the Witbank coalfields region to give an early grass coverage and to prevent any soil erosion, thus thereafter a lasting coverage to establish the organic and micro-biological conditions for sustained growth. The seed mix should maximise the retention of water in the upper layers of organic rich soil, which then acts like a sponge, preventing excessive seepage to lower levels. Table 9.8 is an example of a typical seed mix and their respective application rates.

Table 9.8 Typical Witbank coalfields region seed types and application rates

<table>
<thead>
<tr>
<th>Seed species</th>
<th>Application rate (kg/ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cynodon Dactylon</td>
<td>4</td>
</tr>
<tr>
<td>Digitaria Smutsii</td>
<td>6</td>
</tr>
<tr>
<td>Chloris Gayana</td>
<td>2</td>
</tr>
<tr>
<td>Eragrostus Curvula</td>
<td>2</td>
</tr>
<tr>
<td>Eragrostus Teff</td>
<td>1</td>
</tr>
<tr>
<td>Lucern</td>
<td>3</td>
</tr>
<tr>
<td>Ladino Clover</td>
<td>1</td>
</tr>
<tr>
<td>Lotus Coniculatus</td>
<td>1</td>
</tr>
<tr>
<td>Oats</td>
<td>4</td>
</tr>
<tr>
<td>Triticale Rye</td>
<td>2</td>
</tr>
</tbody>
</table>

The re-vegetating of spoils should include a regular, disciplined after-care programme of maintenance and utilisation. It is not generally acceptable that the mine walks away after two to three years of superficial maintenance. Specifications for the final land use are formally established in the EMP to ensure that the prescribed management programme (maintenance and utilisation) is followed.
This is the ethical cut-off point in transferring the ownership of the liability for surface rehabilitation on the closure of a mine. The after-care programme usually comprises supplementation of the soil nutrients until the monitored levels have stabilised above a sub-critical level, and a pasture-utilization programme with the primary objective of maintaining the vigour of the sward by defoliation. The alternatives to defoliation include fire, grazing, or ideally slashing and baling of the hay, which can be sold to offset the costs of maintenance fertilization.

### 9.4.2 Costing the Rehabilitation of Mined Land

The cost of rehabilitating mined land is a function primarily of the volumes of material to be moved and the final contours required, productivity of the equipment and the operating cost. Costs are also incurred in seeding and maintenance, including cutting and bailing the re-established cover. Table 9.9 summarises a typical cost breakdown for rehabilitation.

#### Table 9.9 Typical cost breakdown for rehabilitation

<table>
<thead>
<tr>
<th>Activity</th>
<th>Percentage of total rehabilitation cost/hectare</th>
</tr>
</thead>
<tbody>
<tr>
<td>Contouring spoils</td>
<td>70</td>
</tr>
<tr>
<td>Topsoil replacement and working</td>
<td>18</td>
</tr>
<tr>
<td>Preparation of topsoil seedbed</td>
<td>9</td>
</tr>
<tr>
<td>Annual maintenance after 1\textsuperscript{st} season growth</td>
<td>3</td>
</tr>
</tbody>
</table>

**VOLUMES OF MATERIAL TO BE MOVED**

An estimate of the volume of material to be moved can be made from consideration of the spoil geometry, the quantity of material to be moved (LCM), being dependent on:

- type of spoil bank
- crest to crest distance
- angle of repose of material
- amount of dozer rehandle

In this case consider a continuous ridge of spoils and assuming an average push distance of 35% of the crest to crest distance, a theoretical estimate can be made of the total amount of LCMs to be moved under various conditions, as summarised in Table 9.10. Figure 9.2 shows the spoil geometry used for the estimation.

Work progresses by firstly dozing a road up the spoils to reach the crest. Once in that position, the dozer then pushes the peaks into adjacent valleys. As the work progresses, the spoil bank height decreases and the push distance increases as a function of the crest to crest distance. The average push distance is usually estimated at 35% of the crest to crest distance, and the number of dozer pushes at this distance determined from blade cut depth and the vertical depth of cut to the required grade.

Table 9.10  Theoretical LCMs per hectare to be handled in normal spoil

<table>
<thead>
<tr>
<th>Spoil center to center line distance (m)</th>
<th>30</th>
<th>40</th>
<th>50</th>
<th>60</th>
</tr>
</thead>
<tbody>
<tr>
<td>Spoil angle of repose(°)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rehandle (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>30</td>
<td>0</td>
<td>10,825</td>
<td>14,434</td>
<td>18,042</td>
</tr>
<tr>
<td></td>
<td>25</td>
<td>13,532</td>
<td>18,042</td>
<td>22,563</td>
</tr>
<tr>
<td></td>
<td>40</td>
<td>15,155</td>
<td>20,207</td>
<td>25,259</td>
</tr>
<tr>
<td>33</td>
<td>0</td>
<td>12,176</td>
<td>16,235</td>
<td>20,294</td>
</tr>
<tr>
<td></td>
<td>25</td>
<td>15,220</td>
<td>20,294</td>
<td>25,367</td>
</tr>
<tr>
<td></td>
<td>40</td>
<td>17,047</td>
<td>22,729</td>
<td>28,412</td>
</tr>
<tr>
<td>36</td>
<td>0</td>
<td>13,623</td>
<td>18,164</td>
<td>22,704</td>
</tr>
<tr>
<td></td>
<td>25</td>
<td>17,028</td>
<td>22,704</td>
<td>28,381</td>
</tr>
<tr>
<td></td>
<td>40</td>
<td>19,072</td>
<td>25,429</td>
<td>31,786</td>
</tr>
</tbody>
</table>
For ‘sugar-pile’ types of spoil dumping, the push takes place around the spoil pile and the vertical distance from top of spoil crest down to final gradient is greater than for continuous ridges. As an estimate, this increases the LCM per hectare by a factor of 1.9 to 2.8, depending on spoil angle of repose and spoil line center to center spacings.

**DOZING PRODUCTIVITY IN REHABILITATION**

An average dozing cycle in levelling spoil consists of three phases;

- loading the blade (20% of cycle time)
- drifting (transferring the material) (50% of cycle time) and dump
- returning (30% of cycle time)

The primary factors that effect both the cycle time and productivity are;

- Push distance
- Grade
- Material type (Heavy blocky, medium or light spoils)
- Operator skill (and use of computerised earthmoving equipment survey system with GPS to control final levels according to contour plan)

- Type of blade used

- Type of dozing (slot dozing is the ideal approach in spoils)

- Visibility

The machine manufacturers usually have an extensive productivity data base which can be used to estimate dozer productivity under specific working conditions. Table 9.11 summarises the correction factors to be applied when estimating dozer productivity.

Dozer basic productivity can be estimated by using the productivity curves for med-large dozers presented in Figure 9.3 or alternatively, from field studies.

The following empirical method can be used to estimate bulldozer production in the field:

- **Dozer operation.** Pick up and drift load onto a level area and stop. Raise the blade directly over the pile pulling forward slightly as blade comes up, leaving a nearly symmetrical pile. Reverse to clear the pile.

- **Measurements.** Measure the average height (H) of the pile, the average width (W) of the pile and the greatest length (L) of the pile.

- **Calculation.** Blade load (LCM) = 0.0138 (HWL)

Productivity can be estimated from field time studies in which an average dozer cycle time is computed, comprising load, drift and dump, return.
Table 9.11  Dozer productivity correction factors

<table>
<thead>
<tr>
<th>Job condition</th>
<th>Correction factors</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Operator</strong></td>
<td></td>
</tr>
<tr>
<td>Excellent</td>
<td>1.00</td>
</tr>
<tr>
<td>Average</td>
<td>0.75</td>
</tr>
<tr>
<td>Poor</td>
<td>0.60</td>
</tr>
<tr>
<td><strong>Material</strong></td>
<td></td>
</tr>
<tr>
<td>Loose stockpile</td>
<td>1.20</td>
</tr>
<tr>
<td>Hard to cut</td>
<td>0.60-0.80</td>
</tr>
<tr>
<td>Hard to drift; “dead” (dry, non-cohesive material) or very sticky material</td>
<td>0.80</td>
</tr>
<tr>
<td>Spoil, ripped or blasted</td>
<td>0.70-0.80</td>
</tr>
<tr>
<td>Large boulders in spoil, few fines</td>
<td>0.60</td>
</tr>
<tr>
<td><strong>Slot dozing</strong></td>
<td>1.20</td>
</tr>
<tr>
<td><strong>Side by side dozing</strong></td>
<td>1.15-1.25</td>
</tr>
<tr>
<td><strong>Visibility, night, or impaired visibility</strong></td>
<td>0.8</td>
</tr>
<tr>
<td><strong>Grade of operation</strong></td>
<td>1.00 + 0.2/% downgrade</td>
</tr>
<tr>
<td></td>
<td>1.00 - 0.2/% upgrade</td>
</tr>
</tbody>
</table>

**Notes**
Estimation factors based on data in Figure 9.3 and:
1 100% efficiency and utilisation
2 Power shift machines; dig 1F, drift 2F, return 2R (slot dozing will require drift in 1F, but productivity should equal or exceed standard conditions due to the larger loads that can be carried in 1F)
3 Machine cuts for 15 m, then drifts blade load to dump over a edge of spoil slope
4 Spoil density of 1.37t/LCM (apply ratio correction for other loose densities)
5 Coefficient of traction 0.5 or better
6 Hydraulic controlled tilt blades used – see blade selection comments

**BLADE CHOICE FOR RECONTOURING**
The combination of blade type and push distance is an important consideration when optimising dozer productivity. From Figure 9.3 it is seen that the smaller the push distance, the higher the productivity, until the blade is full and the operator dumps the load. This is referred to as the blade full point and the closer the operator works to this point, the higher is the production. If the push distance is less than this, productivity will reduce since the blade is not full.

