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CHAPTER 7

Surface Mining

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Surface Mining

Surface mining cost estimation is a complex exercise that requires analysis of a variety of inputs, including the geology and physical properties of the deposit, equipment selection, equipment productivity estimation, workforce roster and personnel planning. Capital and operating cost estimation for the selected fleet also needs to be analysed.

This chapter details factors to be considered when preparing a surface mining cost estimate. The final section provides a worked example of the cost estimation techniques applied to a small open pit gold mining operation.

PLANNING AND DESIGN

For surface mining operations the objectives of the planning process are to:

- define the resource in terms of grade, tonnage and location
- determine if a market exists for the products and if so what are the quality and quantity constraints
- select a mining method that is the most economic given the physical characteristics of the orebody and rate at which the ore is to be mined
- decide if the project is either economically viable or could be viable under a given set of conditions.

Selection of the most economically viable mining method requires an evaluation of the capital and operating costs of the surface mining equipment. In order to establish surface mining equipment costs for a given project, the planning process must define the:

- mining method
- mining sequence (schedule)
- ore production rate and hence mine life, including the overburden production rate required to sustain the ore production rate
- physical characteristics of the materials to be handled.

Mine life

The life of the mine is determined by dividing the quantity of ore in the designed pit to be extracted by the quantity of ore to be mined per annum to meet the marketing requirements for the products. For a constant rate of production:

$$\text{Mine life} = \frac{\text{Ore available}}{\text{Annual production rate}}$$

In calculating the mine life, the economic limits of the open cut mine must be known to determine the quantity of ore available for mining. This requires optimising the pit's limits, which is part of the planning process and a complex iterative exercise.

Ore characteristics

The physical characteristics of the ore and waste materials must be known in order to select appropriate excavation and haulage equipment. The most important factors are:

- excavatability
- density
- volume measures.

Excavatability

Excavatability is a measure of the effort required to remove ore or waste from its *in situ* position. It is generally described as one of:

- drill-and-blast
- free digging
- rippable.

Assessment of the excavatability of the material is not easily determined with accuracy. Many cases can be cited where contract miners in particular have lost a considerable amount of money as a result of an error of judgement in its assessment.

To gain an understanding of excavatability a geotechnical investigation of the ore and waste materials is required in the early stages of the mine planning process. The information required from this study should include:

- engineering logs of boreholes in ore and waste in both weathered and unweathered material
- engineering seismic surveys across the proposed open cut mine.

A typical engineering log includes a description of the hole, groundwater conditions, rock type, strength, degree of weathering and discontinuities. All can indicate the excavatability.

Franklin, Broch and Walton (1971) proposed a method of assessment, based on joint spacing and point load strength as shown in Figure 7.1.

Density

The *in situ* density of the ore and waste must be assessed in order to determine the volume of material

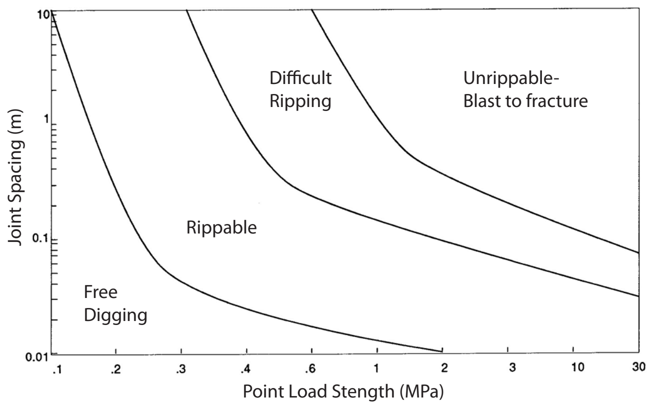

FIG 7.1 - Excavatability assessment (Franklin, Broch and Walton, 1971).

that can be loaded and hauled without overloading the equipment. In dense ores particularly, there is a tendency to overload trucks. A table of typical densities for various ore and rocks in given in Table 7.1. However, it is preferable for the density to be measured in the laboratory using samples from drill core.

TABLE 7.1
Material factors for various ore and rock types.

Material	Bank density (t/m^3)	Swell	Load factor	Loose density (t/m^3)
Basalt	2.97	0.50	0.67	1.98
Clay – bed	2.02	0.22	0.82	1.66
Clay – dry	1.84	0.23	0.81	1.50
Clay – wet	2.08	0.25	0.80	1.66
Coal – 20% ash	1.50	0.35	0.74	1.11
Coal washed	N/A	N/A	N/A	0.80 - 1.00
Dolerite	2.85	0.52	0.66	1.88
Earth – topsoil	1.40 - 1.50	0.43	0.70	0.98 - 1.05
Earth – compacted	1.80	0.25	0.80	1.44
Granite	2.67	0.62	0.62	1.65
Limestone	2.61	0.63	0.59	1.54
Sandstone	2.50	0.63	0.60	1.5
Sand – dry	1.45	0.12	0.89	1.29
Sand – damp	1.90	0.12	0.89	1.7
Sand – wet	2.08	0.14	0.88	1.82
Sand and gravel – dry	1.96	0.12	0.89	1.75

Note: N/A = not applicable.

Dry density is typically used to report mineral resources and ore reserves, while productivity calculations require wet density to estimate bucket capacity and payload.

Volume measures

The volume of material is defined depending on its stage in the earthmoving process.

Bank measure

This is the *in situ* volume of material as it lies in the ground before the excavation process has begun; it is typically referred to as bank cubic metres (BCM). Most contract mining is based on volume measurement, as it is easy to check by surveying.

Loose measure

This is the volume of material after it has been disturbed by drilling and blasting and excavation, and has swollen as a result. It is typically referred to as loose cubic metres (LCM). This measure is important because the measure is used to select appropriate load-and-hauling equipment.

It should be remembered that there are different loose measures at different stages in the mining process. In particular, during the dumping operation some compaction of the material may occur. This is important to consider during dump planning.

Swell factor

Swell is the volume increase after material has been disturbed.

$$Swell = \frac{Disturbed\ volume - undisturbed\ volume}{Undisturbed\ volume}$$

Swell can be expressed either as a percentage or ratio, eg 50 per cent or 0.5:

$$Swell\ factor = 1 + swell$$

where swell is a proportion.

Load factor

This allows for the conversion from loose to bank measure.

$$Load\ factor = \frac{100\%}{100\% + \%\ swell}$$

For example, if swell is 50 per cent:

$$Load\ factor = \frac{100}{100 + 50}$$

$$= 0.67$$

Load factor is usually expressed as a ratio:

$$Volume\ (BCM) = volume\ (LCM) \times load\ factor$$

Example

A granite material during excavation expands from a bank volume of 1500 m^3 to a loose volume of 2500 m^3. What are the swell and load factors? If a ten per cent compaction during dumping is attained, what would the load factor become?

$$Swell = \frac{2500 - 1500}{1500}$$
$$= 0.67 \text{ or } 67\%$$

$$Load\ factor = \frac{100}{100 + 67}$$
$$= 0.60$$

$$Compacted\ dump\ volume = 2500 \times (1 - 0.1)$$
$$= 2250\ m^3$$

$$Swell = \frac{2250 - 1500}{1500}$$
$$= 0.50$$

$$Load\ factor = \frac{100}{(100 + 50)}$$
$$= 0.67$$

Table 7.1 provides example density, swell and load factors for a few common materials.

Mining methods and equipment selection

Selection of an appropriate mining method is primarily dependent on:

- ground conditions
- mine life
- orebody characteristics and selectivity required to minimise mining losses
- required production rate
- topography of the mine site.

These site characteristics will generally limit the choice of mining method to a few options that should be costed in detail to determine the most economic method.

The equipment selected for the operation must be compatible with the site characteristics and mining method adopted. The primary types of surface mining equipment are provided in Table 7.2.

TABLE 7.2
Main types of surface mining equipment.

Excavation equipment	Haulage equipment	Drills
Bucket wheel excavators	Bottom dumpers	Percussion
Draglines	Bulldozers	Rotary
Front-end loaders	Conveyors	
Hydraulic excavators	Rail	
Rope shovels	Rear dump trucks	
Scrapers	Road trains	
Surface miners	Scrapers	

Also important in the selection of equipment is the availability of backup service and spare parts. It is unlikely that the mine will have the facilities to undertake major component overhauls on-site. Therefore, the ability of the equipment supplier to provide exchange units at short notice is of prime consideration if an acceptable level of mechanical availability is to be achieved.

The lead time required to obtain spare parts is of major importance on remote sites. Careful consideration of lead times is required in the planning stage of the project and may necessitate the purchase or hire of additional equipment to be located on-site to cover the periods where a machine is down and waiting on spares.

Scheduling

The marketing plan for the mining venture will determine the required production rate of saleable products over the life of the mine. The purpose of scheduling is to determine how this production will be achieved over time and thus the consequences in terms of provision of labour, equipment and supplies. The scheduling process is complex and involves the tabulation of quantities, qualities and other values for each scheduling period in the mine schedule. Typical tabulations for each period of the schedule include:

- allowances for significant weather events
- average run-of-mine ore grade
- equipment cycle times
- machine hours required
- personnel hours to be worked
- ore and waste quantities.

An accurately prepared schedule of ore and waste quantities to be moved per period is an essential prerequisite to determine the size and number of load-and-hauling units.

As a general rule, it is better to maintain relatively constant volumes of ore and waste to be shifted per period. This allows a constant level of personnel to be established and avoids the need to acquire additional equipment or engage a contractor to overcome short-term peaks in production.

With a new mine, there will be a training period for operators while they become familiar with new equipment and mining procedures. It will generally be several months before the full productivity of a new mine is realised and this inefficiency should be incorporated in the mine schedule.

Working hours

Working hours are key when determining a mining operation's productivity and costs. The variables controlling working hours include:

- award conditions
- operational delays
- plant maintenance
- plant utilisation
- workforce roster.

Table 7.3 provides the calculations used to determine the available working hours per annum. The scheduled hours per shift includes non-productive time resulting from shift changeover, lunch breaks and safety meetings, etc. In the example provided in Table 7.3, this time has been estimated at one hour per shift. This unproductive time is often expressed as a reduction in the minutes available per hour (eg 11/12 × 60 = 55 min/h).

TABLE 7.3
Possible production calculation.

Line no	Material	Units	Resulting time	Calculation
1	Maximum days/annum	d/a	365	
2	Holidays	d/a	0	
3	Weekends	d/a	0	
4	Possible days/annum	d/a	365	1 - 2 - 3
5	Weather delays	d/a	13	
6	Operating days/annum	d/a	352	1 - 2 - 3 - 5
7	Scheduled hours/shift	h/shift	12	
8	Shifts/day	shifts	2	
9	Scheduled hours/annum	h/a	8448	6 × 7 × 8
10	Available hours/shift	h/shift	11	
11	Available hours/annum	h/a	7744	6 × 8 × 10

The available h/a does not take account of equipment availability, which is a function of downtime, resulting from mechanical breakdown and servicing. Plant availability can be increased by providing spare units and servicing machines during lunch breaks or during other non-productive times. For new equipment and efficient servicing the available factor can approach 100 per cent. However, as the hours of service approach the economic life of the machine, utilisation drops and eventually the machine will have to be replaced. Some typical estimates of mechanical availability suitable for use in a feasibility study are provided in Table 7.4.

To estimate the hours worked per employee, rosters, leave and absenteeism must be considered. Table 7.5 provides an example of the methodology used to determine the working hours per employee for an operation working on a three-panel roster of two weeks on, one week off.

Capital expenditure

The operating models typically used in surface mining operations are:

TABLE 7.4
Typical estimates for mechanical availability.

Plant	Mechanical availability (%)
Shovel/excavator	85
Truck	85
Drill	80
Front-end loader	80
Grader	75
Track dozer	75
Wheel dozer	75

TABLE 7.5
Shift calculations (three-panel, two weeks on, one week off).

Line no	Item	Units	Resulting time	Calculation
1	Days on	d/roster	14	
2	Days off	d/roster	7	
3	Maximum days/annum	d/a	365	
4	Weather delays	d/a	13	
5	Annual leave	d/a	20	
6	Sick leave	d/a	10	
7	Funeral leave	d/a	5	
8	Absenteeism	d/a	10	
9	Rostered off	d/a	122	(3/(1 + 2)) × 2
10	Training	d/a	12	
11	Total worked	d/a	186	3 - 5 - 6 - 7 - 8 - 9 - 10
12	Person paid/ possible shift	person/ shift	1.31	(3 - 9)/11
13	Effective hours worked	h/shift	11	
14	Rostered time	h/a	2677	13 × (3 - 9)
15	Effective hours worked	h/a	2050	13 × 11
16	Actual hours worked	h/a	1977	11 × (1 - (4/3)) × 13

- owned and operated plant
- equipment hire, including full (wet) hire or dry hire (where labour and/or maintenance costs may be excluded from the hire rates)
- contract mining.

A mine may use a combination of these options. The decisions to be made regarding capital expenditure on a project are both numerous and complex. The prime consideration is availability of capital or the ability to raise funds for the purchase of equipment. Where the funds are not available or the cost of raising capital is excessive, the alternatives of contract mining or equipment hire should be considered.

Further details regarding capital cost estimation are provided in Chapter 4 – Capital Cost Estimation.

ROCK BREAKING EQUIPMENT

The most commonly used methods of breaking rock in open cut mines are ripping with a dozer and drill-and-blast. The selection of the appropriate method will depend on the nature of the ground. Usually, there is a transition point from one method to the other with increasing depth of excavation and increasing rock strength as the effects of weathering become less. Excavation by ripping is typically limited to rocks that are classified as extremely to moderately weathered.

Ripping

Ripping is used to loosen material prior to loading with excavators, front-end loaders or scrapers. Engineering seismic surveys using shallow refraction techniques have been used extensively to assess rippability of rock. Dozer manufacturers published diagrams showing ease of ripping against seismic velocity. However, seismic velocity alone is not a reliable indicator of the ease of excavation since it does not necessarily give a true indication of the excavation characteristics in areas with boulders and other non-homogeneities typically found in the weathered horizon. Techniques combining engineering and geophysical parameters have been developed by Weaver (1975) and Minty and Kerns (1983). Factors that affect the ripping production of a dozer include:

- crawler traction
- depth of penetration
- dozer power and weight
- groundwater conditions
- joint type, spacing and orientation
- length of rip
- number of rippers per machine
- rock type.

Figure 7.2 shows ripping production against seismic velocity for various sized dozers. It can be used for preliminary estimates in the absence of more comprehensive site information.