Blade choice is dependant on a number of material and machine limitations, primary amongst these being;
Figure 9.3  Production rates as a function of push distance for medium-large track dozers

- Particle size distribution, shape, void ratio and water content
- Blade load (kW per meter of cutting edge) is an indication of a blade’s ability to penetrate and load. The higher the kW/meter, the more aggressive the blade.
- Push load (kW per LCM) is an indication of a blade’s ability to push material. The higher the kW/LCM, the greater the blade’s capability for drifting material at a higher speed.
- Machine limitations. Usable push is determined from the machine weight and its power train. Poor traction conditions (slip) limit the dozer’s ability to use its weight and power.

Three of the most common blade types are described below; the universal (U), semi-universal (SU) and straight blade (S). Blade capacities ($V_s = \text{capacity of straight blade}$, $V_u = \text{capacity of SU or U-blade}$) as determined by SAE recommended practice J1265 are given by;

$$V_s = 0.8WH^2$$
\[ V_u = V_s + ZH(W - Z)\tan \theta \]

Where,

- \( W \) = Blade width (m).
- \( H \) = Effective blade height (m).
- \( Z \) = Wing length at ground line of cutting edges (m).
- \( \theta \) = Wing angle (degrees).

- **U blade.** Large side wings on this blade make it efficient for moving big loads over long distances as in spoil contouring work. The blade has a low kW/meter of cutting edge and kW/LCM compared with a S or SU and is not designed to penetrate and is ideally suited to light spoils or relatively easily dozed material. If equipped with tilt cylinders, its ability to ditch, pry out, and level is extended.

- **SU blade.** As with the U-blade, capacity is increased by the addition of short end wings, giving improved load retention capabilities while maintaining the advantage of the S-blade’s ability to penetrate and load quickly in consolidated spoils.

- **S blade.** This blade physically smaller than the SU or U-blade, but has a higher kW/meter of cutting edge and higher kW/LCM than the SU or U-blade; consequently, the blade is more aggressive in penetrating and obtaining a blade load.

**OPERATING COST ESTIMATION FOR RECONTOURING**

Operating costs estimation techniques have been discussed in the previous Module for a range of equipment including track dozers. Data pertaining specifically to dozer operation in spoil recontouring includes fuel consumption which can range between 110-140 litres/hour, undercarriage costs, in which a basic factor of between 35-50 applies, a repair parts basic factor of 60-72 and a wear parts cost of typically R15 – R45 per operating hour.

Available time for dozers varies considerably with age and severity of application and quality of maintenance, etc. As a general guide, availability should be in the region of 75-90% whilst utilisation (of available operating time) typically 70-80%.
9.5 Mine Closure

The previous sections of this Module have introduced the scope and planning required for the development of an approved EMP, which includes aspect pertaining to closure planning. Normally, this aspect is addressed only superficially as it is perceived that the degree of confidence surrounding any closure plans so far into the future, does not warrant detailed planning. However, when possible closure at any point in time requires evidence of enough financial reserves or guarantees in place to cover decommissioning and closure costs up-front, planning many years prior to the anticipated ending of operations is useful. The owner of a mine is not relieved from any responsibility regarding the post-mining safety of the site or long-term pollution control, particularly if such pollution extends beyond the boundary of the mine and affects adjacent land users.

Motivating factors for early intervention in closure planning include;

- Good corporate governance dictating that mines practice responsible environmental management. This includes adequate planning for decommissioning and closure.
- Financial policy and the need to reduce liabilities. Adequate planning to reduce overall closure liabilities enables appropriate measures to be implemented during the operational phase of the mine.

As a result of variations in each mine’s;

- remaining life
- extent of existing operations
- socio-economic setting

A prescriptive approach, specifying rigid closure requirements, is not always appropriate. As an example, in many instances the closure of a mine leaves a region with little or no economic base and with ‘ghost towns’ or depressed communities. Closure requirements can also be influenced by the remoteness of the site, with, for example, discards or fines dumps not being revegetated as a result of the fact that dust is not considered to be a significant impact.

9.5.1 Closure Planning Process

The starting point in the closure planning process is the formulation of closure objectives. These will form the basis of conceptual planning,
during which time the objectives may change as needs become better defined. Detailed planning should only commence once clear, well formulated objectives which meets the requirements of all stakeholders have been defined. At this stage all impacts need to be identified and the need for remediation adequately understood so that alternative remediation options can be costed and evaluated in detail.

The acceptability of any decommissioning and closure plan will depend on the process of participation that was followed in its compilation. The nature of surface strip coal mining is such that there will always be residual impacts in a region broader than that defined by the mine boundaries. Public consultation and liaison with the authorities throughout the planning process is therefore essential.

The closure planning process consists broadly of the following elements:

- A review of mining and environmental data, I&APs and the socio-economic setting of the region in which the mine is located
- A closure impact study based on the issues defined above
- Development of a closure strategy, based on the impact study
- Initiate operational changes to align the mine with the stated closure strategy and objectives
- Initiate a closure strategy review process every two to five years
- Formulation of a decommissioning plan two to five years before the planned run-down of mining operations
- Decommissioning
- Post decommissioning monitoring
- Closure.

This process will ensure that adequate attention is paid to all possible mining impacts, risks are identified and mitigation measures planned and costed and that closure principles are adhered to. The potentially greatest costs are frequently associated with the least well defined or quantified impacts. These most often include;
Water pollution potential, in the case of closure this could mean 'treatment in perpetuity'

Contamination of soil, the removal and replacement of extensive areas of such soil.

Decommissioning and closure, if successfully managed, will satisfy the general principles of:

- Safe and stable remaining structures
- All legal and corporate requirements complied with
- Minimisation of negative socio-economic impacts associated with closure
- Mitigation of adverse environmental impacts
- Provision of a productive post-closure land use and, if possible, an economic base to replace mining as a source of income.
<table>
<thead>
<tr>
<th>Learning outcomes</th>
<th>Knowledge and understanding of</th>
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<td>Mine development phases and planning input required</td>
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<td>Surface strip coal mine development phases and cyclic approach</td>
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<td>Components of an integrated mine plan</td>
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<td>Approach to limits assessment and block ranking for conceptual and pre-feasibility plans</td>
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<td>Levels and order of mine planning studies</td>
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<td>Difference between planning and scheduling time frames</td>
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<td>Key factors in production scheduling</td>
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<td>Stripping sequencing (maximum, minimum, constant, phased and max-min)</td>
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<td><strong>Apply, calculate or predict</strong></td>
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<td>Present value of waste and ore mining sequences</td>
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<td></td>
<td>Stripping ratio as instantaneous and overall for various stripping sequences</td>
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<td></td>
<td>A suitable stripping schedule and ratio based in max-min stripping curves</td>
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<tr>
<td><strong>Evaluate or design</strong></td>
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<td></td>
<td>A planning process used to establish a new mine</td>
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<td></td>
<td>The data requirements for various levels of mine planning study</td>
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<td></td>
<td>Relate mine development phases to cost influences</td>
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</table>
10.1 Introduction to Planning and Scheduling

This module introduces surface strip coal mine planning and scheduling. Mine planning is the engineering process that converts a resource into an economically mineable reserve, with the overall aim of extracting maximum value from the resource. Mine planning generally assumes the role of mining method selection and integration of all the diverse sub-systems required. Accordingly, it is common for the mine planning function to be the one which combines the technical evaluations, cost schedules and financial evaluations for the entire project.

A mine planning exercise involves the compilation and integration of all relevant deposit, infrastructure, legal, financial, regulatory, mining sub-systems engineering and economic data into a single document to define and describe the exploitation strategy for a particular coal deposit. Mine planning is an iterative and continuous process which starts at the conceptual stages of a new mining project and continues through to decommissioning and closure.

Design applies to the traditionally held engineering-based design of the various sub-systems which combine to form the mining operation as a whole. The design approach is structured to allow a rigorous procedure of review, based on the evaluation of alternative options. An optimum design for any mining sub-system is one that has considered the effects of each sub-system on all others and optimises the objectives of the whole system. Figure 10.1 highlights the process of design, within the mine planning system.

Mine scheduling is the process of calculating what is happening in a mine over time. Mine scheduling is the most calculation intensive part of the mine planning process, and probably the principal use of most computerised planning systems. It is a process of assigning equipment or operations to be undertaken on the units (or 'blocks') of coal or waste in the mine, determining the time taken for that equipment to undertake the operation, and reporting on it. The aim is to produce a tabulation of quantities, values, qualities etc on a time basis. At the most primitive level, scheduling may consist of simply stepping through a mine plan knowing only the year by year quantity of coal required to be produced - calculating simultaneously the quantities of waste which need to be excavated. At a more detailed level, it may consider specific information and constraints (e.g. coal qualities) within which the mine needs to operate, usually in a much shorter time frame (month by month as opposed to year by year).
A complete mine planning exercise generally includes an assessment of:

- **the technical component**, concerning the analysis of various operating requirements based largely on technical criteria. This component of the planning exercise is typically unchanged whether the project is viable or uneconomic. The technical component defines all of the important elements concerning the implementability of the project.

- **the economic component**, concerning the analysis of various options within the mine plan in terms of operating and capital costs. The object of this component of the project is to allow the evaluation of options as objectively as possible. Its aim is to build up all of the necessary information to enable the viability of the project to be ascertained in economic terms. Additionally, this component could also address a single component or sub-system activity in the mine plan - aiming to find the lowest cost approach to that activity.
the financial component, which aims to make decisions in absolute terms of the viability of a project. In addition, this component of the project examines the relative risk associated with an investment decision, including its sensitivity to internal project and external factors and the probability that what is planned is actually achievable.

Table 10.1 lists some of the typical data requiring consideration in a mine planning exercise, from which it is clear that a structured or systematic approach to mine planning is needed, since it would not be cost-effective to gather and analyse all the data listed immediately, nor at the level of accuracy and reliability required for a bankable feasibility study. There is much data that can be inferred or assumed early in the life of a project and, if the project is evaluated as probably feasible, more resources spent on better defining the data, such that when a full feasibility study is undertaken, there is a good chance of the project developing into an operating mine. It should also be possible to highlight those issues which, due to lack of defining data, may contribute more risk to a project than others and earmark them for a more detailed study as the plan moves from initial concept to feasibility levels. Figure 10.2 illustrates this concept.

Figure 10.2   Stages of mine planning and associated level of data confidence (modified after Fourie et al, 2001)
Table 10.1 Typical data requirements in mine planning studies

<table>
<thead>
<tr>
<th>Information on Deposit</th>
<th>General Project Information</th>
<th>Development and Extraction</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Geology: Overburden</strong></td>
<td></td>
<td></td>
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<tr>
<td>b. Geologic structure</td>
<td>b. Customers</td>
<td>a. Surface and coal</td>
</tr>
<tr>
<td>c. Physical properties (highwall and spoil characteristic degree of consolidation)</td>
<td>c. Product specifications (tons, quality)</td>
<td>b. Isopach development.</td>
</tr>
<tr>
<td>d. Thickness and variability</td>
<td>d. Locations</td>
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<tr>
<td>e. Overall depth</td>
<td>e. Contract agreements</td>
<td></td>
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<tr>
<td>f. Topsoil parameters</td>
<td>f. Spot sale considerations</td>
<td></td>
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<tr>
<td></td>
<td>f. Preparation requirements</td>
<td></td>
</tr>
<tr>
<td><strong>Geology: Coal</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>a. Quality (rank and analysis)</td>
<td>a. Property access</td>
<td></td>
</tr>
<tr>
<td>c. Variability of chemical characteristics</td>
<td>b. Accesorries</td>
<td>b. Economics</td>
</tr>
<tr>
<td>d. Structure (particularly at contacts)</td>
<td>c. Use of water in operations</td>
<td></td>
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<tr>
<td>e. Physical characteristics</td>
<td>d. Cost</td>
<td></td>
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<tr>
<td><strong>Hydrology</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>a. Permeability</td>
<td>a. Availability</td>
<td></td>
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<tr>
<td>b. Porosity</td>
<td>b. Location</td>
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<tr>
<td>c. Transmissivity</td>
<td>c. Right-of-way</td>
<td></td>
</tr>
<tr>
<td>d. Extent of aquifer(s)</td>
<td>d. Cost</td>
<td></td>
</tr>
<tr>
<td><strong>Geometry</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>a. Size</td>
<td>a. Land and Mineral Rights</td>
<td></td>
</tr>
<tr>
<td>b. Shape</td>
<td>a. Ownership (surface, mineral)</td>
<td>a. Topography</td>
</tr>
<tr>
<td>c. Attitude</td>
<td>b. Area requirements (onsite, offsite)</td>
<td>b. Geotechnical factors</td>
</tr>
<tr>
<td>d. Continuity</td>
<td>c. Location of boresoles, SSSI, etc.</td>
<td>c. Production requirements</td>
</tr>
<tr>
<td><strong>Geography</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>a. Location</td>
<td>a. Water</td>
<td></td>
</tr>
<tr>
<td>b. Topography</td>
<td>a. Potable and preparation</td>
<td>a. Extent of pit area</td>
</tr>
<tr>
<td>c. Attitude</td>
<td>b. Sources</td>
<td>b. Pit orientation</td>
</tr>
<tr>
<td>d. Climate</td>
<td>c. Quantity</td>
<td>c. Services</td>
</tr>
<tr>
<td>e. Surface conditions</td>
<td>d. Quality</td>
<td>d. Pit dimensions and geometry</td>
</tr>
<tr>
<td>f. Drainage patterns</td>
<td>e. Costs</td>
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<tr>
<td>g. Political boundaries</td>
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<tr>
<td><strong>Exploration</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>a. Historical (area, property)</td>
<td>a. Labour</td>
<td></td>
</tr>
<tr>
<td>c. Sampling (types, procedures)</td>
<td>b. Rates and trends</td>
<td>b. Mine support equipment</td>
</tr>
<tr>
<td><strong>Regulatory</strong></td>
<td>c. Degree of organization</td>
<td>c. Office, shop, and other facilities</td>
</tr>
<tr>
<td>a. Taxation</td>
<td>d. Labour history</td>
<td>d. Auxiliary facilities</td>
</tr>
<tr>
<td>b. Royalties</td>
<td></td>
<td>e. Manpower</td>
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<tr>
<td>c. Reclamation</td>
<td></td>
<td></td>
</tr>
<tr>
<td>d. Zoning</td>
<td></td>
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</tr>
<tr>
<td>e. Mining legislation</td>
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<tr>
<td><strong>Environmental</strong></td>
<td></td>
<td></td>
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<tr>
<td>a. Regional setting</td>
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<tr>
<td>b. Regulatory requirements</td>
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10-5
Surface mine planning and scheduling relies heavily on the use of computerised tools, from determining the nature and grade of a coal reserve, through mining method simulation and scheduling, to the final economic analysis. However, before a computer model is developed, and a complete mine designed and scheduled according to this model, some basic planning phases and design parameter inter-relationships need to be understood.

10.2 Mine Development Phases

Mine development typically consists of the following five unique and identifiable phases;

- target generation for exploration and investigations
- conceptual, pre-feasibility and bankable feasibility studies, comprising planning, design and evaluation stages. The ‘planning’ phase
- construction and mine establishment
- mining operations
- mine decommissioning and closure.

10.2.1 Target Generation Phase

The target generation phase has been previously discussed in an earlier module, in terms of the approaches to exploration and associated data requirements. The coal supply process, shown diagrammatically in Figure 10.3 is the starting point. A positive change in the market place creates a new or increased demand for coal. In response to the demand, financial resources are applied in an exploration phase resulting in the discovery and delineation of the coal seams. Through increases in price and/or advances in technology, previously located deposits may become interesting.

Within the target generation phase, the job of the geologist is to supply a representation of the orebody - initially making little or no judgments as to economic viability or potential mining constraints until these are examined later. Detailed geological evaluation is clearly directed at the most economic targets, however, the decision on which economic criteria to use is left for the mine planning process. This evaluation process will be termed the ‘planning phase’ of a project. Figure 10.4 summarises the target generation phase.
Figure 10.3 Coal supply process

Figure 10.4 Exploration sequential decision-making process (modified after Bailey, 1968)
The process of resource estimation consists of:

- resource data collection
- resource model derivation and validation
- resource estimation and classification

A 3D solid model is usually developed to clarify the quality distributions, limits and volume available for possible economic extraction. The solid model is used to construct a block model utilising various block sizes according to the level of study being undertaken, for simply outlining the ore body, for calculating the block values, for designing the mine or for detailed scheduling requirements.

10.2.2 Planning Phase

The planning phase offers the greatest opportunity to minimize the capital and operating costs of the ultimate project, while maximizing the operability and value of the venture. But the opposite is also true; no phase of the project has the potential for instilling technical or fiscal disaster into a developing project than as in this phase. If the initial planning is incorrect or otherwise technically flawed, it is extremely difficult to right these problems when the mine moves forward from a plan to an operation. Figure 10.5 is a time line showing the relationship of the different phases and their stages.

At the start of the planning phase, there is a relatively unlimited ability to influence the cost of the emerging project. As decisions are made, correctly or otherwise, during the balance of the planning phase, the opportunity to influence the cost of the project diminishes rapidly. The ability to influence the cost of the project diminishes further as more decisions are made during the design stage. At the end of the planning period there is essentially very little opportunity to influence costs.