Drilling

The principal rock-drilling methods are rotary/percussion and rotary. Rotary/percussion drills may be air, air-hydraulic or hydraulic and may have the hammer located at the top or bottom of the drill string. With top-hammer drilling, the hammer's piston hits the shank adaptor and creates a shock wave that is transmitted through the drill steels to the bit. With

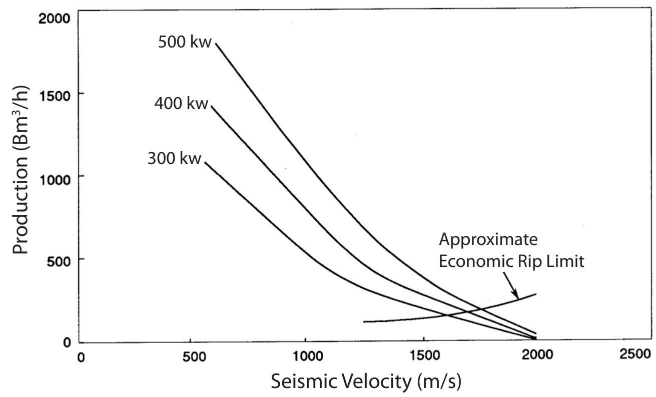

FIG 7.2 - Rock ripping production rate (in bank cubic metres).

down the hole (DTH) drilling, the piston strikes directly on the bit and no energy is lost through the joints in the drill string. The larger mast-type rotary/percussion rigs often use air-percussion hammers with independent hydraulic drill steel rotation and hydraulic-powered steel handling and traction motors.

Rotary drilling has been adapted from oil well drilling to large diameter holes for surface mining. A roller cone bit is used with high feed pressure and slow rotation to crush the rock. The relationship between pressure and rotation speed varies with the type of rock, with lower pressure and higher speeds used in softer formations.

The selection of a drilling machine for a surface operation depends on:

- hole depth
- hole diameter
- nature of the terrain
- rate at which ore and waste is to be removed to comply with the mining schedule
- rock hardness
- rock size that can be handled by the load-and-haul equipment.

Small top-hammer drills are used for small diameter drilling of shallow holes in situations where a larger rig cannot be used. For holes about 85 to 200 mm in diameter and up to 20 m deep, DTH hammer drilling is commonly used. Rotary drilling is generally used for larger diameter drilling of up to about 400 mm in diameter and up to 100 m deep, mostly using tricone bits, although drag bits have some application in soft ground.

It is necessary to design a drilling pattern in order to determine the number and type of drilling machines required. The procedure to be followed for a first-pass design is given in publications issued by drilling and blasting companies and explosives manufacturers as well as in various texts. Essentially, the design process involves the selection of the following key parameters:

- bench height (H)
- burden (B)
- hole diameter (D)

- spacing (S)
- subdrill (SD).

The volume (V) of rock per drill hole can then be determined by:

$$V = H \times B \times S \ (m^3)$$

The length of drilling (L) per cubic metre of rock is given by:

$$L = (H + SD)/V \ (m/m^3)$$

If Q is the volume of rock to be excavated per hour to comply with the mining schedule, then the required drill metres per hour (M) is given by:

$$M = Q/L \ (m)$$

The required number of drills (N) to achieve the production rate is given by:

$$N = M/R$$

where:

R is the average drilling rate (m/h) for each drill

R can be determined from the estimated instantaneous penetration rate (m/h) for a given rock type plus an allowance for relocation and set-up over a new hole and other non-productive drilling time

An example of the calculation of the average drilling rate for a crawler-mounted pneumatic rig drilling an 89 mm diameter hole to a depth of 9 m is given below.

Drilling	**Minutes**	**Per cent**
Instantaneous penetration rate	21.6	85
Medium to hard granite, 25 m/h		
Delays		
Two rod changes	0.8	3
Cleaning hole	0.4	2
Pulling drill rods	1.3	6
Moving drill	1.0	4
Total delays	3.5	15
Total Cycle	25.1	100

Production

Metres drilled per 47 min	= (47/25.1) × 9 = 16.8 m
Instantaneous penetration rate	= 9/21.6 = 0.42 m/min
Average drill rate	= 16.8/60 = 0.28 m/min
Drilling efficiency	= 0.28/0.42 × 100 = 67%

Obtaining the estimation of the average drilling rate for a particular site is difficult. The rate depends on the type of drill, available power, rock hardness, rock discontinuities and operator skill. In the planning stage of a mine, it is advisable to conduct field drilling trials in conjunction with a drilling contractor or equipment manufacturer.

Table 7.6 lists some average rates that may be used as a guide in preliminary studies in the absence of specific data. The rates assume a 67 per cent drilling efficiency.

TABLE 7.6
Average drilling rates.

Class of rock	Average drilling speed (m/h)				
	Jack-hammer	Air-track	Air-hydraulic	Rotary tricone	Rotary drag bit
Soft	8 - 12	25 - 35	30 - 45	21 - 30	35 - 70
Medium	5 - 8	12 - 25	15 - 30	12 - 21	0 - 35
Hard	0 - 5	0 - 12	0 - 15	0 - 12	N/A

Note: N/A = not applicable.

EXCAVATING AND LOADING EQUIPMENT

In the past, the term 'shovel' has been used as a generic title for all loading units. This refers back to the days when the electric rope shovel was the undisputed king of the open pit mine. This is no longer true. Hydraulic excavators have captured a large slice of the market, with buckets of more than 40 m^3 now available. Larger front-end loaders with bucket sizes of up to 40 m^3 are also available. Figure 7.3 categorises the major loading units. Rope shovels, hydraulic excavators and front-end loaders are the main truck loading tools.

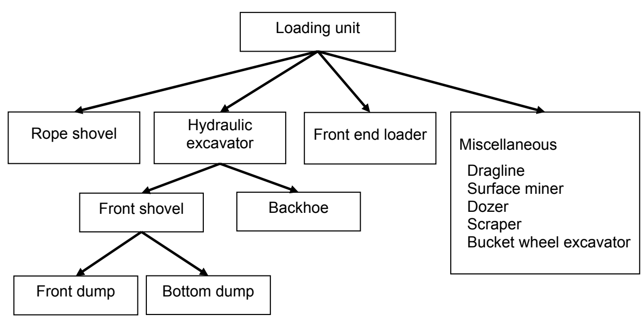

FIG 7.3 - Loading unit types.

Loader specifications

Important loading unit specifications are given in this section.

Bucket size

Bucket sizes can be given in two ways: either struck or heaped volume. The usual capacity given for a bucket is a 2:1 heaped capacity. It should be remembered that the rated capacity is more a method of comparing bucket sizes than a measure of real capacity in any specific application. Selecting the correct bucket size and wear package is very important.

Another way of rating front-end loader capacity is by operating load. This is a nominal value intended to represent the normal load under typical operating conditions. The rated operating load is defined as the lesser of 50 per cent of wheel loader tipping load or 100 per cent of lifting capacity. Manufacturers often provide a rating in tonnes for a machine. This figure is the sum of the bucket load and bucket weight.

Standard bucket sizes are usually based on a loose density of about 1.7 t/m^3. If the density is substantially different from this value, then either a smaller or larger bucket may be required.

Bucket fill factor

Bucket fill factor is a measure of the real volume excavated compared to the stated bucket size. Typical bucket fill factors are provided in Table 7.7.

TABLE 7.7
Bucket fill factors.

	Bucket fill factors		
	Shovel	Front-end loader	Excavator
Fine loose material	1.05	0.95	1.05
Coarse loose material	1.03	0.95	1.03
Well blasted	1.00	0.85	1.00
Medium blasted	0.95	0.80	0.95
Poorly blasted	0.90	0.65	0.90

Ground pressure

Ground pressure of loading units can be important in some applications. In wet, soft conditions, low ground-pressure units are often required. The hydraulic excavator has the least ground pressure followed by rope shovels and finally by front-end loaders. With tracked machines, the ground pressure can be modified by increasing track size. The disadvantage of using larger tracks is that they are harder to turn and place more stress on the track system and propel motors. Table 7.8 gives the typical ground pressure for different loading units.

TABLE 7.8
Loading unit ground pressure.

Unit	Ground pressure – typical values	
	(psi)	(kPa)
Rope shovels	30 - 50	207 - 345
Hydraulic excavator	15 - 30	104 - 207
Front-end loader	>50	>345

Power

Rope shovels with their electric power have few problems with motors. Electricity is generally a lower cost item than fuel. Also, electric motors are usually cheaper to maintain than their diesel counterparts and give excellent availability. The requirement to connect power to the shovel and relocate cables during shovel moves creates operational problems and extra costs. When comparing diesel units like hydraulic excavators and front-end loaders to rope shovels, it is important to include the cost of power distribution.

The effective use of power in front-end loaders is limited by tyre traction and equipment tipping.

Table 7.9 gives the average engine power per cubic metre of bucket capacity. Hydraulic excavators have a higher ratio, which gives them an advantage in digging ability.

TABLE 7.9
Loader horsepower to bucket capacity ratio.

Equipment	Ratio (kW/m^3)
Hydraulic excavator	56 - 114
Front-end loader	49 - 62

Reach and loading height

Reach and loading height are usually not a problem for either rope shovels or hydraulic excavators. It is, however, essential to match truck, excavator and shovel sizes appropriately to ensure that productivity is not reduced due to a poor truck fill factor. This could be created by pairing a large truck with a small shovel/excavator.

On the other hand, for front-end loaders these specifications can be critical. Should extra loading height be required, it is possible to obtain a high-lift option. An example of the difference a high-lift option makes for a Cat 992K front-end loader is given below (Table 7.10).

TABLE 7.10
Example of front-end loader reach and truck loading height.

Front-end loaders	Load height (m)	Reach (m)	Bucket size (m^3)
Cat 992K standard	4.6	2.1	10.7
Cat 992K high lift	5.5	2.2	10.7
Trucks	Load height (m)	Width (m)	Payload (t)
Cat 777F	4.2	6.1	91
Cat 785D	5.0	7.1	133
Cat 789C	5.2	7.7	177
Cat 793	5.9	7.6	218

Weight

Weight varies by the bucket size of a machine. Table 7.11 shows the approximate tonnes of machine weight by bucket size for the different machine types.

Shovel-type loaders

Loader types, characterised by individual shovels including dozers, front-end loaders, rope shovels and

TABLE 7.11
Weight to bucket capacity ratio.

Unit	Capacity (t/m^3)
Rope shovel	23 - 44
Hydraulic excavator	16 - 22
Front-end loader	8 - 10

hydraulic excavators, are detailed below. An example of shovel productivity calculation is provided in this section.

Front-end loaders

The front-end loader is a self-propelled crawler or wheeled machine with an integral front-mounted bucket that loads with the motion of the machine. Crawler loaders are usually very small and generally not used for mine production applications. The main advantage of the front-end loader is its mobility. Even if it is not the primary production unit on a mine site, there will almost certainly be one or two units in backup or ancillary roles. Front-end loaders are primarily used to load trucks; however, they are also used in a load-haul-dump operations, for example, at ore stockpiles and crushers.

Front-end loaders operate more efficiently in very well blasted or stockpiled material because they are limited in their breakout force. This restricted digging envelope also sets a ceiling on the safe bench height under which a front-end loader can work.

Rope shovels

The standard rope shovel is a crawler-mounted electrically-powered machine with an upper structure capable of 360° rotation. It uses wire ropes to pull a bucket on the end of a dipper stick through the bank. This geometry means that the bucket angle of attack is always the same. The main advantage of a rope shovel is its heavy-duty construction and rugged nature. A rope shovel has a projected life of 100 000 hours or more.

Rope shovels are at their best in heavy-duty conditions with long faces at long-life operations. Almost all rope shovels these days are electric. Diesel and diesel-electric shovels are used in rapidly declining numbers.

Hydraulic excavators

A hydraulic excavator is a self-propelled crawler machine with an upper structure capable of 360° rotation. It excavates by using a bucket fitted to a boom and is powered by hydraulic motors.

The hydraulic excavator is a very flexible machine. Its high breakout force, selective digging and good mobility allow it to be used almost anywhere. Typical lives for smaller machines (up to 8 m^3) are 25 000 hours, while for larger machines, machine life can be 50 000 hours or more.

Unlike rope shovels, hydraulic excavators do not undergo lengthy rebuild sessions. Normal maintenance involves replacing components at set hourly intervals. Good preventative maintenance is particularly important for ensuring high availabilities from hydraulic excavators.

Most hydraulic excavators are diesel-powered, although it is possible to get electrically-powered machines with an obvious loss of mobility.

The main classification for hydraulic excavators is load-and-dump. The front-shovel-fitted excavator can have two bucket styles, as shown in Figure 7.3. The backhoe configuration is common in gold mines and operations requiring a high degree of selectivity. The backhoe bucket is usually slightly smaller than the front-shovel bucket on the same machine. Backhoes are quite flexible and it is possible to load trucks either on the same level or at a lower level to the excavator.

Shovel productivity

Shovel productivity is detailed and calculated in Table 7.12.

Each step in the productivity estimation process and definition of the input parameters is explained below by line number:

1. Equipment name – plus any special characteristics, eg if high-lift bucket is used.
2. Bucket size – rated capacity of the bucket for the equipment in line 1. This is usually the 2:1 heaped capacity.
3. Fill factor – measure of how well the bucket gets filled. Rope shovels and excavators achieve better fill factors than front-end loaders. Table 7.7 gives typical fill factors for the various loading units.
4. Material – description of material type.
5. Bulk density – *in situ* density of the material being loaded.
6. Swell factor – as per tests or table of material characteristics.
7. Bucket load – estimated load that the bucket can carry in BCM.
8. Bucket load – recalculated to tonnes.
9. Nominal truck payload – rated truck payload in tonnes.
10. Calculated passes to fill – estimate of how many bucket loads (passes) are required to fill the truck to its nominal capacity. The number of passes should be an integer for the best match between loading unit and truck. Typically the number of passes should be between four and eight. For short travel times the number of passes can be less, while as travel time increases the number of passes to fill a truck becomes less important.
11. Use passes to fill – some engineers will only use integer values, saying that is all the shovel can deliver. In fact, operators are quite able to deliver less than a full bucket to fill a truck. Another way

TABLE 7.12
Shovel production estimation.