As alluded to in Figure 10.2, the planning process should incorporate increasing complexity of design and increasing reliability of data, these factors being optimised to give the best return on the capital that is invested, at the lowest technical risk. These results of the design stage are then judged and compared with alternatives. The whole process is one of systematic repetition of the same analysis, but on a more accurate and reliable basis.
10.2.3 Construction and Mine Establishment Phase

The construction and mine establishment phase consists of two stages;

- The construction stage includes the design, procurement and construction activities. Since it is the period of major cash outflows for the project, economies generally result by keeping the time frame to a realistic minimum and identifying key lead-time items in the critical path to mine commissioning.

- The second stage is commissioning. This is the trial operation of the individual sub-system components to integrate them into an operating system and ensure their readiness for start-up. It is conducted without feedstock or raw materials. Frequently the demands and costs of the commissioning period are underestimated.
10.2.4 Mining Operations Phase

The operations phase also has two stages;

- The start-up stage commences at the moment that feed is delivered to the plant with the express intention of transforming it into product. Start-up normally ends when the quantity and quality of the product is sustainable at the desired level. There is also a pre-production start-up period, often associated with waste stripping to expose the ore prior to ore mining. This can vary in length considerably as a function of the eventual stripping schedule developed.

- Operation commences at the end of the start-up stage.

Changing technology and market conditions invariably require many changes to the operational mine plan, throughout the mine life. This requires that the mine plan is not rigidly designed for any optimum case but is designed to handle the flexibility required by changing conditions. For these changes to be rigorously assessed, the establishment mine plan should be detailed enough to allow review and benchmarking – particularly as regards knowledge management within the process.

10.2.5 Mine Decommissioning and Closure Phase

Once a mine reaches the end of its operating life, the decommissioning and closure phases begin. Decommissioning is the activity or process that begins after coal production ceases (including processing plant production) and it ends with closure. Even in the initial planning stages of a project, and certainly in the on-going production planning during operations, planning for closure must take place. The starting point in the closure planning process is the formulation of closure objectives and these must be incorporated into the operations phase, otherwise excessive costs will result prior to eventual closure.
10.3 Steps and Phases Involved in a Mine Planning Study

The systematic mine planning methodology described here is adapted from an approach developed and described by Runge Mining (Australia, 1991). It is characterised by a staged development, from a basic to advanced plan and highlights the associated level of decision making warranted at each stage.

The mine design selected from the mine planning process as the optimum is one which yields the highest return on investment for the lowest risk. In other words; the aim is to;

- Minimise costs
- Minimise risk

For strip coal mine design, these design aims require;

- **Efficiency in waste mining.** Waste mining can account for over 70% percent of the cost per ton coal produced from a strip mining operation and therefore the most obvious approach to cost efficiency is through optimisation of waste mining.

- **Optimal use of labour.** Designing for efficiency in use of labour requires consideration of maximum productivity per person, and selection of methods, equipment and techniques which are commensurate with the pool of labour available and its quality.

- **No under-design in coal extraction.** Since the coal extraction cost (expose, drill, blast, load and haul) is often only a minor cost component in the operation, the cost of over design is not significant. The penalty associated with under-design, leading to loss of production, is enormous. The biggest risk is normally loss of revenue due usually to loss of production.

First and foremost, mine planning is an iterative process. The first ‘iteration’ starts off quite primitively, and successive iterations are characterised by having different aims, progressively increasing effort in undertaking them, progressively increasing reliability in the results, and added detail in the costs derived from them. Nevertheless, the
basic steps involved in undertaking them are quite similar and can be summarised as;

1. Establish the guidelines for undertaking the study. Companies have various corporate objectives which may include life of mine, maximum cost of production, scale of operations, ability to adjust to external factors, exposure to risk or utilization of the resource. The cost associated with these objectives has to be quantified and the objectives understood and agreed upon in order to optimize the plan within the set framework.

2. Determine the extent of (potential) mining reserves to be blocked out for the study, (Limits assessment - physical limits and economic limits), from geological model data

3. Determine the mining method to be used

4. Prepare a mine layout, with the mining reserves subdivided into blocks of a size and shape consistent with the mining method and degree of reliability over the time frame required (block size may be related to time frame, method and equipment requirements, for instance annual volumes or for detailed week-by-week analysis of volumes, etc.)

5. Determine the geological, mining and other characteristics of the mining blocks

6. Select the equipment and/or production rate to be used for mining of the blocked out reserve. Determine the production rates and the constraints on usage of the equipment

7. Determine quantities, qualities and resource requirements for the mining over time. (Mine scheduling, preliminary at conceptual study level to detailed at feasibility study level)

8. Establish the unit costs of operating the equipment and the estimated revenue from sale of the products

9. Apply unit costs to the equipment and other resources required, to determine mine operating costs over time

10. Establish equipment life, timing of replacements and capital expenditures over time

11. Prepare economic analysis of cost schedules

12. Make a decision based on project guidelines.
Since strip mine planning is iterative, it makes sense to re-evaluate the proposed mine plan with ever-increasing detail so as to increase the reliability of the final mine plan – and so as to recognise and limit expenditure on reserves that show themselves to be uneconomic at some point. Figure 10.6 shows diagrammatically how the planning process proceeds and integrates within the planning stages – conceptual, pre-feasibility and bankable feasibility studies.

The generic content of various stages phases of mine planning studies are given in the following sections.

Figure 10.6  Components of mine planning

10.3.1 Conceptual Mining Feasibility Study

This class of study is not so much concerned with economics, as it is with examining a mining concept and proving that it is technically feasible. Conceptual studies should be taken to the point of costing, so that when subsequent more detailed studies are completed, comparisons can be more readily made. It is important at this stage to develop a concept with reference to generalities and not detail, since the data and evaluation methodology is not geared to explore these poorly defined premises at this stage. Table 10.2 summarises this planning stage in terms of;
LIMITS OF MINING

A conceptual mine planning exercise often commences with an assessment of the physical limits of mining. In coal mining, the coal 'orebody' is normally well defined from a geological perspective. This simplistic approach is changing as coal mines develop deeper and more complex deposits and are therefore requiring more sophisticated methods of limits assessment than that provided by geological limits alone. Mining limits are used to establish the total coal reserve (economical or un-economic). A knowledge of the approximate box-cut, strips and final void location is also important for planning of dumps, services facilities and surface plants. In deciding the limits, flexibility should be considered in the light of how the limits might change, so that the mining method can be assessed in terms of its potential to accommodate the likely changes in limits over the life of the mine. A typical example would be a mine supplying an tied power station designed for the particular grade of coal produced. It requires far less flexibility than a similar mine supplying a diversified market with a varied product mix - even though the mine plan will probably be developed to support one particular mix initially.

Table 10.2 Conceptual planning phase in outline

<table>
<thead>
<tr>
<th>Purpose</th>
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<tbody>
<tr>
<td>Justify in-fill drilling program</td>
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<tr>
<td>Provide more refined exploration targets.</td>
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<tr>
<td>Identify all major constraints affecting development of project.</td>
</tr>
<tr>
<td>Define mining method or method options.</td>
</tr>
<tr>
<td>Identify any factor which would render the mine unworkable.</td>
</tr>
<tr>
<td>Highlight the major cost components of the mining method.</td>
</tr>
<tr>
<td>Allow for objective decision-making with respect to further project evaluation.</td>
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</tbody>
</table>
### Basic design data
- Topographical Data, 5 meter contours, 1:5000 scale preferable.
- Geological information required at same scale as topography.
- Geotechnical information including samples of waste material, estimates of weathering, likely aquifers, type of floor material. Unusual or troublesome material flagged (e.g. clays).
- Coal preliminary yield and product quality data available for most important seams. Unusual characteristics flagged (e.g. spontaneous combustion effects, high sulphur, etc.).
- Plans with base of all major seam horizons. Coal thickness and all major seams. Seam parting thickness and aggregates for minor seams.
- Overburden/Interburden intervals and thickness.
- Faults/Dykes identified even if position and extent not clearly known.

### Methodology
- Determine **limits of mining** based on technical considerations (authorisation boundaries, dykes, major watercourses, roads etc.)
- Determine limits of mining based on economic criteria (depth of excavation, waste/coal ratio, marginal ratios by extent, width and depth). Rules-of-thumb may be used pending a more sophisticated economic ranking study.
- Assess any other significant constraints on development (e.g. outside dump volumes, seam dips, location of surface facilities).
- Nominate potential mining method(s) and assess project in light of constraints imposed by this method (e.g. lead times).
- Block out conceptual mine plan using large size blocks, calculate volumes, etc., and determine approximate mineable ratio/tonnage curve. Nominate production rate(s).
- Undertake (several) preliminary schedule(s) of production at nominated rate(s).
- Tabulate operating schedules and apply budget estimates of capital and operating costs from manufacturers and others.

### Deliverables and reliability
- General description of mining development proposed.
- Detailed discussion of major constraints affecting development and influence of constraints on production rates, cost, development timing, etc.
- Tabulation of preliminary equipment requirements, operating hours and costs, manning and production volumes.
- Approximately ten drawings with proposed general layout and plans and sections of proposed mining method.
- Bar chart of total project development timing

The principal aim of this study is to supply reliable information about constraints affecting development (production rates achievable, etc.). Equipment list reasonably reliable in terms of type size but not in number required. Requirements for unusual capital equipment would be flagged, but capital or operating costs only likely to be reliable within approx. 25%.

### Decision criteria
- Budget approval for in-fill drilling geotechnical logging and reporting, large diameter cores, detailed coal quality assessment.
- Budget approval for technical evaluation to detailed design stage.
- **OR**
- Decision to undertake further conceptual work in order to bring project into more favourable economic situation (additional drilling, additional evaluation, project “on-ice” until improvement in market or other constraints outside control of company).
The definition of the physical limits of mining is typically:

- Lease boundary
- Other physical authority or regulatory boundaries
- Clear geological limits such as outcrop or boundary faults
- Clear topographical limits such as major watercourses, abrupt changes in topography
- Clear limits associated with the local infrastructure
- Economic limits - an economic limit is defined as where the coal seam within the physical mining limits can actually be mined profitably, and also as an indication of the relative economics of one part of the mine with another.

### 10.3.2 Cost Ranking Pre-feasibility Study

A pre-feasibility study is defined following the South African Guide to the Systematic Evaluation of Coal Resources and Reserves (SABS 0320: 2000) and the SAMREC Code as:

"the first evaluation, planning and design study following a successful geological exploration campaign. It provides a preliminary assessment of the economic viability of a coal deposit and forms the basis for justifying further investigations. It summarises all geological, mining, coal processing, engineering, environmental, marketing, legal and economic information accumulated on the project".

In surface strip coal mining, the stripping ratio defines the quantity of waste to be removed to expose a unit quantity of ore. In essence it reflects a cost structure based on costs of mining being proportional to the quantity of waste, whilst the revenue is proportional to the quantity of ore. In shallow, single seam, strip coal mining operations, where all of the overburden is moved by the same equipment, typically a dragline whose costs accounts for up to 70% of the total operating costs and if the coal yield and characteristics do not vary greatly, the strip ratio formula is a useful relative measure of economic worth.

When the factors dictating where to mine become more economically based rather than physically based (such as depth or strip ratio), the cost ranking approach, within a pre-feasibility type study is often used.
The ranking uses the equipment characteristics and operating constraints of the conceptual study and applies their associated costs to rank the mining reserves on (typically) a profitability basis. This refines the limits of mining and assigns priority for development preparatory to layout of the detailed mine plan. In contrast to the conceptual study, a cost ranking analysis is usually not an implementable mine plan and may be based on geological (grid) blocks rather than mining sequential and method-related blocks.

Table 10.3 summarises this planning stage in terms of;

- Purpose
- Basic design data
- Methodology
- Deliverables and reliability
- Decision criteria

**COST RANKING**

The cost factors required for a block-by-block economic ranking analysis are;

- Cost of removing a unit of waste (grub, soil pre-strip, drill, blast, load, haul and dump etc)
- Cost of mining and transporting a unit of coal (clean, drill, blast, load, haul and dump etc)
- Cost of beneficiating coal and transporting and marketing the product
- The percentage recovery, after providing for pit losses, dilution, plant yield, moisture changes etc
- The net revenue per unit of saleable coal

A more robust economic ranking mechanism is essential for multi-seam mines, or where varied mining equipment is used for waste removal, since in these types of mines, the simplistic stripping ratio formula does not faithfully represent the true cost structure.
Table 10.3 Pre-feasibility cost ranking analysis

<table>
<thead>
<tr>
<th>Purpose</th>
<th>Define the optimum size and shape of mine reserves for the detailed mine layout. Define the priorities for scheduling the detailed mine plan, particularly the optimum areas for mining in the initial years of mine life.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Basic design data</td>
<td>As per conceptual study. In addition, the results of the conceptual Mining study must be known, to provide the basic costs to be applied to the cost-ranking blocks.</td>
</tr>
<tr>
<td>Methodology</td>
<td>The potential mining area should be blocked out to include all reserves which are clearly not excluded by some obvious factor. The blocks may be the same as for the conceptual mine plan, but more commonly are regular shaped blocks approximating the possible shape of mining blocks in the detailed mine plan. Block sizes, shapes and orientations should be selected so that the probable cost of mining is consistent from one part of the block to another. Calculate waste and coal quantities and important quality characteristics for each block. Dissect each block into characteristics associated with mining operations to be performed on it. For example: dissect waste volumes into components to be moved by dragline and components to be moved by truck. Calculate all of the costs associated with the above mining operation and include these on a block-by-block basis. Apply revenue for the mining blocks (coal quantity by yield by price/ton) to determine the gross profit from mining block. Where the cost of mining a particular block is dependent on whether the block above is mined or not, then incremental costs must be derived. Tabulate mining reserves by increasing cost/profitability. Examine the extent of profitable reserves to ensure that there are sufficient reserves to support the size and type of equipment upon which the cost ranking was based. If the reserves are inconsistent with the basis of the ranking, then the ranking must be undertaken again, until the results are consistent. The cost ranking must be repeated over a range of costs for any cost or revenue elements which are largely unknown (e.g. selling price). Capital costs should be included in the ranking for new projects. The ranking should also be undertaken on an operating cost only basis as well to determine the lateral extent to profitable mining for current operations. Mining schedules may be undertaken on economically viable mining blocks. A simple schedule should also be undertaken on blocks in order of profitability (most profitable to least profitable) even if this sequence is physically impossible - since this represents the ultimately most profitable (operating cost only) case. If such a case is unprofitable, then there is little point in studying any others. Tabulate reserves and priorities for mine development.</td>
</tr>
</tbody>
</table>
Deliverables and reliability

- General description of mining concept used as the basis for the evaluation(s).
- Table of reserves and plans showing the economic and less economic reserves graphically and the economic limits of mining on a seam by seam basis.
- Pie Charts demonstrating the breakout of costs on a cost-center basis and on a cost category basis.
- Recommendations as to which seams to mine and the extent of mining, which areas to schedule for early excavation and suggestions for locations for surface facilities and other infrastructure.

The principal aim of this study is to provide reliable guidelines before commencement of detailed design. If the mine plan is to be non-viable, then the cost ranking analysis should be reliable enough to show this.

Decision

- Budget approval for technical evaluation to detailed feasibility design stage.
- OR
- Decision to undertake further conceptual work in order to bring project into more favourable economic situation (additional drilling, additional evaluation, project "on-ice" until improvement in market or other constraint outside control of company).

Conceptually, flat lying tabular orebodies (such as coal seams) should be much simpler to assess than geologically complex disseminated orebodies. For complex multi-seam coal mines however, waste stripping equipment is often highly sensitive to depth and size (and relative economics) of equipment is a function of the size of the total reserve. In addition, since the majority of costs are associated with mining as opposed to beneficiation, small errors in mining cost estimates greatly impact on the ranking.

The costs of mining are normally prepared assuming typical equipment, using manufacturers operating costs models. Since the lateral dimensions in many coal mines are normally big compared to the vertical dimensions, the influence of slopes can often be ignored for the purpose of block ranking. In preparing the costs of each component operation to be undertaken on the mining block, both the operating and some capital component of the costs should be included - so that blocks which can be mined using equipment which is more labour intensive are rigorously analysed and costed.

If, after the ranking exercise, the total quantity and shape of the mining reserve is inconsistent with the equipment on which the cost structure was based, then new equipment must be selected and the exercise re-run. One of the deliverables from the cost ranking study is cumulative mineable quantity against increasing cost - thus allowing quick examination of various scenarios for development, ultimate production rate and production rate build-up. Figure 10.7 shows a typical result. It is essentially similar to the grade/tonnage curve in other types of mining. It can be easily used at this level of planning study to estimate for instance, the maximum capital expenditure allowable, based on a specified rate of return, discount
factor and sales income (but note the simplifications inherent in the analysis also).

Figure 10.7 Cumulative coal reserves as a function of cost, derived from cost ranking analysis, together with simple evaluation of present value of profits in the first five years of production.

10.3.3 Detailed Mining Feasibility Study

A feasibility study assesses in detail the technical soundness and economic viability of a mining project. It is seldom undertaken unless there is a reasonable assurance that the proposal is viable and represents a detailed economic evaluation that serves as a basis for the investment decision and allows for the preparation of a bankable document for project financing. The study constitutes an audit of all geological, geotechnical, mining, engineering, coal processing, environmental, marketing, legal and economic information accumulated on the project. (SAMREC Code and SABS 0320:2000.).

Table 10.4 summarises this planning stage in terms of;
Purpose

- Basic design data
- Methodology
- Deliverables and reliability
- Decision criteria

Table 10.4 Feasibility study analysis

<table>
<thead>
<tr>
<th>Purpose</th>
<th>Allow for project commitment with respect to definite sales contracts, raising of finance and government approvals. Provide details required by regulatory and permitting processes. Provide formal basis for structuring of mine operating company.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Purpose</td>
<td>As for pre-feasibility, extended to accommodate the requirements outlined broadly in Table 10.1. Information on Deposit Geology: Overburden Geology: Coal Hydrology Geometry Geography General Project Information Market Transportation Utilities Land and Mineral Rights Water Labour Governmental Environmental Development and Extraction Geological Models Mine Size Determination Reserves Mining Method Selection Pit Layout Equipment Selection Project Costs Estimation Development Schedule</td>
</tr>
</tbody>
</table>
### Methodology

- Determine new limits of mining based on results from pre-feasibility study and new geological data. Draw up limits in three dimensions having due regard to mining method to be used.
- Critically assess and specify pit design parameters.
- Block out detailed mine plan using nominated bench widths and offsets for each mining bench.
- Calculate volumes, thicknesses, densities and quality data for all the blocks, and carry out manipulations on data to categorise reserves and select equipment sizes in more detail. Calculate and input other characteristic data into each block (e.g. mining recoveries, swell factors).
- Determine volumes of waste to be disposed and location (both in-pit and ex-pit) for disposal on a strip-to-strip basis. Calculate haul truck and/or dragline rehandles for disposal.
- Determine operating schedules, build-up and full production rates and timing for all equipment.
- Undertake detailed scheduling (several times if necessary) to arrive at production schedule for mine life. Tabulate operating schedules.
- Develop operating, maintenance and capital costs for all equipment and supplies used.
- Tabulate capital and operating costs schedules.