Line no	Name of factor	Units	Calculation
1	Equipment		
2	Bucket size	m^3	
3	Fill factor	number	
4	Material		
5	Bulk density	t/m^3	
6	Swell factor	number	
7	Bucket load	BCM	2 × 3/6
8	Bucket load	t	7 × 5
9	Nominal truck payload	t	
10	Calc passes to fill	number	9/8
11	Use passes to fill	number	
12	Calc truck payload	t	11 × 8
13	Load factor		12/9
14	Time per pass	min	
15	Load time	min	11 × 14
16	Spot time	min	
17	Load + spot	min	15 + 16
18	Efficiency	min/h	
19	Propel factor	%	
20	Presentation factor	%	
21	Productivity	t/h	18/17 × 12 × 19 × 20
22	Productivity	BCM/h	21/5
23	Scheduled hours/ annum	h	
24	Mechanical availability	%	
25	Use of availability	%	
26	Utilisation	%	24 × 25
27	Operating hours/annum	h	26 × 23
28	Production/annum	t	27 × 21
29	Production/annum	BCM	27 × 22
30	Required production/ annum	BCM	
31	Operating hours/annum	h	30/22
32	Units required	number	31/27

Note: BCM = bank cubic metres.

to consider it is that 4.4 passes is an average over different operators – some four-pass loading, others five-pass loading. This factor allows the engineer to choose a philosophy and, therefore, the number of passes for input.

12. Calculated truck payload – estimated average payload that the truck will carry after considering all the above factors.
13. Load factor – percentage of truck filled compared to its nominal or rated payload.
14. Time per pass – time taken for a loading unit to complete one pass. In the case of a rope shovel or excavator, this is the time taken to fill the bucket in the bank, swing the load to the truck, drop the load and then return empty to the face. The time for a front-end loader to complete a pass is similar except for the extra time taken for the machine to manoeuvre. The pass time is measured from the dumping of one pass to the next. Table 7.13 provides some typical values for the various loading units.

TABLE 7.13
Loading times.

Loading unit time per pass (min)	
Front-end loader (large >8 m^3)	0.75
Excavator/shovel	0.50
Excavator/shovel >25 m^3	0.55

15. Load time – time taken to load the truck. There are two approaches that can be taken to calculate this:
    a) Simple – this is easy to understand, but is not 100 per cent rigorous.

       Load time = input passes × time to complete a pass

    b) Rigorous – this method takes into account that on the first pass the bucket should already be spotted. Therefore, the first pass should not be counted. The formula becomes:

       Load time = (input passes – 1) × time to complete a pass

    It is generally advisable to use (a) as it is slightly conservative. Calculation (b) is optimistic as it assumes perfect loading synchronisation between truck and shovel.
16. Spot time – time during which the loading unit has the bucket in place to dump, but is waiting for the truck to move into position. Spot time will depend on the truck driver's ability and the loading system. Double-side loading should almost eliminate spot time. Table 7.14 gives some typical values.
17. Load + spot – addition of load and spot time.
18. Efficiency – measure of how much productive work is done in one operating hour (ie to excavate material). This does not mean that the non-productive fraction is useless, only that the work done is not moving primary production ore or waste. The activities that the efficiency factor allows for are:

TABLE 7.14
Typical spot times.

Conditions	Spot time (min)
Good	0.25
Average	0.60
Poor	0.75

- clean-up by the loading unit or dozer
- crusher and dump slow-downs
- fuelling
- inspections
- loading unit movement
- operator experience
- under trucking
- unusual delays due to weather.

Efficiency is measured either as a proportion of an hour or as the number of productive minutes in an hour. Table 7.15 gives typical values.

TABLE 7.15
Efficiency factors.

Conditions	Proportion	Minutes/hour
Good	0.87	52
Average	0.83	50
Poor	0.75	45

19. Propel factor – accounts for time lost due to movements of the loading units around the mine. The efficiency factor accounts for the normal movements of a loading unit as it moves itself along the face during excavation. This factor depends on the type of loading unit, size of pit and amount of movement required. Typical values might be 0.95 for a rope shovel or 1.0 for a front-end loader.
20. Presentation factor – attempts to account for the time a loading unit must wait for a truck. This area is covered in more detail in the sections on queuing theory and simulation. It can also take into account the priority a loading unit gets for trucks. A primary production unit would probably have a presentation factor between 0.95 and 1.0, while an ancillary unit could be substantially less at maybe 0.80.
21. Productivity – tonnes of production excavated in an operating hour.

    Productivity = efficiency/(load + spot) × truck payload × propel × presentation factor

22. Productivity – BCM of production excavated in an operating hour.
23. Scheduled hours/annum – as discussed in the previous section on 'Scheduling'.
24. Mechanical availability – input value depending on machine type, age and maintenance philosophy. Typical values are shown in Table 7.16.

TABLE 7.16
Mechanical availability.

	New/ good	New/ poor	Old/ good	Old/ poor
Electric shovel	0.92	0.88	0.82	0.75
Hydraulic excavator	0.90	0.86	0.77	0.70
Front-end loader	0.88	0.84	0.75	0.65
Truck	0.90	0.85	0.75	0.65

25. Use of availability – input value depending on the operating philosophy, roster, management efficiency and whether shift change and meal losses are included in reduction hours. Some typical values are given in Table 7.17.

TABLE 7.17
Use of availability.

Conditions	Losses in reduction hours	No losses in reduction hours
Good	0.90	0.80
Average	0.85	0.75
Poor	0.75	0.65

26. Utilisation – operating time divided by scheduled time, also equals mechanical availability × use of availability.
27. Operating hours/annum – the potential operating hours that a loading unit could work in a year, if required.
28. Production/annum – hourly productivity × operating hours.
29. Refer to 28.
30. Required production/annum – input value.
31. Required operating hours/annum – calculated operating hours for shovel fleet to excavate the required production input in the line above.
32. Required units – number of loading units required to achieve required production considering, for example, mechanical availability.

Other equipment

Equipment that does not use a single bucket, including dozers, draglines, bucket wheel excavators and surface miners.

Dozers

Dozers can be either crawler or rubber tyred. Rubber-tyred wheel dozers have better mobility, speed and manoeuvrability and are suited to applications with long push distances with loose lightweight materials on level or downhill grades. These conditions are typically found at coal stockpiles or in a pit around shovels or excavators. The crawler-mounted dozer is usually fitted with a blade and a ripper. Typical specifications for a dozer are given in Table 7.18. These machines have many uses around the mine site, including:

- clearing and grubbing
- drainage construction
- haul road formation
- push loading scrapers
- pushing up and loading machines
- restoration and clean-up
- ripping and loading.

TABLE 7.18
Crawler dozer specifications.

	Flywheel power (kW)			
	100	200	300	500
Machine weight (t)	15	30	50	75
Ground contact area (m^2)	2.5	3.5	4.5	5.5
Blade width (universal type) (m)	3.2	4.2	5.0	6.0
Average life (h)	40 000	40 000	48 000	48 000

In the context of this section, the dozer can be considered an excavating and loading machine when it pushes loose or loosened material to a loading point. The rate at which a dozer can move material depends on the specifications, ground slope, pushing distance and material characteristics.

Figure 7.4 gives some curves for estimating the approximate production that can be expected from different sized dozers.

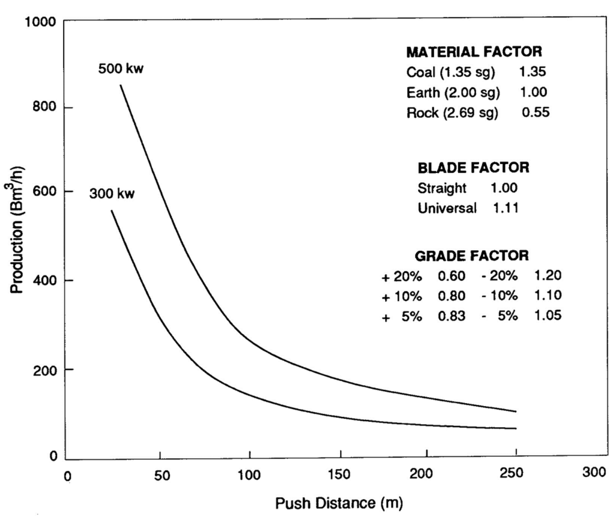

FIG 7.4 - Crawler dozer push capacity (in bank cubic metres; sg = specific gravity).

Draglines

Dragline bucket sizes range from less than 1 m^3 to over 150 m^3, although the commonly used size for stripping overburden in coalmines is 40 to 60 m^3. The smaller machines are crawler-mounted and diesel-powered. Machines above about 5 m^3 are base-mounted with walker feet. Above about 15 m^3 draglines are electrically powered with AC mains supply driving DC generators, which in turn power DC motors for the essential drive functions of drag, hoist, swing and return propel.

Most walking draglines comprise a central base on which the machine normally operates and two walking shoes that work on a cam shaft to raise and lower the base, while also advancing the machine on each step. Typical machine specifications are given in Table 7.19.

The dragline operates by the drag machinery pulling the bucket along the ground towards the operator. The full bucket is hoisted, while the drag cable keeps the bucket from dumping. When the bucket is clear of the ground the boom is swung, while the bucket continues to rise to the desired dump height. As the dump position is reached, the tension on the drag cable is released to dump the load, while coming to a smooth stop. The machine then swings back to the excavating position.

The cycle time can be estimated by first calculating the theoretical time for completing the above cycle components from the manufacturers' specifications. Then add delay factors for acceleration and deceleration, angle of swing, depth of cut, excavatability of material and experience of the operator. Some typical cycle times are given in Table 7.19. It should be noted, however, that draglines have more operating variables than other loading machines and these variables significantly affect the performance of the machine. It is advisable to perform a computer simulation for each particular application. Figure 7.5 is a diagram defining the dragline variables of digging depth, dumping height and dump radius.

Example

An example of dragline productivity calculation is given below, assuming the following conditions:

Material	weathered sandstone
Density	2.2 t/m^3
Swell	1.3
Machine size	50 m^3
Swing angle	90°
Base cycle time	1.0 min
Bucket fill factor	0.85

The productivity per hour can be calculated as follows:

Material per bucket	= 50 m^3 × 0.85/1.3 = 32.7 m^3
Cycles per hour	= 60 min/h/1.0 min/cycle = 60 cycles/h
Productivity factor	= 60 cycles/h × 47 min/h/60 min/h = 47 cycles/h

TABLE 7.19
Dragline specifications.

	Capacity (m^3)						
	0.75	2.5	5.0	15	50	75	135
Weight (t)	25	80	150	900	3 200	6000	9000
Maximum suspended load (t)	2.4	6	11	40	135	225	415
Power (kW)	90	165	300	1600	4000	7500	13 500
Boom length (m)	15	22	30	80	95	100	100
Dumping radius (m)	14	20	27	65	85	95	95
Dumping height (m)	7	10	15	40	35	40	40
Digging depth (m)	10	12	18	55	40	50	55
Cycle time[a]							
Easy digging (min)	0.40	0.55	0.60	0.65	0.75	0.75	0.75
Medium digging (min)	0.45	0.60	0.65	0.80	0.85	0.85	0.85
Hard digging (min)	>0.50	>0.70	>0.80	>0.95	>1.00	>1.00	>1.00
Walking speed (km/h)	2.5	1.5	1.2	0.27	0.22	0.22	0.20
Average life (h)	14 000	16 800	28 000	70 000	180 000	180 000	180 000

a. 90° swing, optimum dig depth, casting specifications and typical overburden material characteristics.

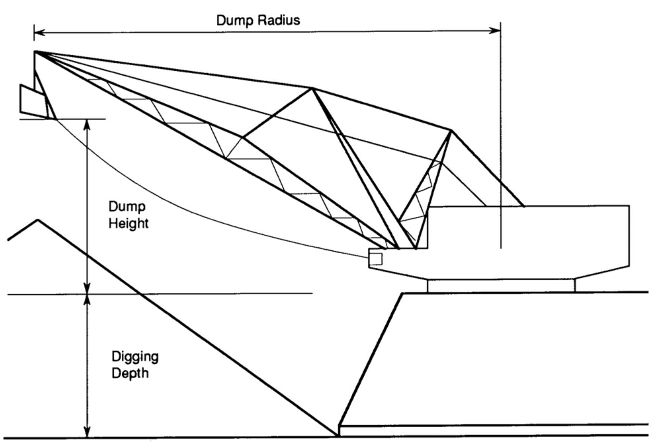

FIG 7.5 - Dragline range diagram.

Production per hour = 47 cycles/h × 32.7 m^3/cycle = 1537 bank cubic metres (BCM)

Assume that the dragline is worked 19 h/d, 7 d/wk for 46 w/a, then:

Scheduled work days	= 46 wk × 7 d = 322
Less provision for disruptions	= 10 d
Available work days	= 312 d
Total available production hours per annum	= 312 d × 19 h = 5928
Hence production per annum	= 5928 h × 1537 m^3/h = 9.1 M m^3

Bucket wheel excavators

Bucket wheel excavators are suitable for removing large volumes of soft overburden or ore that is free of hard bands or boulders and is free-flowing when excavated. They have been used to mine brown coal and weathered overburden with compressive strengths up to about 10 MPa.

Essentially, the machines comprise a rotating wheel with buckets attached. The buckets have teeth on the leading edge that cut the material. The bucket wheel ranges in both vertical and horizontal planes over the face. The excavated material drops onto a belt, which conveys the material away from the face for transport from the mining area by rail, truck, or more commonly, belt conveyor. Typical specifications for bucket wheel excavators are given in Table 7.20. The 12.5 m diameter machine includes an 85 m conveyor bridge, which increases the cost of the machine, but improves relative output by reducing the number of bench conveyor moves.

Determination of the output from a bucket wheel is a complex task and is highly dependent on the site conditions. The machines also tend to be customised to a particular application with regard to power and gearing, making comparison between different machines difficult. The theoretical production in LCM/h from a bucket wheel is given by:

$$Q = W \times N \times C \times 60$$

where:

Q	theoretical output (m^3/h)
W	wheel speed (rpm)
N	number of buckets per wheel

TABLE 7.20
Bucket wheel excavator specifications.

	Wheel diameter (m)			
	5.5	7.5	10	12.5
Theoretical capacity (LCM/h)	1200	2500	4500	6200
Effective output (BCM/h)	480	1150	1950	3250
Bucket volume (m^3)	0.25	0.60	1.20	2.35
Machine weight (t)	170	415	850	3 000
Power				
Bucket drive (kW)	200	500	900	1250
Connected load (kVA)	400	1000	1850	4000
Bench dimensions				
Bank height (m)	9.0	14.0	19.0	30.0
Below grade reach (m)	5.0	7.0	10.0	5.0
Cut width (m)	13.0	19.0	26.0	50.0
Bench conveyor width (mm)	900	1350	1600	1800
Typical usage (hours/a)	3500	3500	3500	3500
Average life (year)	20	20	20	20

Notes: BCM = bank cubic metres; LCM = loose cubic metres.