### Deliverables and reliability

- Bankable feasibility study containing inter-alia: Detailed description of mining method.
- Detailed description of all major pit design parameters and justification for selection.
- Tabulation of mining reserves and overburden volumes.
- Detailed tabulation of derivation of major equipment production rates including some sensitivity analysis.
- Specification of all major equipment (size, type and number).
- Tabulation on a month-by-month or quarter-by-quarter basis initially, hence on a yearly basis all mine operating schedules (production, manning, operating hours, purchase and replacement, other).
- Detailed tabulation of operating, maintenance and capital costs for all major equipment.
- Tabulation (on same basis) of capital and operating cost schedules.
- Fulfils the statutory requirements for the reporting of a mineral reserve.

Document sufficiently reliable to withstand scrutiny by technical auditors for raising of finance and for equipment selection. Further optimisation studies would be necessary on a large number of features of the design, but these optimisations would not materially affect the project economics. The study would not be sufficiently reliable for implementation of the mine plan as a short term (2 year) plan, and if immediate development is contemplated, work should be commenced on the first year plan immediately. Capital and Operating Costs estimates within 90% of actual.

### Decision criteria

Budget approval for project development and implementation.
Proceeding with all land purchases and regulatory approvals.
Recruitment of project staff to bring project to development.
Detailed planning of all aspects of development.
Posting of tenders on long lead time.
OR
Further studies in order to bring project into a more favourable economic situation.
From this point onwards, a bankable feasibility document is available for the purposes of informing potential investors of the exploration results and mining prospect. Thereafter, once funding is in place and a mining authorisation is secured, mine establishment and construction would begin.

10.4 Mine Scheduling

In its simplest form, scheduling is concerned with what happens in a mining operation over time. It is an intrinsic component of both the planning and operating phases of a mine and, to a lesser extent, closure and decommissioning. The time frame over which the schedule is undertaken is determined by its application; for planning (concept, pre-feasibility cost ranking and full feasibility), operation (long- medium- short-range planning and production control. Similarly, the time frame determines which data is necessary and its importance (detail and reliability).

In Figure 10.8, the time frame of the schedule has been broadly defined. In actuality, when a schedule is developed, it is often analysed on a month-by-month basis for six months, following which it is done at six monthly intervals for perhaps the next five years, and thereafter at five-yearly intervals until the end of the life of the mine.

![Diagram](image-url)  
Figure 10.8 Diagrammatic representation of scheduling time frame and data requirements according to phases of mine life
A mine operations (production) schedule must be developed in several phases with different levels of detail. The common steps in the mine planning and associated scheduling process are the development of:

- a long-range mine plan and schedule (life of mine, annual periods, limited amount of detail)
- a mid-term mine plan and schedule (next five years, monthly/quarterly periods, updated annually, detailed)
- a short-term mine plan (twelve months, weekly/monthly periods, very detailed).

The long-range plan defines the area to be mined during the life of the mine and ideally should schedule mining of the ore in a sequence that realises maximum return at the end of the life of the mine. In addition, the plan should result in a technically feasible production schedule that satisfies quality and contractual requirements. The long-range plan should also serve as a framework in which shorter term, more detailed planning takes place.

The mid-term plan and associated schedule is prepared annually and provides details of stripping and mining equipment requirements. It should meet tonnage and quality requirements within the scheduling time period. The short-term mine plan is concerned with the present operating state of the mine and is developed within the confines of the mid-term mine plan. This plan is used for quality control as well as equipment scheduling, personnel scheduling, maintenance scheduling and inventory control.

10.4.1 Primitive Scheduling for Conceptual Planning

At this level, scheduling consists of simply evaluating a conceptual mine plan in some order knowing only the quantity of ore required to be produced. Associated with this ore production rate is a waste removal rate which needs to occur simultaneously (or approximately so at this stage – later refinements would include stripping 3-6 months ahead of ore production to produce buffer block stocks). Primitive scheduling at this level may not even consider any equipment, since the purpose of such a schedule is to establish approximate quantities to be moved. Usually, the size and amount of equipment cannot be ascertained until the results of these rougher schedules are available.
With increasing sophistication, the scheduling process involves stepping through the mine plan in increasing detail. With increasing schedule sophistication, more constraints can be considered, and correspondingly examination of the results made for shorter time periods.

How the ore and waste quantities are scheduled can have a significant influence on mine economics. With surface strip coal mining, the waste stripping schedule is relatively inflexible, for a tabular-type coal seam or several seams. Terrace mining exhibits more options in this regard, as does the more convention pit-type operations. For this analysis, a simple illustrative model, shown in Figure 10.9 is used in which 10 ore blocks are overlain by 10 waste blocks.

![Figure 10.9 Scheduling example orebody](image)

A production rate of 5 blocks per year (irrespective of whether the blocks are ore or waste) will be assumed. The value for an ore block is R30 and the cost of mining the ore or waste is R10/block, hence the total cost involved in waste removal would be R100 and the ore value generated is R200. If both ore and waste could be mined instantaneously, the net present value (NPV) would be R200-R100 = R100. However, for this orebody geometry, some waste must be removed to expose the ore, the amount of waste removal giving rise to a number of possible scheduling scenarios.

**SCHEDULE 1: PRE-STRIP WASTE PRIOR TO ORE MINING**

For operational simplicity, the orebody depicted in Figure 10.9 is stripped of all waste prior to ore mining. This is shown in Figure 10.10 and the NPV, at a discount rate of 20%, given in Table 10.5 as R29.71.
Figure 10.10 Schedule 1 mining method

Table 10.5 NPV for schedule 1

<table>
<thead>
<tr>
<th>Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste costs (R )</td>
<td>-50.00</td>
<td>-50.00</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Ore income (R )</td>
<td>0.00</td>
<td>0.00</td>
<td>100.00</td>
<td>100.00</td>
</tr>
<tr>
<td>Profit (R )t</td>
<td>-50.00</td>
<td>-50.00</td>
<td>100.00</td>
<td>100.00</td>
</tr>
<tr>
<td>PV of schedule (R )</td>
<td>-41.67</td>
<td>-34.72</td>
<td>57.87</td>
<td>48.23</td>
</tr>
<tr>
<td>NPV</td>
<td></td>
<td></td>
<td></td>
<td>29.71</td>
</tr>
</tbody>
</table>

**SCHEDULE 2: PRE-STRIP WASTE FOR ONE YEAR FOLLOWED BY ORE AND WASTE MINING**

This schedule requires pre-stripping 5 blocks of waste in year 1. In years 2 and 3 three blocks of ore are mined for every two blocks of waste. The final year would have one block of waste and 4 blocks of ore mined. This is shown in Figure 10.11 and the NPV, at a discount rate of 20%, given in Table 10.6 as R43,02.
Figure 10.11 Schedule 2 mining method

Table 10.6 NPV for schedule 2

<table>
<thead>
<tr>
<th>Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste costs (R)</td>
<td>-50.00</td>
<td>-20.00</td>
<td>-20.00</td>
<td>-10.00</td>
</tr>
<tr>
<td>Ore income (R)</td>
<td>0.00</td>
<td>60.00</td>
<td>60.00</td>
<td>80.00</td>
</tr>
<tr>
<td>Profit (R)t</td>
<td>-50.00</td>
<td>40.00</td>
<td>40.00</td>
<td>70.00</td>
</tr>
<tr>
<td>PV of schedule (R)</td>
<td>-41.67</td>
<td>27.78</td>
<td>23.15</td>
<td>33.76</td>
</tr>
<tr>
<td>NPV</td>
<td></td>
<td></td>
<td></td>
<td>43.02</td>
</tr>
</tbody>
</table>

SCHEDULE 3: PRE-STRIP WASTE ONE BLOCK AHEAD OF ORE MINING

Comparing scenarios 1 and 2, there was an improvement in the NPV when the time lag between stripping and mining was shortened. In this scenario, stripping is maintained one block ahead of ore mining, to make the time lag even shorter. This is shown in Figure 10.12 and the NPV, at a discount rate of 20%, given in Table 10.7 as R59,45.
Comparing scenarios 2 and 3, there was a further improvement in the NPV when the time lag between stripping and mining was shortened. In this scenario, stripping lead is further shortened. This is shown in Figure 10.13 and the NPV, at a discount rate of 20%, given in Table 10.8 as R60,73.
Using this approach, it is reasonable to expect costs of mining to increase, say by 10%, due to reduced pit room, etc. In this case the NPV would be reduced to R47,82 as shown in Table 10.9. Thus far, in terms of NPV, schedule 3 remains the most favourable.
Table 10.9  Schedule 4 cost amendment due to reduced pit room

<table>
<thead>
<tr>
<th>Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste costs (R )</td>
<td>-30.25</td>
<td>-30.25</td>
<td>-24.75</td>
<td>-24.75</td>
</tr>
<tr>
<td>Ore income (R )</td>
<td>42.75</td>
<td>42.75</td>
<td>47.50</td>
<td>57.00</td>
</tr>
<tr>
<td>Profit (R )t</td>
<td>12.50</td>
<td>12.50</td>
<td>22.75</td>
<td>32.25</td>
</tr>
<tr>
<td>PV of schedule (R )</td>
<td>10.42</td>
<td>8.68</td>
<td>13.17</td>
<td>15.55</td>
</tr>
<tr>
<td>NPV</td>
<td></td>
<td></td>
<td></td>
<td>47.82</td>
</tr>
</tbody>
</table>

**SCHEDULE 5: MINING RATE DOUBLED**

If the mining rate could be increased to 10 blocks per year by purchasing more equipment, this should further improve the NPV. However, there would be an increase in the equipment ownership costs to be charged against both ore and waste, for example a 20% increase in the cost of mining. This schedule is shown in Figure 10.14 and the NPV, at a discount rate of 20%, given in Table 10.10 as R43,75.

Without any additional costs, this schedule would have been the most favourable, returning an NPV of R74,31 as shown in Table 10.11.

Figure 10.14  Schedule 5 mining method
This example demonstrates some important aspects of production scheduling. In particular, the NPV is dependent upon:

- The time interval between waste stripping and ore mining. The NPV is highest when the lead time is short. With added costs associated with shortening the lead time, however, there may or may not be an improvement in NPV.

- For the same unit costs, the best NPV is achieved with the highest production rate. As increasing production rates also imply an added cost, there may or may not be an improvement in NPV.

In addition to the NPV options considered above, there are a number of stripping ratio options that can be considered also.

### 10.4.2 Stripping Sequence Options

These options are illustrated by using an idealised steeply dipping vein type deposit, more typical of open-pit mining than terrace or strip mining. However, it is the principles of the approach that is important more than the applicability of the orebody geometrics. If a typical stratified coal deposit is considered, as shown in Figure 10.15, apart from the initial waste pre-stripping to expose the coal (the box-cut),
with a strip mining method (and more especially when using a dragline as the primary stripping equipment) there is little possibility of varying the stripping ratio over and above that required to expose the coal immediately below, apart from a limited dragline advanced bench chopdown or truck and shovel pre-strip in deeper overburden areas.

Figure 10.15 Stripping ratio sequencing – simple strip coal mine

Figure 10.15 introduces the concept of stripping ratio variation over time. At any point in time, an instantaneous stripping ratio can be defined for the mining scheduled for that period. These instantaneous ratios can differ over time, but will eventually sum to the overall stripping ratio. When scheduling stripping options, the instantaneous ratio may differ, but will always sum to the overall stripping ratio at the end of the life of mine.

THE MAXIMUM (DECREASING) STRIPPING RATIO METHOD

If each bench is mined to the final pit limit before the next one is started, the stripping ratio will decline from a maximum at the
beginning of the operation to a minimum at the end of the life of mine. It is not often used except if tax concessions are made, since early profit is necessary to offset capital and interest payments early in the mine’s life. Problems can also be experienced with over- (grade) mining if the price of the material falls (thus too much waste removed if the ore price drops and the mine’s economic bottom level isn’t so deep). The method is illustrated in Figure 10.16 by way of cross sectional stripping volume changes with time.

![Diagram showing stripping volume changes with time](image)

Figure 10.16 Maximum (decreasing) stripping ratio method

**THE MINIMUM (INCREASING) STRIPPING RATIO METHOD**

Only the waste that is necessary to expose the orebody for immediate mining is removed. Benches are mined in the waste in an overall slope parallel with the final bench face slope. The maximum profits are made in the early years (assuming ore grade constant with depth) and there is no danger of over-grade mining if the price of the mineral falls. The main problem is associated with pushing back all the mine benches as the mine gets deeper, a lot of blasting and excavation machinery is necessary to mine each bench to the final bench face or to push back to expose more ore. It is not ideal to purchase more capital equipment as the mine nears the end of it’s life. The method is illustrated in Figure 10.17.
CONSTANT STRIPPING RATIO METHOD

Waste is removed over the life of the mine at the overall (averaged) stripping rate. Faces in the waste begin shallow, but increase steadily in steepness up to the overall pit slope angle. A good compromise method, where machines and labour requirements stay nearly constant (for a constant ore tonnage per year). It also requires many work faces and excavating equipment later in the life of the mine which is not necessarily optimal in terms of cash flow and capital pay back periods. The method is illustrated in Figure 10.18.

PHASE STRIPPING METHOD

From a financial point of view, the best stripping methods will be the one’s with a low stripping ratio at the beginning of the mine life (to support capital pay back and to show profits faster) and also at the end of the mine’s life (to make larger positive cash flows to support decommissioning and closure, or re-investment), together with the
fact that the unit costs are inclined to increase when the last bits of ground are taken out – commonly referred to as pigging. There are several advantages in the use of phased stripping, namely;

- Labour and equipment fleet are built up slowly and then reduce as they get older without the need for late replacement.
- It is not necessary to reduce the equipment fleet at the end of the mine’s life. (Life of the machines etc. will be finished when the mine is finally worked out).
- Excavating and stripping can be divided in two or more areas which will make production scheduling easier.

The method is illustrated in Figure 10.19 and Figure 10.20 shows the representation of these methods as stripping curves over time.

Figure 10.19    Phase stripping ratio method

MAXIMUM – MINIMUM STRIPPING METHOD

Through the combination of the first two methods, a valuable scheduling method is created, by determining the maximum and minimum stripping ratio variation during mining. The plot comprises:

- Increasing ore tonnage (or time if ore production is constant) on the X axis.
- Increasing waste tonnage on the Y axis.
Figure 10.20 Stripping curves for the various methods of waste and ore mining schedules introduced previously

It is easy to establish the maximum stripping curve, the whole pit is operated by mining one bench at a time to its limit. The minimum stripping curve is more difficult to define and is usually found by block models of the method, which maintains the lowest immediate stripping ratio during any moment in the mine’s life.

The maximum and minimum curves give an area between the two curves in which various short term stripping schedules can be applied without influencing the overall mine profitability, but allowing mining to respond to short term conditions. In scheduling, when the actual (short term) stripping curve is near the minimum curve, there is not enough waste stripping being done to satisfy future ore production requirements. If the price of the mineral should fall, stripping can be reduced right to the minimum curve, or when the price is good, stripping can be increased to the maximum curve. The waste stripping methods can also be scheduled according to the specific financial policy of the company (low to start with and also low towards the end of the mine life). Figure 10.21 shows an example of the technique in comparison with the maximum-minimum curves determined earlier. The mine can schedule stripping on a level to level or year to year basis, as suits the conditions at the time, as long as they stay within the max-min envelope.
Using a strip mining operation as an example, at this level of scheduling, actual blocks of coal to mine are considered, together with the associated waste volumes to be removed. However, although this approach enables an overview of mining volumes and rates to be found, it doesn’t provide any indication of what equipment will be used, nor address issues of coal quality, etc. over time. The purpose of schedules at this level is only to establish approximate quantities and qualities. Usually, the size and amount of equipment cannot be ascertain until the results of these rougher schedules are available.

### 10.4.3 Broadbrush Scheduling for Pre-feasibility Planning

At this level of scheduling, actual mining requirements are looked at in some detail. These requirements are a function of the particular type of schedule being analysed, be it waste types, waste volumes to be stripped, coal qualities or production rates, etc. Initially, waste and coal are scheduled, each according to a set parameter. Typically, scheduling based on a set coal production rate (coal-controlled schedule) is established initially, following which the waste stripping rate is determined. Often the waste stripping schedule is problematic,
due to periodic increases and decreases in production rates required, resulting in stripping equipment over-capitalisation and periods of under-utilisation.

To explore how a more logical stripping schedule would impact on coal produced, a waste-controlled schedule is followed, where waste mining is assigned a constant rate over the life of mine and coal production examined. Again, this first broad-brush approach may only serve to highlight where compromise is necessary between the two approaches. Progressively more refined broadbrush schedules are then undertaken as the constraints and subtleties become better understood, perhaps varying the build-up rates, thicknesses assigned to particular equipment, or by scheduling on calorific value, ash content or some other parameter.

At this point, it becomes possible to estimate major equipment requirements for the balanced broad-brush schedule. In doing this, the schedule itself may have to be modified to accommodate the generic types of equipment considered, for instance a dragline operation compared to a pure truck and shovel operation in the overburden. The representative data requirements for this type of schedule are given below in Table 10.12.

10.4.4 Detailed Scheduling for Feasibility Planning

As a mine plan develops from concept to full feasibility, so should the associated scheduling requirements and sophistication. Detail changes according to:

- Scheduling time period reductions
- Scheduling data gets progressively more detailed, is categorised more, and pertains more and more to the mining operation and less and less to geological parameters
- Logic and constraints become better defined.