C bucket capacity (m^3)

S swell factor

The bucket fill factor typically equals one for the types of material commonly excavated.

The typical effective output in BCM/h for each machine specified is given in Table 7.20. The figures include the effect of material swell, excavator and conveyor availability, mining efficiency and conveyor moves.

Surface miners

Several designs of surface miners are applied in open cut mining where medium-strength materials (up to 50 MPa) are to be excavated. Their advantages are eliminating the need to drill-and-blast, mining with minimal dilution, eliminating the need for primary crushing and providing a continuous flow of material from the face. However, pick wear and power consumption can be high in hard abrasive material, making them more suitable for coal mining applications in similar materials.

There are two main types of cutting head used on the machines. One type is similar to underground continuous coalminers and has an oscillating drum head with picks attached. Cut material is fed onto a central conveyor by gathering arms at floor level for discharge at the rear of the machine. The other type of head has some features in common with the bucket wheel excavator, including that the cutting is done by a drum with bucket-type segments across the width of the machine. Buckets lift the cut material and discharge it onto a conveyor for delivery at the rear of the miner. This can be either loaded directly onto a truck or conveyor or indirectly by windrowing where the material will be picked up later.

Typical cut depth is 0.3 m. Because surface miners can take time to turn and set up at the end of each cut, a minimum of about 600 m of strike is required. Consideration needs to be taken into account of any hard zones that might be cut across on a bench. Otherwise pick wear can be high.

HAULAGE EQUIPMENT

The haulage equipment covered in this section includes dump trucks, scrapers and conveyors. With the exception of conveyors, which provide continuous haulage, the equipment is cyclical and the estimation of the number of units required is dependent on accurate estimation of the productivity of a haulage unit.

Truck types

Trucks can be classified as:

- dumping
  - rear dump
  - bottom dump
  - side dump
- steering
  - front wheel steer
  - articulated steer
- drive
  - rear wheel
  - all wheel
  - centre drive
- axles
  - two axles
  - three axles
  - more than three axles
- power train
  - mechanical
  - electric.

The following sections discuss some of the distinctions among these types.

Dumping configurations

Rear and bottom dumps only are discussed in this section.

Rear dumps

In these units the body is mounted on the truck frame. Dumping is carried out by a hydraulic hoist system raising the body to greater than 45°. These are very flexible units capable of handling all types of material. They have good gradeability and are easily manoeuvred. They are the most common haulage unit.

The standard haul unit has two axles with two wheels on the front axle and four wheels on the rear axle. The rear wheels are usually the only ones driven. Three-axle trucks are less common in mines, but are used for on/off highway hauls. An example is a coal unit loaded in-pit, but which does most of its hauling on good roads at high speed. The extra axle reduces the tyre loadings and so improves tyre life at high speeds.

Bottom dumps

These units provide faster dump times and higher payloads for the same engine horsepower, but at the cost of gradeability and manoeuvrability. In general, their use is in strip coal mines where the ramp gradients are kept at five per cent or less.

Articulated steering

Articulated units tend to be smaller and of lighter construction. Maximum size would be of the order of 50 t. Their main application is in wet and poor underfoot conditions. Their lighter construction results in a shorter life.

Power train

The two basic power trains in large haul units are electric and mechanical.

Electric

Electric units use a generator-alternator, driven by a diesel engine to power electric motors in the hubs of the rear wheels. Retarding (braking) is provided by working the system backwards. The electric motors are used as generators with the electric power created as feed to large resistor banks. This retardation will reduce the speed to a few kilometres per hour where the disc-drum braking system (service) can be used.

The engine of the electric drive units runs at set revolutions per minute (rpm) all the time. This means that it uses substantial fuel even under braking conditions.

Electric drive manufacturers counter that their units are cheaper to maintain. Electric drive trucks can be matched to a particular haul cycle and may outperform mechanical drive trucks in this situation; however, mechanical drive trucks are more flexible.

Mechanical

These operate in much the same manner as a car with an automatic gearbox. Trucks are available in a range of sizes up to a payload of more than 300 t.

The main advantages of mechanical drive units lie in their better gradeability and lower fuel consumption.

Truck specifications

This section discusses truck specifications that can be important in analysing truck shovel systems. Some important specifications are:

- payload, net vehicle weight and maximum gross vehicle weight
- dimensions
- rimpull curve
- braking curve
- drive system type
- power
- tyres.

Payload, net vehicle weight and gross vehicle weight

The amount of useful material carried by trucks is measured either as bank cubic metres (BCM) or tonnes.

It is the weight of the load that is important in terms of the vehicles performance. Manufacturers either define the capacity of their trucks in terms of nominal payload (tonnes) or as a maximum gross vehicle weight (GVW).

The GVW system is very sensible as it accounts for the changes in net vehicle weight (NVW) due to specification differences between units:

$$\text{Payload} = \text{GVW} - \text{NVW}$$

As the above formula shows, payload is the difference between GVW and NVW. The NVW for the same trucks at different mines can be quite large. This is because different options selected by each operation include tyres, wear packages, air conditioning, size of fuel tank and body size.

When calculating productivities we are usually only interested in average payloads. However, it must be understood that payloads follow a distribution of some sort. This will not necessarily be a normal distribution. However, as a rule-of-thumb it can be assumed that a standard deviation will be ten per cent of the average payload and that payloads will be normally distributed.

Therefore, assuming an average payload of 172 t and a ten per cent standard deviation (17.2 t) with a normal distribution, the truck payloads could be expected to range from 138 t to 206 t for 95 per cent of loads (Figure 7.6).

The two equations below give the calculations for maximum and average payloads:

$$\text{Maximum payload} = \text{average payload} + \text{number of SD} \times \text{SD as a per cent of mean}$$

$$Average\ payload = \frac{Maximum\ payload}{\left(1 + \frac{\%SD}{100} \times number\ of\ SD\ from\ mean\right)}$$

where:

SD standard deviation

Example

What will be the average payload if the maximum GVW of a truck is 317 t and the NVW is 130 t? Assuming a normal distribution, a standard deviation of seven per cent of average payload and acceptance of five loads in 100 being more than the maximum, the maximum load is:

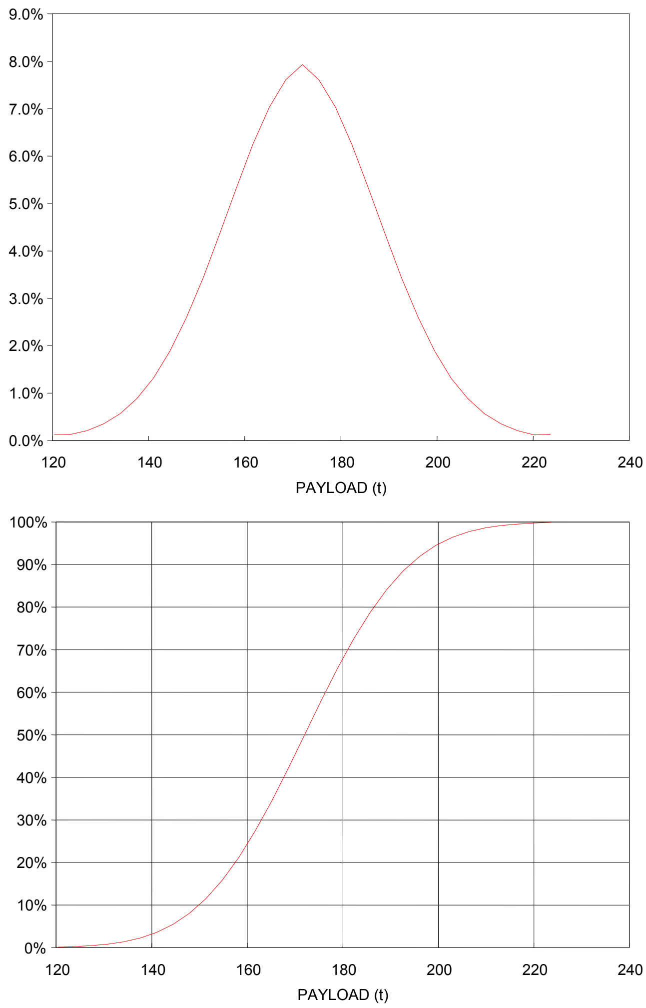

FIG 7.6 - Payload distribution. Top: distribution of payload. Bottom: cumulative distribution of payload. Mean = 172 t, standard deviation = 17.2 t.

Maximum load = 317 - 130
= 187 t

The number of standard deviations from the mean to ensure only five per cent of loads exceed maximum payload is 1.65.

Average payload = 187/(1 + (7/100 × 1.65))
= 168 t

Dimensions

Some dimensions and their uses are:

- body size – loading equipment compatibility, size versus payload
- height – truck bay and dump design and suitability, loading equipment compatibility
- length – parking and maintenance area requirements
- raised body height – maintenance bay and power line height
- width – road and crusher width design, parking and maintenance area required.

Dimensions will change depending on tyre type, body size and whether the unit is empty or loaded.

A good scaled diagram is very useful. Combined with one for the loading equipment and careful scaling on a photocopy, they are invaluable for ensuring units are a good match.

Equipment manufacturers are a good source of data for initial analysis to match loading and hauling equipment.

Power

Power is usually quoted as either gross power or flywheel power. Gross power is the maximum power that can be produced by the engine. Flywheel power is gross power less the power used by ancillary equipment on the truck. This equipment includes fan, air cleaner, alternator, water pump, fuel pump, oil pump and muffler. Flywheel power is typically 90 to 95 per cent of maximum power.

Operating altitude can affect the power output of an engine because there is less oxygen with increased elevation. Most equipment these days is turbo-charged and the effect of altitude will not be noticed until above 2300 m.

Performance charts

Manufacturers' performance charts provide the maximum speed of a truck under given total resistance and truck weight. It also gives information on rimpull available. Figure 7.7 is a typical chart for a mechanical drive unit. To read the chart the following rules apply:

- read down the appropriate weight line to the total resistance line (percentage)
  - total resistance is the sum of the rolling resistance (RR) and the grade resistance
  - RR is always positive
  - grade resistance will be positive for an uphill haul and negative for downhill
- from the point where the total resistance line and weight line meet move horizontally until the speed curve is met
- then on the x-axis read the maximum speed attainable under the conditions.

Example

What is the maximum speed for a loaded unit on an uphill ten per cent grade with a rolling resistance of two per cent?

Total resistance = 2% + 10%
= 12%

Figure 7.8 shows maximum speed is 12 km/h in first gear.

Rimpull

Rimpull is the force available at the tyre that is required to move the vehicle forward. This force is limited by traction. The difference between the rimpull required to overcome total resistance and the available rimpull

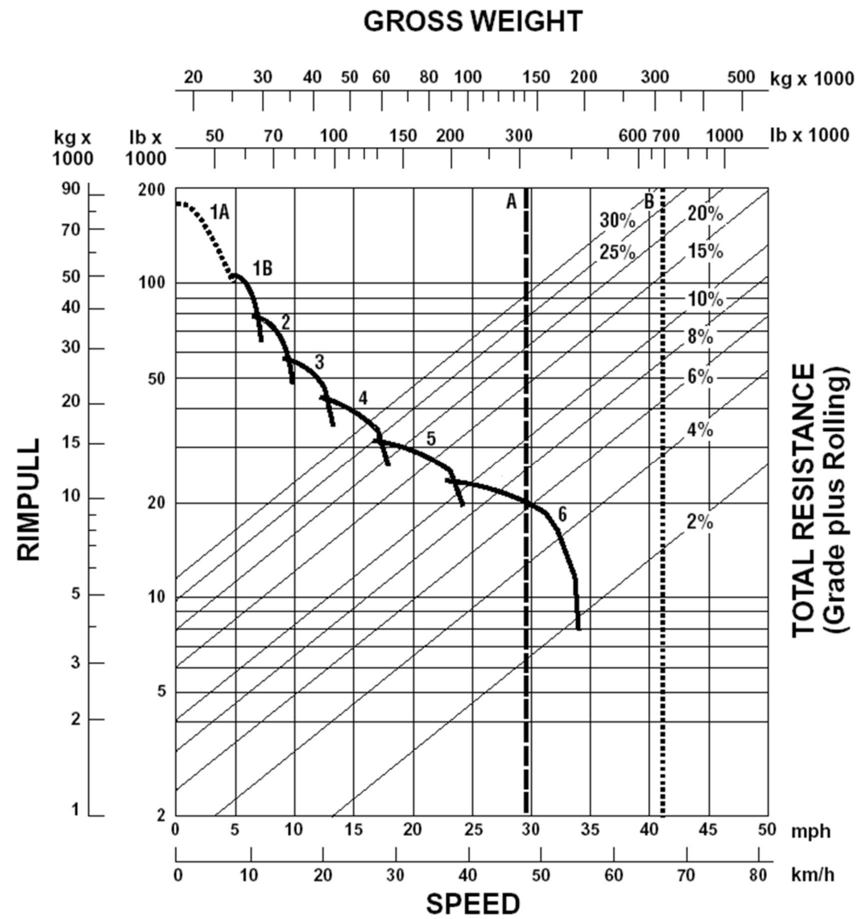

FIG 7.7 - Sample rimpull curve.

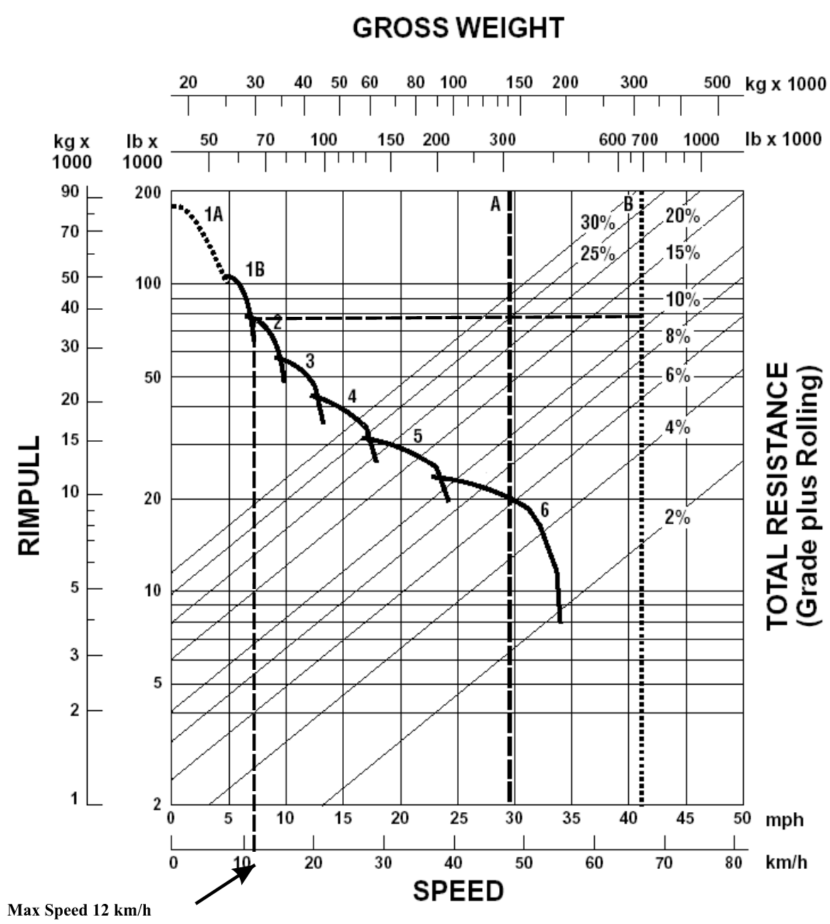

FIG 7.8 - Sample use of rimpull curve to find maximum speed.

determines vehicle acceleration. Rimpull from the charts is useful for input into computer simulations. To pick rimpull from the chart:

- choose velocity and move vertically up until the speed curve is hit
- move horizontally from that point toward the rimpull axis and read off the value.

Example 1

What is the rimpull of the vehicle at 30 km/h? Figure 7.9 shows the rimpull is 15 000 kg force.

The rimpull curve can also be used to estimate power train efficiency using the following formula.

$$\text{Efficiency} = (\text{Rm} \times \text{V})/(\text{Power} \times 367)$$

where:

Rm rimpull (kg)

V velocity (km/h)

Power engine power (kW)

Example 2

What is the power train efficiency of the vehicle at 30 km/h if the gross engine power is 1342 kW?

$$\text{Efficiency} = (15\,000 \times 30)/(1342 \times 367)$$
$$= 0.91 \text{ or } 91\%$$

Although each manufacturer's charts differ to some extent, the general principles are the same. Note that on electric trucks the speed curve will be smooth. Figure 7.10 is an example of an electric truck rimpull curve.

Braking/retarding charts

The braking/retarding curves are read in a similar way to the performance chart. In this case the speed obtained will be the maximum speed that the vehicle can travel and still stay within the braking envelope of the unit.

Different braking systems have different characteristics. For example, the curves shown in Figure 7.11 are the braking curves for the Cat 789. The Cat 789 uses disc brakes as the primary braking system. The longer the period of braking required, the hotter and less efficient the brakes become. Caterpillar supplies a series of curves for various distances on grade. Thus the maximum speed on a 450 m grade distance will be higher than that for 1500 m.

Example

What is the maximum speed for a loaded unit on a ten per cent downhill grade with two per cent RR over grade distances of 450 m and 1500 m?

Total resistance = 2% + (-10%)

= –8% or 8% favourable

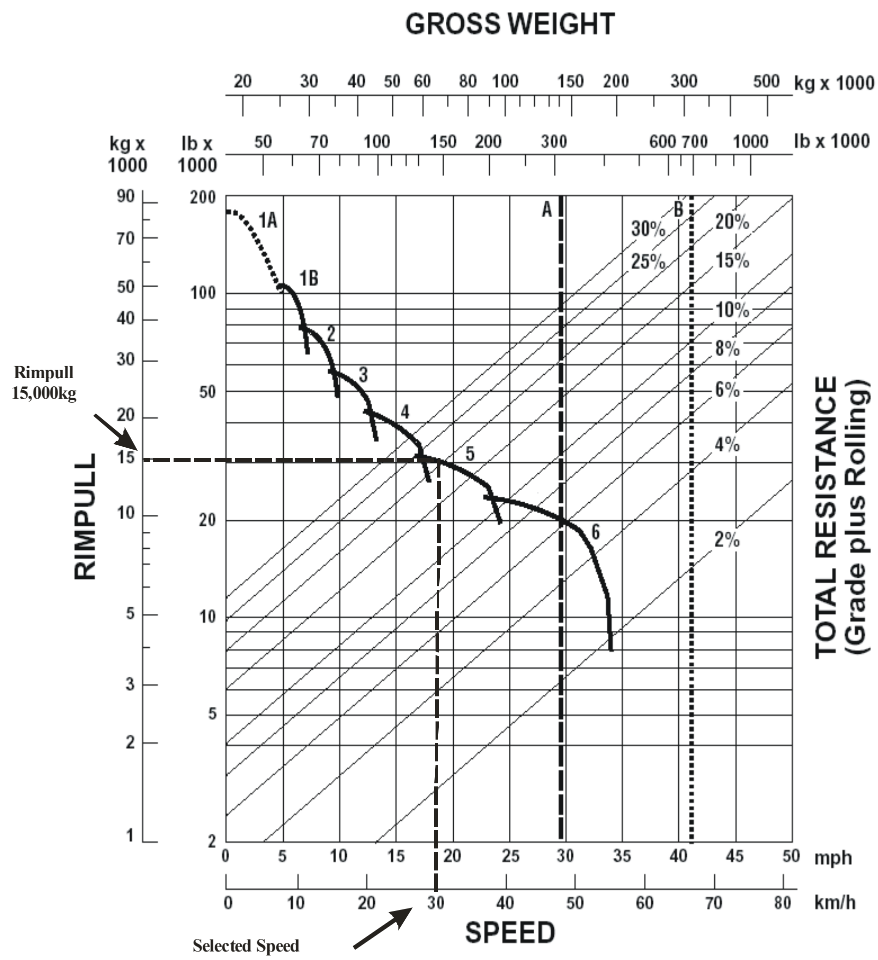

FIG 7.9 - Sample use of performance chart to obtain rimpull.

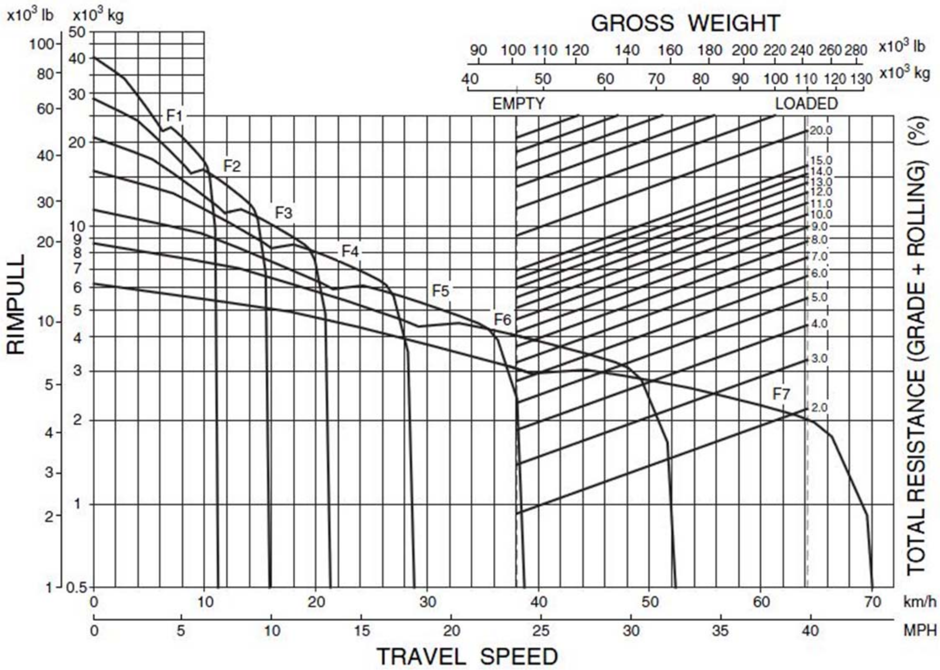

FIG 7.10 - Sample electric rimpull curve.

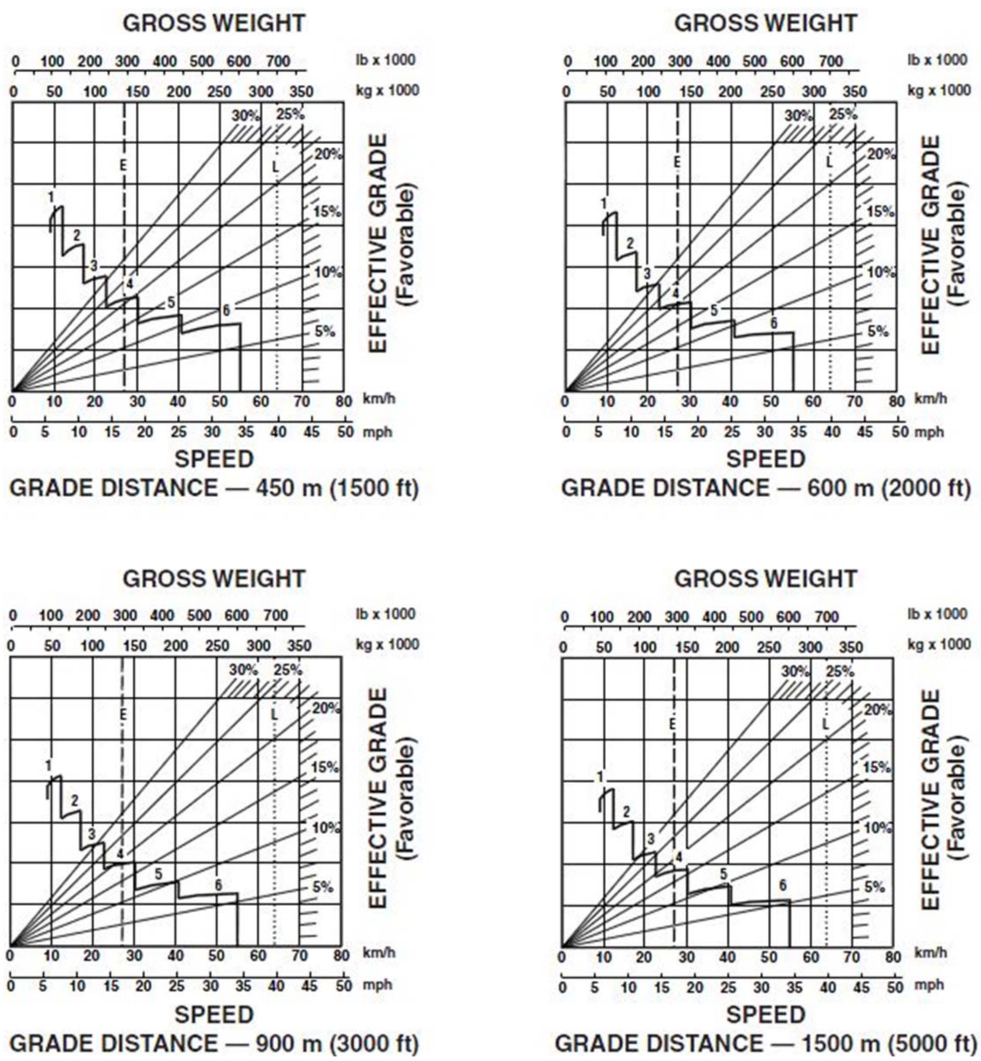

FIG 7.11 - Sample retarding curves for the Cat 789 truck.

Maximum speed on a 450 m grade is 41 km/h in fifth gear or 30 km/h in fourth gear. Over a 1500 m grade distance the maximum speed drops to 22 km/h in third gear (Figure 7.12).

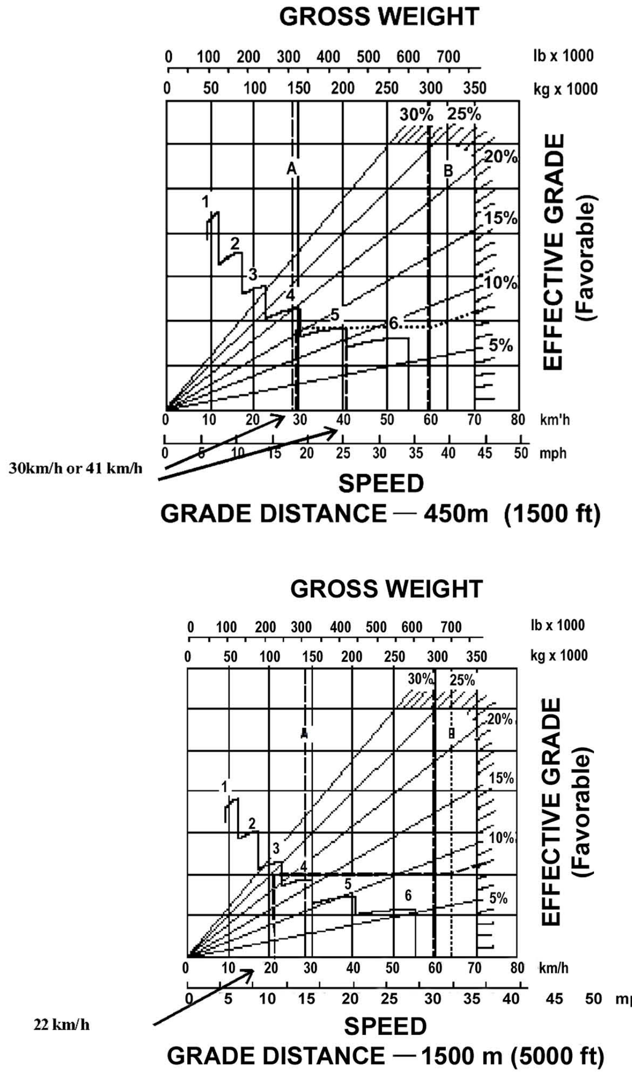

FIG 7.12 - Use of retarding curve.