At the end of the detailed scheduling exercise an implementable mine plan should be evident. This is determined by applying an excavation sequence schedule which satisfies the constraint criteria established in the previously schedule, and by scheduling the utilisation of supporting equipment. The detailed scheduling phase will also indicate all the major equipment items required over the life of the mine (minor equipment is not normally considered as these will be
specified later according to the support requirements of major equipment).

Table 10.12 Data requirements for a broad-brush balanced coal-waste schedule

<table>
<thead>
<tr>
<th>Data</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste area</td>
<td>square metres</td>
</tr>
<tr>
<td>Waste thickness</td>
<td>metres</td>
</tr>
<tr>
<td>Waste volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Tons in-situ coal</td>
<td>tons</td>
</tr>
<tr>
<td>Specific coal quality requirements (sulphur, density, volatiles, ash fusion temp, moisture, etc.)</td>
<td>As required by parameter</td>
</tr>
<tr>
<td>Calorific value</td>
<td>MJ/kg</td>
</tr>
<tr>
<td>Nominal mining loss</td>
<td>metres seam thickness</td>
</tr>
<tr>
<td>Mining recovery</td>
<td>percent</td>
</tr>
<tr>
<td>Mineable coal quantity</td>
<td>tons</td>
</tr>
<tr>
<td>Dragline prime volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Dragline rehandle</td>
<td>percent</td>
</tr>
<tr>
<td>Dragline total volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td>cubic metre/ton</td>
</tr>
</tbody>
</table>

At this level of detail and time-frame, it should also become possible to examine the ease of operation of the mining plan, especially in terms of what buffers, inventories and stockpiles are maintained. To determine this, the mining operation sequential steps have to be identified, together with the realistic requirements for these buffers. These are the guidelines for use in the schedule. Every step in the sequence of excavation of a mining block requires some sort of buffer
between it and the other steps in the sequence. Understanding the requirements of, and planning the operation for this is the function of the detailed schedule. Typically, when some equipment is used over a different time period or shift roster from others, then for full utilisation of the equipment, a buffer is required. This is usual for dragline operations, where it is critical that the machine does not become coal-bound and has to be stood down for a period of time.

Similarly, in-pit inventories of coal can be examined according to the requirements of the operation, more so when blending is an issue. High inventories allow for a greater amount of operational flexibility, but will again possibly result in the dragline becoming coal-bound. A reduced inventory of floor stocks will reduce the likelihood of this, but on the other hand is associated with a greater risk. Low inventories equate to lower holding costs, but complicate the scheduling of coaling equipment due to fluctuations in excavation requirements to accommodate a more flexible short-term blending requirement.

A typical example would be a strip coal mine which feeds a power station directly (without a preparation plant). Achieving consistency of ash, specific energy and volatiles is often quite critical in the development of a schedule. On a global scale, the average characteristic of a blended product is controlled by fixing the approximate proportions of coal extracted from the various parts of the mine. Usually these are fixed by the proportions of seams. On a short term basis, the actual coal quality is a function of the equipment deployed in the various mining areas and haulage from these areas; and may be affected by delays caused by blasting, or by equipment moving from one working area to another. Scheduling would be required over various time frames to ensure that coal sourced from the strips can be blended to provide the customer with the correct blend at the lowest possible cost to the mine. Similarly, excessive over-specification supply could result in the incomplete utilisation of the resource, since the opportunity to fully exploit lower quality coal has been lost by supplying over-specification.

Due to the site-specific nature of schedules at this level of detail, a comprehensive data list is difficult to provide, but a generic list for a typical dragline operation with truck and shovel pre-strip is given in Table 10.12, in terms of major equipment requirements and Table 10.13 in terms of the scheduling data requirements.
Table 10.12  Major equipment unit operations and scheduling requirements for a strip coal mine

<table>
<thead>
<tr>
<th>Unit operation</th>
<th>Equipment</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste Drilling</td>
<td>Rotary Blasthole Drill</td>
</tr>
<tr>
<td>Waste &amp; Coal Drilling</td>
<td>Blasthole Drill</td>
</tr>
<tr>
<td>Overburden Excavation</td>
<td>Dragline</td>
</tr>
<tr>
<td>Coal Excavation</td>
<td>Front-end Loader</td>
</tr>
<tr>
<td>Overburden Excavation or Pre-strip</td>
<td>Hydraulic Excavator</td>
</tr>
<tr>
<td>Overburden Haulage or Pre-strip</td>
<td>Rear Dump Haul Truck</td>
</tr>
<tr>
<td>Coal Haulage</td>
<td>Rear or Bottom Dump Haul Truck</td>
</tr>
<tr>
<td>Coal Ripping, Dragline Pad</td>
<td>Tracked Dozer</td>
</tr>
</tbody>
</table>

Table 10.13  Generic data requirements for a typical detailed schedule.

<table>
<thead>
<tr>
<th>Data</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Pre-stripping</strong></td>
<td></td>
</tr>
<tr>
<td>Prestripping Area</td>
<td>square metres</td>
</tr>
<tr>
<td>Prestripping Thickness</td>
<td>metres</td>
</tr>
<tr>
<td>Prestripping Volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Prestripping T/S Production Rate</td>
<td>bcm/op.hr.</td>
</tr>
<tr>
<td>Prestripping Drill Hours</td>
<td>Operating Hours</td>
</tr>
<tr>
<td>Prestripping Haulage Truck Rate</td>
<td>tons/op.hr.</td>
</tr>
<tr>
<td>Prestripping Truck Hours</td>
<td>Operating Hours</td>
</tr>
<tr>
<td>Prestripping Powder Factor</td>
<td>kg/m$^3$.</td>
</tr>
<tr>
<td>ANFO Quantity (pre-strip)</td>
<td>tons</td>
</tr>
<tr>
<td><strong>Overburden</strong></td>
<td></td>
</tr>
<tr>
<td>Waste Area</td>
<td>square. metres</td>
</tr>
<tr>
<td>Waste Thickness</td>
<td>metres</td>
</tr>
<tr>
<td>Waste Volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Waste Volume Thickness Increments</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Drilling Rate</td>
<td>bcm/op.hr.</td>
</tr>
<tr>
<td>Drilling Time</td>
<td>Operating Hours</td>
</tr>
<tr>
<td>Waste powder factor</td>
<td>kg/m$^3$.</td>
</tr>
<tr>
<td>Metric</td>
<td>Unit</td>
</tr>
<tr>
<td>--------------------------------------------</td>
<td>------------</td>
</tr>
<tr>
<td>ANFO quantity (waste)</td>
<td>tons</td>
</tr>
<tr>
<td>Dragline Rehandle</td>
<td>percent</td>
</tr>
<tr>
<td>Dragline Total Volume</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Dragline Production Rate</td>
<td>bcm/op.hr.</td>
</tr>
<tr>
<td>Dragline Digging Time</td>
<td>Operating Hours</td>
</tr>
<tr>
<td>Truck- Shovel Volume</td>
<td>cubic meters</td>
</tr>
<tr>
<td>T/S Production Rate</td>
<td>bcm/op.hr.</td>
</tr>
<tr>
<td>T/S Haul Time</td>
<td>Operating Hours</td>
</tr>
</tbody>
</table>

**Coaling**

<table>
<thead>
<tr>
<th>Metric</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal area</td>
<td>square metres</td>
</tr>
<tr>
<td>Coal Thickness</td>
<td>metres</td>
</tr>
<tr>
<td>Coal Volume in-situ</td>
<td>cubic metres</td>
</tr>
<tr>
<td>Coal Density In-situ</td>
<td>ton/m³</td>
</tr>
<tr>
<td>Coal Quantity In-situ</td>
<td>tons</td>
</tr>
<tr>
<td>Mining Recovery</td>
<td>percent</td>
</tr>
<tr>
<td>Mineable Coal Quantity</td>
<td>tons</td>
</tr>
<tr>
<td>Contamination Quantity</td>
<td>tons</td>
</tr>
<tr>
<td>ROM Coal Quantity</td>
<td>tons</td>
</tr>
<tr>
<td>As-mined Ash</td>
<td>percent</td>
</tr>
<tr>
<td>As-mined CV</td>
<td>MJ/kg</td>
</tr>
<tr>
<td>In-situ Volatiles</td>
<td>percent</td>
</tr>
<tr>
<td>Ash Fusion Temp</td>
<td>deg.</td>
</tr>
<tr>
<td>In-situ Moisture</td>
<td>percent</td>
</tr>
<tr>
<td>Coal Haulage Truck Rate</td>
<td>tons/op.hr.</td>
</tr>
<tr>
<td>Coal Truck Hours</td>
<td>Operating Hours</td>
</tr>
<tr>
<td>Coal Rip/Doze Quantity</td>
<td>tons</td>
</tr>
<tr>
<td>Coal Drill/Blast Quantity</td>
<td>tons</td>
</tr>
<tr>
<td>Coal Blasting Powder Factor</td>
<td>kg/m³</td>
</tr>
<tr>
<td>Coal Explosive Requirement</td>
<td>tons</td>
</tr>
</tbody>
</table>

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