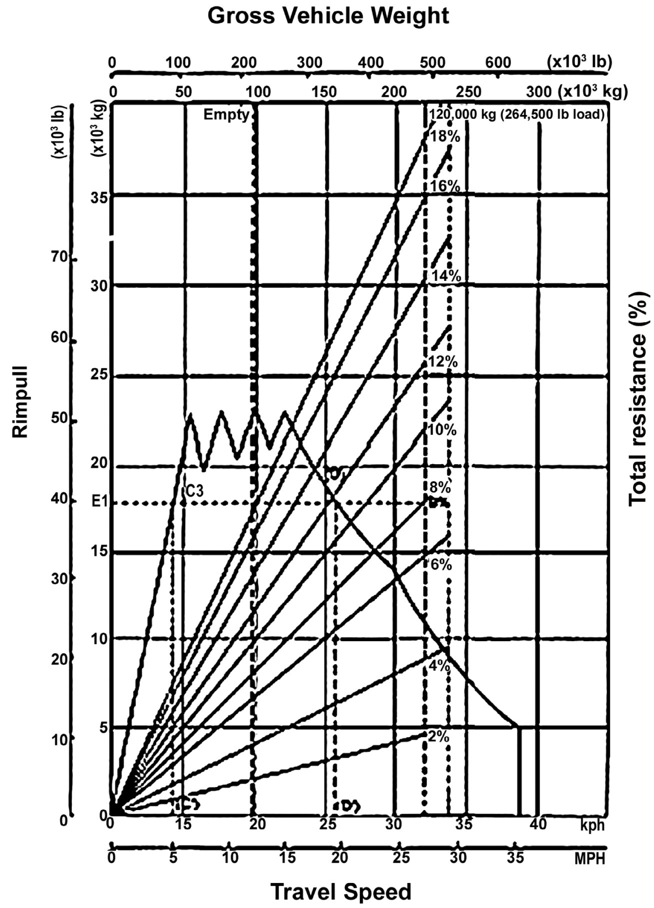

FIG 7.13 - Electric retarding curve.

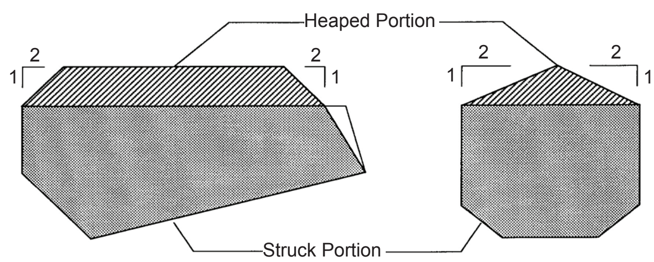

FIG 7.14 - Tray size measurement.

Electrical retarding systems

The electrical retarder acts to control the acceleration of a truck or to slow it down to the point where the truck's service brakes can halt the vehicle. If a truck moves outside the retarding curve, the truck will start to accelerate. The secondary braking system must be used to bring the unit back into the retarding envelope before the retarder will control speed again. Figure 7.13 is an example of an electric retarding system chart.

Tray size

Tray size and weight will depend on material loose density, sizing and abrasiveness. Tray sizes, like bucket sizes, are defined in terms of struck and heaped (2:1) capacity. Figure 7.14 demonstrates the standard method of measuring tray size. A standard tray would be designed for unabrasive material of 1.7 t/m^3 to 1.8 t m^3 loose density. On Figure 7.14, numerals 1 and 2 are ratios representing the slope angles of the load.

The actual tray size and wear package requirements need to be determined in consultation with the distributor and manufacturer. Too small a tray will under-load the vehicle, with a potential large opportunity cost. Too large a tray will mean carrying around metal not required as well as potentially overloading the unit, with risk to tyres, parts and warranty.

Tyres

Tyres are one of the most important elements of truck cost and of its successful operation. There are two basic tyre types – bias and radial.

In general, radial tyres are becoming more popular in open pit mines. Potential advantages include:

- good flotation
- good grip
- long tyre life
- low fuel consumption
- smooth ride.

There are situations in which radials have problems. These include:

- high gradients, where dumping is difficult
- poor floor conditions, where sidewall cuts are common
- sharp switchbacks, where tyre flexibility can cause rubbing on the flange.

Operating tyres at temperatures above their capability will result in ply separations and other temperature-related failures. Heat is generated in tyres as they roll and flex. If heat is created faster in the tyre than it can be expelled to the atmosphere, then heat build-up occurs. In the extreme, this can reverse the vulcanisation process and initiate tyre failure. Even if this critical level is not reached, the tyre loses strength as temperature increases and becomes more liable to damage from braking, cornering, impact and cuts.

The tonne kilometre per hour (TKPH) formula is designed to predict such potential failure. Temperature is a function of speed, load and the time that tyres are not working and are allowed to cool:

$$\text{TKPH} = \text{mean tyre load (t)} \times \text{average hourly speed (km/h)}$$

The mean tyre load is the average weight carried on the tyre over the haul cycle. It is the average of the empty and loaded machine weight on each tyre:

$$\text{Mean tyre load} = (\text{tyre load empty} + \text{tyre load full})/2$$

The average hourly speed is the average speed (km/h) of the unit over a one-hour period. It is important to realise that the TKPH rating should *not be exceeded for any one hour.* Just because the overall average for the shift is within specification does not mean that damage could not have occurred over one particular hour.

Example

What is the TKPH of a 136 t class truck with an empty weight of 100 t, loaded weight of 236 t and operating at an average hourly speed of 18 km/h?

$$\begin{aligned}\text{Mean tyre load} &= (100 + 236)/2/6 \\ &= 28\text{ t} \\ \text{TKPH} &= 28 \times 18 \\ &= 504\text{ t.km/h}\end{aligned}$$

A truck of this class could use either 33.00-51 bias ply or 33.00R51 radials in E-4 class.

Typical bias tyre TKPH ratings for this size range from 394 to 533, while radial tyres offer ratings of 480 to 820.

Based on the required work and tyre selected, tyre life can be estimated. Typical tyre life ranges between 4500 and 5000 h/a. However, tyre life can reduce significantly if haul road conditions are not well maintained.

Trolley assist

Trolley assist for trucks uses an overhead electric power line to supply power directly to the hub motors. A trolley pole or pantograph system is used to collect the electricity from the overhead power line. The truck can be used in either the trolley assist mode or in normal diesel mode. The main advantages for trolley assist are:

- decreased diesel engine maintenance – less work carried out by motor
- decreased fuel consumption – uses electricity rather than fuel
- increased productivity – faster speed on loaded uphill section
- improved deep pit performance – can run for longer periods up ramp.

The main use for trolley assist is in deep pit operations where fuel is very expensive while electric power is cheap. Disadvantages include:

- capital cost of truck modifications
- cost of initial power distribution and maintenance
- relocation of power system as pit expands.

Truck productivity

Calculation of truck productivity is discussed with reference to the worksheet shown in Table 7.21. Steps and definition of the input parameters are explained.

It is important to remember that truck productivity depends on haul profile, which affects the truck cycle time per load. This is discussed in more detail in the section on 'Truck travel time'.

Each step in the truck productivity estimation process and definition of the input parameters is explained below by line number:

1. Equipment name – plus any special characteristics, eg if high-lift bucket is used.
2. Nominal payload – rated capacity of the payload of the equipment in line 1.
3. Material – description of material type.
4. Bulk density – *in situ* density of the material being loaded.
5. Swell factor – from material characteristic tables or tests.
6. Bucket load – estimated bucket load that the loading unit can carry in BCM. This is from line 8 in the shovel productivity estimation process sheet.
7. Calculated passes to fill – estimate of how many bucket loads (passes) are required to fill the truck to its nominal capacity. The number of passes should be an integer for the best match between loading unit and truck. Typically, the number of passes should be between four and eight. For short travel times the number of passes can be less, while as travel time increases, the number of passes to fill a truck becomes less important.
8. Use passes to fill – some engineers will only use integer values, saying that is all the shovel can deliver. In fact, operators are quite able to deliver

TABLE 7.21

Truck productivity estimation sheet.

Line no	Factor	Units	Calculation	Input data or result
1	Equipment			Cat 789
2	Nominal payload	t		172.0
3	Material			Granite
4	Bulk density	t/m^3		2.65
5	Swell factor			1.50
6	Bucket load	t		38.87
7	Calc passes to fill		2/6)	4.4
8	Use passes to fill			4.0
9	Calc truck payload	t	8 × 6	155.5
10	Load factor		9/2	90.4%
11	Time per pass	min		0.50
12	Load time	min	11 × 8	2.00
13	Spot time	min		0.80
14	Dump time	min		1.00
15	Fixed time	min	12 + 13 + 14	3.80
16	Travel time	min		12.00
17	Wait time	min		0.00
18	Cycle time	min	15 + 16 + 17	15.80
19	Efficiency	min/h		50.0
20	Queue factor			1.00
21	Productivity	t/h	19/18 × 20 × 9	492.0
22	Productivity	BCM/h	20/4	185.7
23	Scheduled hours/annum	h		7580
24	Mechanical availability			82.0%
25	Use of availability			85.0%
26	Utilisation		25 × 24	69.7%
27	Operating hours/annum	h	26 × 23	5283
28	Production/annum	t	27 × 21	2 599 498
29	Production/annum	BCM	27/22	980 943
30	Required production	BCM		10 000 000
31	Required operating hours	h	30/22	53 859
32	Required units		30/29	10.2

Note: BCM = bank cubic metre.

less than a full bucket to fill a truck. Another way to consider it is that 4.4 passes is an average over different operators – some four-pass loading, others five-pass loading. This factor allows the engineer to choose a philosophy and, therefore, the number of passes for input.

9. Calculated truck payload – estimated average payload that the truck will carry after considering all the above factors.
10. Load factor – percentage of truck fill compared to its nominal or rated payload.
11. Time per pass – time taken for a loading unit to complete one pass. In the case of a rope shovel or excavator this is the time taken to fill the bucket in the bank, swing the load to the truck, drop the load and then return empty to the face. The time per pass for a front-end loader is similar except that it includes extra time for the machine to manoeuvre. The pass time is typically measured from the dumping of one pass to the next. Table 7.13 provides some typical values for the various loading units.
12. Load time – time taken to load the truck. There are two approaches that can be taken to calculate this:
    - Simple – easy to understand but not 100 per cent rigorous:

      Load time = input passes × time to complete a pass

    - Rigorous – takes into account that on the first pass the bucket should already be spotted. Therefore, the first pass should not be counted. The formula becomes:

      Load time = (input passes – 1) × time to complete a pass

13. Spot time – time during which the loading unit has the bucket in place to dump, but is waiting for the truck to move into position. Spot time will depend on the truck driver's ability and the system of loading. Double-side loading should almost eliminate spot time.
14. Dump time – time taken for the truck to manoeuvre and dump its load either at a crusher or dump. Table 7.22 gives some typical values.

TABLE 7.22

Typical dump times.

Conditions	Rear (min)	Bottom (min)
Good	0.70	0.30
Average	1.00	0.60
Poor	1.50	1.50

15. Fixed time – sum of load, spot and dump time. It is called 'fixed' because it is essentially invariable for a truck and loading unit combination.
16. Travel time – time taken to haul and return the load. This is discussed in more detail in the section on 'Truck travel time'.

17. Wait time – time the truck must wait before being served by the loading unit. In general, this number will not be known and is handled by other methods (eg queuing theory or simulation). If this number is known, because of time studies, it can be added in here.
18. Cycle time – round trip time for the truck. It is the sum of fixed, travel and wait times.
19. Efficiency – measure of how much productive time is achieved in one hour of operating time (ie to excavate material). This does not mean that the non-productive time is useless, only that it does not produce primary material movement. The sort of activities that the efficiency factor includes is:
    - clean-up by the loading unit or dozer
    - crusher and dump slow-downs
    - fuelling
    - inspections
    - loading unit movement
    - operator experience
    - under trucking
    - unusual delays due to weather.

    Table 7.15 gives typical values of efficiency. Efficiency is measured either as a proportion of an hour or as the number of productive minutes in an hour.
20. Queue factor – accounts for time lost due to queuing. It is another measure of wait time. In general, it should be left at 1.0 unless some estimate of queuing losses is available. A typical queuing factor might be 0.90.

Productivity – tonnes of production hauled in an operating hour

21. Figure 7.15 shows how the various factors interact to affect productivity.

Productivity = efficiency/(cycle time) × truck payload × queuing factor

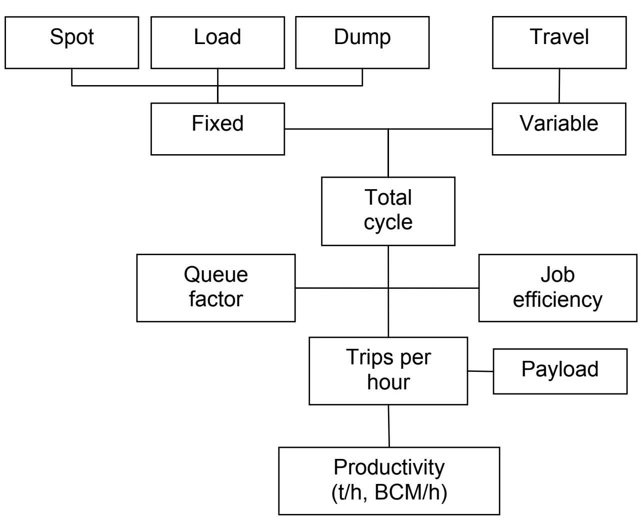

FIG 7.15 - Factors influencing truck productivity.

22. Productivity – BCM of production hauled in an operating hour.
23. Scheduled h/a – as discussed in the previous section on 'Scheduling'.
24. Mechanical availability – input value depending on machine type, age and maintenance philosophy. Typical values are shown in Table 7.16.
25. Use of availability – input value depending on the operating philosophy, roster and management efficiency, and whether shift change and meal losses are included in reduction hours. Typical values are given in Table 7.17.
26. Utilisation – operating time divided by scheduled time; also equals mechanical availability × use of availability.
27. Operating hours/annum – potential operating hours that a trucking unit could work in a year, if required.
28. Production/annum – hourly productivity × operating hours.
29. Refer to 28.
30. Required production/annum – input value.
31. Required operating hours/annum – calculated operating hours for truck fleet to move the required production input in the line above.
32. Required units – number of trucks required to achieve required production, considering both mechanical availability and use of availability.

The use of spreadsheets for calculating productivity has a number of advantages, including the ability to:

- carry out sensitivity analysis
- compare different truck/loading unit combinations, by copying the last column
- carry out back-analyses to check various factors.

To make the best use of this system, it is necessary to develop a set of factors that are suitable for each operation.

Truck travel time

The four ways of calculating travel time – that most important factor of truck productivity– include:

1. time study
2. rimpull curves
3. empirical
4. computer simulation.

With the availability of cheap personal computers, simulation is now open to all. There are a number of good and readily available programs such as Talpac by Runge Associates and FPC by Caterpillar. These not only calculate travel time, but also go through the whole process of calculating productivity. This section will briefly discusses items 1 to 3.

Haul profile

Before going further, the basic information for estimating travel time needs to be collected. This is called the haul profile and breaks the truck route into sections. Each section has information on its distance, rolling resistance, grade and speed limitations.

Distance

The distance is the one-way distance per section in metres. There is usually only a need to note the loaded sections. The return sections, when the truck is empty, will usually be the same with the exception of the grade, which will be reversed.

Rolling resistance

The RR of the road is measured as a percentage of the vehicle weight. RR is a measure of the force required to overcome the retarding effect between the tyres and the road. It includes the resistance caused by tyre penetration of the ground and tyre flexing. Some typical RRs are given in Table 7.23.

TABLE 7.23
Typical rolling resistances.

	Rolling resistance (%)
Bitumen, concrete	1.5
Dirt – smooth, hard, dry and well maintained	2.0
Gravel – well compacted, dry and free of loose material	2.0
Dirt – dry but not firmly packed	3.0
Gravel – dry not firmly compacted	3.0
Mud – with firm base	4.0
Gravel or sand – loose	10.0
Mud – with soft spongy base	16.0

The typical main mine road would have an RR of two per cent if it is hard and well maintained. On the bench and close to the dump end, the road quality drops and a RR of three per cent could be expected. During wet periods when the road conditions deteriorate, the RR might increase to four per cent. Under very poor conditions, the RR could rise from ten per cent to 16 per cent. This would be only over very small sections of road and hopefully only for short periods. When in doubt, an average RR of three per cent over the whole profile gives reasonable results.

Grade

Grade is the slope of the section, measured as a percentage. Slope is the ratio between the rise of the road and the horizontal length. Therefore, a section of road that rises at 10 m over 100 m has a ten per cent grade.

Speed limitations

Speed limitations are placed on road sections for reasons including:

- operational constraints
- operator capability
- safety.

Many times equipment will have theoretical capabilities beyond practical ones. For example, although a retarding curve suggests that a unit can travel down a grade at 40 km/h, road conditions may mean it is not physically safe to achieve that speed on that section of road. The operators may use a lower gear on that section and, therefore, only achieve a speed of 35 km/h. Company policy may also set speed limits for various reasons. These speed limitations are very important, particularly on the return journey (empty) where maximum speeds are usually achievable and the speed limits will determine the travel time.

Time studies

One simple way to get good data is simply to carry out time studies in the pit. Even if the data are not used directly, time study data are useful to validate computer generated times and for calibration purposes. Radar guns are useful for checking downhill speeds. They are particularly useful on downhill runs where the operator, rather than the engine power, determines the speed.

Performance curves and factors

It is quite possible to use performance charts to determine the speed on a section of a haul profile. The actual average speed achieved on the section, however, will be affected by a number of factors such as:

- initial and end speeds
- length of haul
- power to weight ratio.

Speed factors are used to convert the performance chart speed to estimated average speed on a section. Figures 7.16 to 7.18 present these factors in graphical form.

$$\text{Average section speed} = \text{maximum attainable speed} \times \text{speed factor}$$

$$\text{Travel time (min)} = \text{section length (m)} \times 0.06/\text{average speed (km/h)}$$

Example

What is the travel time for a truck over a 500 m section of road where the maximum attainable speed is 16 km/h?

Average speed = 16 × 0.75
= 12 km/h

Travel time = 500 × 0.06/12
= 2.5 min

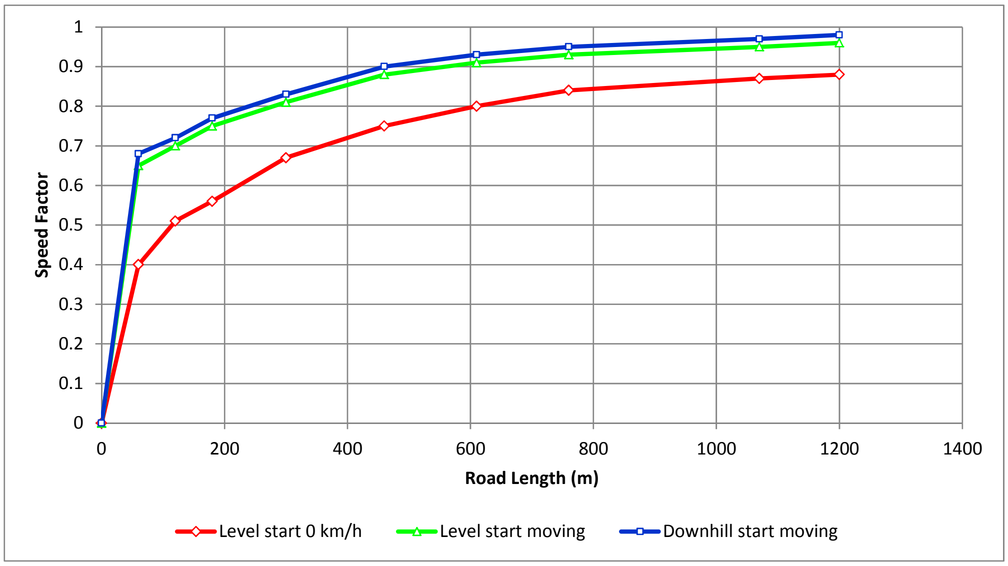

FIG 7.16 - Speed factor charts (less than 180 kg/kW).

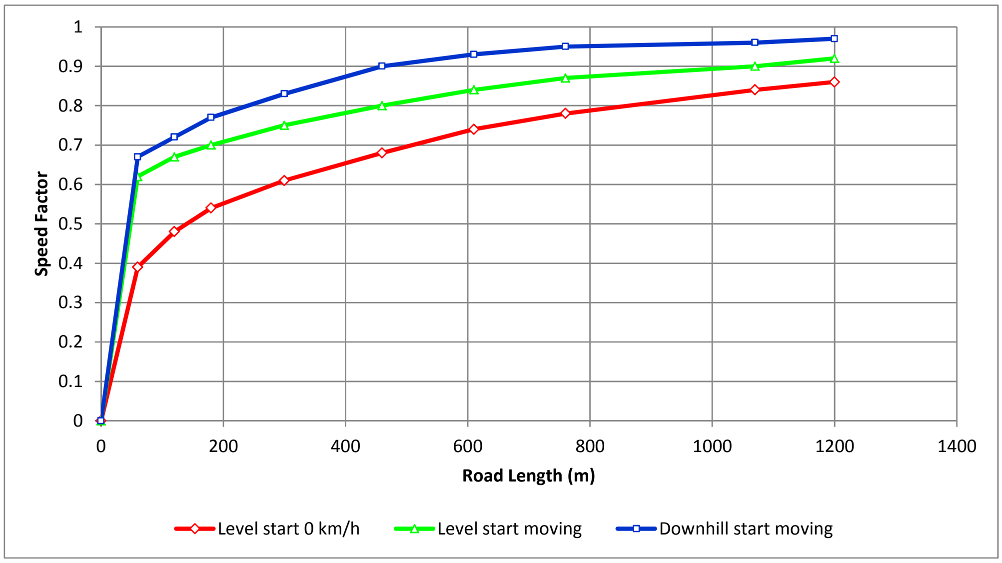

FIG 7.17 - Speed factor charts (180 - 230 kg/kW).

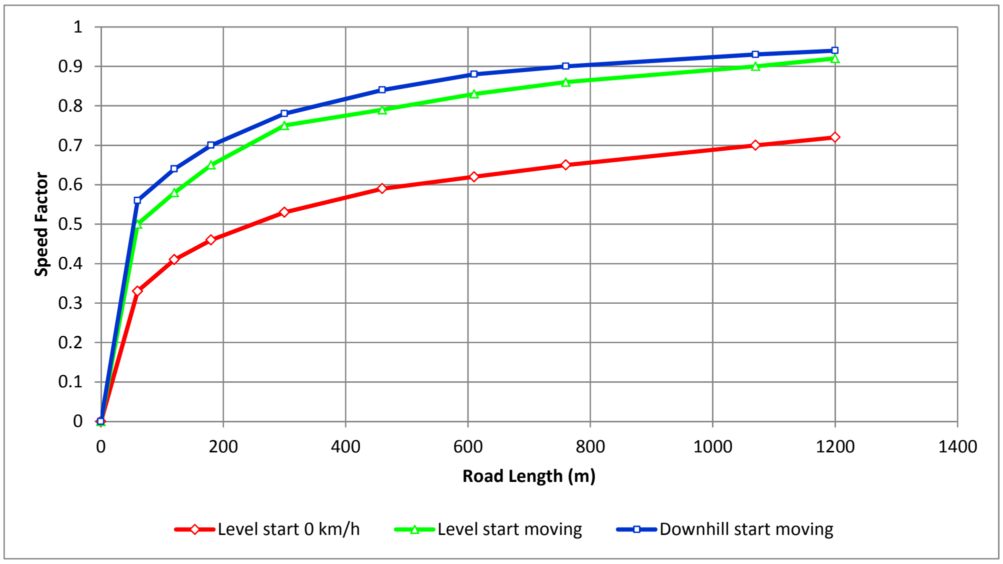

FIG 7.18 - Speed factor charts (greater than 230 kg/kW).

Empirical approach

This system uses previous simulations, or time studies, to provide a rate in terms of seconds per metre of haul.

Example

Create a rate for an uphill loaded haul with average speed of 14 km/h loaded and 35 km/h return. Use this rate to calculate the travel time over a 500 m section.

Speed loaded	= 14 km/h
	= 3.89 m/s
Haul rate	= 0.257 s/m
Speed empty	= 35 km/h
	= 9.72 m/s
Empty rate	= 0.103 s/m
Total rate	= 0.103 + 0.257
	= 0.360 s/m
Travel time	= 0.360 × 500
	= 180 s
	= 3 min

The advantage of the empirical system is that a number of rates may be calculated for typical haul sections. These can be used to very quickly calculate travel time for a profile without using a computer. Put into a spreadsheet, the rates can be used to analyse different haul routes for productivity.

Queuing approach

In a truck-shovel system, trucks do not arrive at the shovel to be serviced in a predictable manner. It does not take exactly the same time for the shovel to serve each truck. The interaction between the randomness of the inter-arrival times of the trucks and the shovel service time causes either a waiting line to form at the shovel, or leaves it idle. This situation suggests that a queuing approach may help to assess how much time is lost in waiting both by the shovel and the trucks.

Scraper types

Rubber-tyred scrapers consist of one or more prime movers and a central bowl structure, which carries the payload. The scraper is loaded by lowering the bowl until a cutting edge at the front of the bowl engages the ground and then as the scraper moves forward, the front apron of the bowl opens to allow material to enter the bowl. The power required to force the material into the bowl is considerable and comes from the scraper itself, with or without the assistance of another machine, which is usually a dozer or unloaded scraper.

The two common types of scraper are elevating and open bowl.

Elevating type

The elevating type of scraper incorporates a flight elevator at the front of the bowl to assist loading the material into the bowl. They have application in unconsolidated material such as topsoil and alluvium. The advantages of the elevating scraper are that they:

- are self-contained load-and-haul units
- may work as individual units.

Open bowl

The open bowl type of rubber-tyred scraper generally requires assistance in loading from another machine, either a dozer or another scraper, depending on the type of machine. They can either be single-engine or twin-powered. The single-engine machines perform well over relatively flat terrain with medium-haul distances. The twin-powered machines have better traction and can handle steeper grades. However, they are more expensive to purchase and operate and their increased productivity has to be large enough to offset the increased costs of owning and operating. Typical performance curves for an open bowl scraper are given in Figures 7.19 and 7.20.

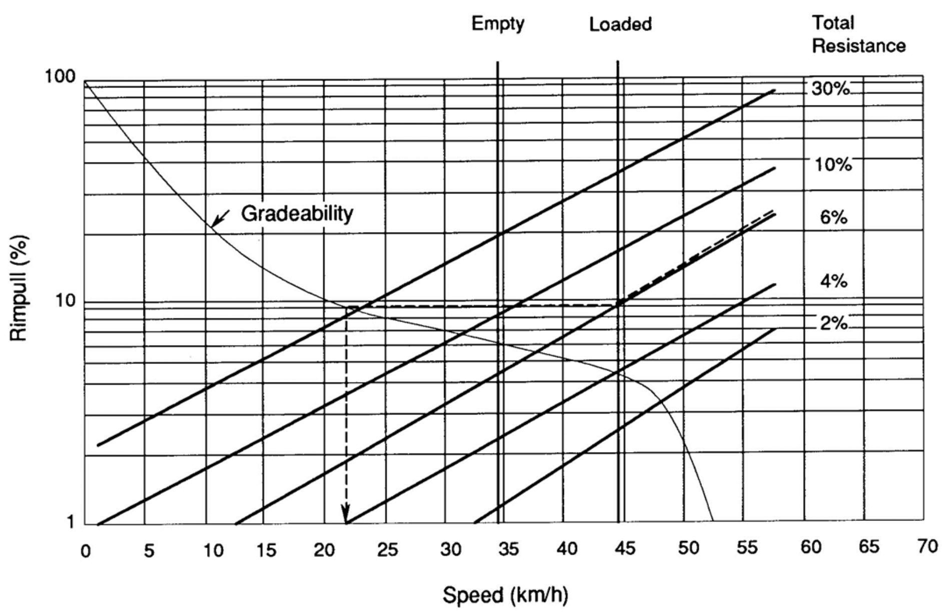

FIG 7.19 - Open bowl scraper performance on uphill grades.

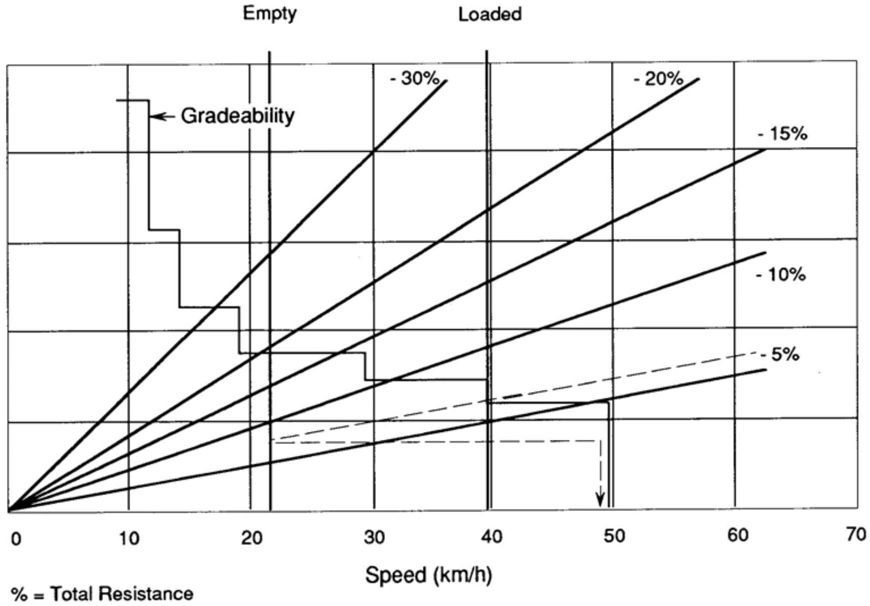

FIG 7.20 - Open bowl scraper performance on downhill grades.

Scraper productivity

In order to determine the number of scrapers required on a mine site, the cycle time and quantity of material carried per cycle must first be determined. The procedure adopted in making these determinations is illustrated in the following example:

Site conditions

Haul road	Maintained dirt
RR	3%
Material	Clay overburden
Swell	1.2
Density	2.0 t/m^3
Haul	1.5 km at 3% grade
Production required	4000 BCM per eight-hour shift
Machine specifications	
Scraper	23 m^3, single engine, open bowl
Dozer power	315 kW
Calculations	
Scraper capacity	= 23/1.2 = 19 m^3 × 2.0 = 38 t
Rated load	= 34 t
Maximum load	= 34/2 = 17 m^3
Total resistance	= 3.0 + 3.0 = 6.0%
Loaded haul speed (from Figure 7.19)	= 21.5 km/h × 0.85 = 18.2 km/h
Loaded haul time	= 1.5 × 60/18.2 = 4.9 min
Empty haul speed (from Figure 7.20)	= 50 km/h × 0.85 = 42.5 km/h
Empty haul time	= 1.5 × 60/42.5 = 2.2 min
Total cycle time	
Load	= 0.7 min
Haul full	= 4.9 min
Haul empty	= 2.2 min
Dump	= 0.7 min
Total	= 8.5 min
Productivity	
Cycles per hour	= 60/8.5 = 7.1
Productivity (47 min hour)	= 7.1/(47/60) = 5.5 cycles/h
Production per hour	= 5.5 × 17 = 93 m^3/h
Scraper production	= 93 × 8 = 744 m^3/shift
Scrapers required	= 4000/744 = 5.4
	= say 6 units

Dozers required – assume dozers push load then back-track to push next scraper

Dozer cycle time	= 0.7 + 0.7 = 1.4 min
Number of scrapers per dozer	= 8.5/1.4 = 6.1
Dozers required	= 1 unit

Conveyors

Conveyors are typically used in surface mine haulage for overburden disposal, using conventional or stacking conveyors and for transporting ore from the face to an in-pit crusher or external beneficiation plant. They are particularly suited when a bucket wheel excavator or surface miner continuously excavates the mining face.

MISCELLANEOUS EQUIPMENT

Numerous items of minor equipment are required around a surface mine to support major equipment. The minor items include haul road maintenance machines, dewatering plants, lighting plants and light service vehicles.

Haul road maintenance

Haul roads are usually constructed with unsealed gravel pavement and, in order to reduce RR, they must be constantly graded and repaired. Motor graders are the main item of plant required. The frequency of grading is dependent on the standard of road construction and traffic density. The grading frequency can be as often as once per hour or up to once per shift. Grader specifications are given in Table 7.24.

TABLE 7.24
Grader specifications.

Blade width (m)	3.7	4.3	4.9	7.3
Power (flywheel kW)	100 - 120	190	220	400
Machine weight (t)	13 - 15	21	27	62
Typical operating speed (km/h)	5.0	5.0	6.5	6.5
Typical pass width (m)	2.0	2.4	2.8	4.5
Area covered per 47 min/h[a] (m^2)	7800	10 300	14 250	22 300
Average life (h)	30 000	30 000	30 000	30 000

a. This is the efficiency, as discussed and shown in Table 7.15.

The number of graders required can be calculated by dividing the surface area of roads to be maintained by the hourly production per machine.

The other essential item for haul road maintenance is the water truck. The surface of an unsealed haul road should be kept moist (but not wet) to reduce dust, and hence improve driver visibility. Watering roads also keeps the surface compact and prevents the development of surface irregularities. The amount of water required depends on:

- density of traffic
- evaporation
- humidity
- natural ground moisture
- rainfall
- surface material type.

Of the above, the most important factor is the evaporation rate. Values of evaporation per day for various climatic regions and months are given in Table 7.25. Under conditions of little rainfall, it will be necessary to apply water to the road surface at least equal to the rate of evaporation to maintain constant moisture content in the pavement material.

TABLE 7.25
Average daily evaporation rates for four Australian sites.

Month	Evaporation (mm)			
	Meekatharra, WA	Mt Isa, Qld	Cowra, NSW	Queenstown, Tas
January	16.7	10.3	7.4	3.5
April	9.0	8.5	2.8	1.3
July	4.1	5.5	1.0	0.7
October	11.4	10.6	3.8	2.0

The number of water trucks required can be calculated by analysis of the haul cycle time. The time to fill is dependent on filling-point arrangement. If an overhead tank is installed, filling can be very rapid. If filling is directly from a suction pump at a lagoon, the filling time will depend on the pump capacity. In rare situations, no filling facilities are provided and the spray pump on the truck is used for filling, which can be quite slow. Typical times are 2.5 minutes for overhead facilities, five minutes for pup-stand-type arrangements and 15 minutes for self-loading. Discharge time depends on the truck-mounted pump size.

Dewatering plants

Surface mining operations accumulate water from groundwater ingress, surface run-off and rainfall. The amount of water to be pumped from an excavation must be determined by a hydrological study for each site. The quantity can be reduced by constructing surface water diversion drains and channels and levee banks around the excavation. A sump must be maintained at the lowest point in the excavation for the collection of water and to facilitate pumping. Various pump types are available, including submersible, suction centrifugal and diaphragm. Motors can be diesel or electric.

Lighting plants

Mining operations carried out in the field at night require good illumination for safety and efficient operation. Relatively permanent installations, such as dump hoppers, can be lit by mains-powered lights. However, other areas, such as loading points for mobile machines, will change position as mining progresses. In these cases, portable lighting plants are often used and typically consist of a diesel generator with lamps, located on a mast about 10 m high.

Many factors are involved in determining the number and location of lights. The most important factor is the type of work undertaken, and hence the intensity of light required and the area to be illuminated. On an area of relatively low-level illumination, such as a haul road, one 1000 W tungsten halogen lamp should illuminate about 0.25 ha. The same wattage high-pressure sodium lamps would cover about 0.75 ha. However, areas of high activity could require more than ten times this illumination level.

Light vehicles

Mobile equipment, such as loaders and dozers, are normally fuelled and serviced in the field, while trucks often travel to a central fuelling point at the end of a shift. When the machines are worked on single shift, fuelling and servicing is done at the end of the shift. On multiple shifts, servicing will be undertaken during the productive shift. Meal breaks and shift changeovers are used, as far as possible, to minimise downtime.

Servicing involves checking and topping up hydraulic and lubricating oils, greasing, cleaning and changing filters, adding air to tyres and carrying out preventative maintenance. Two people operating from a well-equipped service truck can complete the average mobile equipment service in about ten minutes. Refuelling in the field is undertaken from fuel tankers, equipped with pumps that deliver about 5 L/s of fuel. Thus, with extra tasks such as connecting and disconnecting hoses and caps, a machine requiring 500 L of fuel can normally be refuelled in about five minutes. Tyres often have to be replaced in the field and, unless a custom tyre-changing facility is close at hand, this can best be accomplished with a tyre servicing truck. A well-equipped tyre truck comprises a tray body large enough to hold the largest tyre, hydraulic crane and a compressor.

WORKED EXAMPLES

This section provides a worked example of how mining costs could be estimated for a small contract gold mining operation. It is planned to run the operation on a contract mining basis, with a workforce roster of two weeks on followed by one week off. All personnel would operate on a fly-in, fly-out basis.

The mining schedule developed during a feasibility study is provided in Table 7.26.

The anticipated annual production rate for this study is a mill ore feed rate of 2 Mt/a.

The load-and-haul equipment for this study is:

Loading (ore + waste): 15 m^3 excavator

Haulage (ore + waste): 90 t rear dump (non-automated)

Mining productivity

The following sections detail the parameters, costs and equipment selection methodology used as inputs for the mining cost models for each of the cases assessed.

Ore and waste mining

The ore and waste will be mined using a conventional truck and shovel fleet. This study assumes that the fleet will be predominantly Caterpillar equipment. Mining equipment will be selected to suit the size and selectivity of the selective mining unit block size.

Operating hours

The equipment operating hours determine the production and cost of a mining unit. The mining operation will run on a three-panel, 12-hour shift basis.

The mine is planned to be operated 365/a, 24 h/d. An allowance has also been made for bad weather, other stoppages and unproductive time. Table 7.27 gives the estimated annual operating hours for the mining operation.

Table 7.28 gives the operating hours potentially available for the main production equipment.

TABLE 7.27
Possible production calculation.

Line no	Calculation	Material	Units	Resulting time
1		Maximum days/annum	d/a	365
2		Holidays	d/a	0
3		Weekends	d/a	0
4		Possible days/annum	d/a	365
5		Weather delays	d/a	13
6	4 - 5	Operating days/annum	d/a	352
7		Scheduled hours/shift	h/shift	12
8		Shifts/day	d	2
9	6 × 7 × 8	Scheduled hours/annum	h/a	8448
10		Available hours/shift	h/shift	11
11	6 × 8 × 10	Available hours/annum	h/a	7744

TABLE 7.26
Mining schedule.

Material quantities	Units	2015	2016	2017	2018	2019	2020	Total
Total movement	Mt	11.50	16.80	14.60	15.00	12.10	9.45	79.45
Strip ratio	Waste:ore	22.0	7.4	6.3	6.5	5.1	6.0	7.1
Waste	Mt	11.00	14.80	12.60	13.00	10.10	8.10	69.60
Ore	Mt	0.50	2.00	2.00	2.00	2.00	1.35	9.85
Au	g/t	1.90	1.93	1.99	2.01	1.94	1.96	1.96
Au recovery	%	85	85	85	85	85	85	85
Au *in situ*	oz	30 543	124 102	127 960	129 246	124 745	85 071	621 656
Au produced	oz	25 962	105 487	108 766	109 859	106 033	72 310	528 417

TABLE 7.28
Operating hour parameters.

Unit	MA (%)	UA (%)	Potential operating hours	Calculation
Shovels/ excavators	85	80	5745	8 448 × 0.85 × 0.80
Front-end loader	80	75	5069	8 448 × 0.80 × 0.75
Trucks	85	70	5027	8 448 × 0.85 × 0.70
Dozers track	75	75	4752	8 448 × 0.75 × 0.75
Dozers (RT)	75	75	4752	8 448 × 0.75 × 0.75
Graders	75	75	4752	8 448 × 0.75 × 0.75
Drills	80	70	4731	8 448 × 0.80 × 0.70

Notes:

MA = mechanical availability – proportion of time that a unit is mechanically available.

RT = rubber-tyred.

UA = use of availability – proportion of mechanically available time that a unit is actually operated.

Equipment productivity

Examples of productivities of various equipment items are presented in this section.

Excavator productivity

Excavator productivities are estimated, as shown in Table 7.29.

Haul unit productivity

Haul truck rated payloads are 91 t for both ore and waste. Cycle times for both ore and waste hauls were calculated empirically using the assumptions provided in Table 7.30. The estimated haul truck requirements are provided in Table 7.31.

Drilling productivity

Material will be drilled on 5 m benches. Drilling and blasting will be carried out in advance of ore and waste removal. Material will be drilled with top-hammer blasthole rigs.

Drilling production rates depend on penetration rates and the blasting design parameters. Typical ore and

TABLE 7.29
Excavator productivity model for Cat 6030 excavator.

Line no	Calculation	Type	Units	Ore	Waste
1		Dipper capacity	m^3	15.0	15.0
2		Fill factor	number	0.95	0.95
3		Swell factor	number	1.3	1.3
4		Material density	t/m^3	2.8	2.8
5	1 × 2/3 × 4	Dipper load	t	30.7	30.7
6		Truck payload	t	90.7	90.7
7	6/5	Passes to fill	number	2.96	2.96
8		Use passes	number	3.0	3.0
9		Actual payload	t	90.7	90.7
10		Minutes/pass	min	0.5	0.5
11		Load time single side	min	1.0	1.0
12		Spot time + first bucket	min	1.0	1.0
13	11 + 12	Loading + spot time + first bucket	min	2.0	2.0
14	50 min/13	Trucks/hour (assuming 50 min/h)	number	25.1	25.1
15		Propel factor	%	100	100
16		Presentation factor	%	80	80
17	9 × 14 × 15 × 16	Tonnes/operating hour	t/op.h	2165	2165
18		Scheduled hours/annum	h	8448	8448
19		Mechanical availability	%	90	90
20		Use of availability	%	80	80
21	18 × 19 × 20	Operating hours	op.h/y	5745	5745
22	17 × 21	Capacity	kt/a	12 473	12 473

TABLE 7.30
Haul truck segment speeds.

Haul segment	Speed full (km/h)	Speed empty (km/h)	Haul segment	Speed full (km/h)	Speed empty (km/h)
First 50 m	20	25	Dump ramp uphill (downhill return)	12	45
Pit floor	25	30	Dump ramp downhill (uphill return)	26	26
Hair pin	20	20	Dump flat	35	45
Downhill ramp (uphill return)	26	26	Dump last 50 m	25	25
Uphill ramp (downhill return)	10	30	Ore ramp downhill (uphill return)	26	26
Ex-pit flat	35	45	Ore ramp uphill (downhill return)	10	30

TABLE 7.31
Estimated haul truck requirements.

Line no	Calculation			2015	2016	2017	2018	2019	2020
		Material movement	Units						
1		Ore	Mt	0.50	2.00	2.00	2.00	2.00	1.35
2		Waste	Mt	11.00	14.80	12.60	13.00	10.10	8.10
		Travel times							
3		Ore	min	15.20	16.10	17.50	18.90	21.30	25.40
4		Waste	min	12.50	16.50	19.10	22.00	24.70	28.30
5		Ore trucks	t	90.7	90.7	90.7	90.7	90.7	90.7
6		Turn and dump	min	1.0	1.0	1.0	1.0	1.0	1.0
7		Spot and manoeuvre	min	0.95	0.95	0.95	0.95	0.95	0.95
8		Load truck	min	1.04	1.04	1.04	1.04	1.04	1.04
9		Waste trucks	t	90.7	90.7	90.7	90.7	90.7	90.7
10		Turn and dump	min	1.00	1.00	1.00	1.00	1.00	1.00
11		Spot and manoeuvre	min	0.95	0.95	0.95	0.95	0.95	0.95
12		Load truck	min	1.04	1.04	1.04	1.04	1.04	1.04
13	3 + 6 + 7 + 8	Ore cycle time	min	18.19	19.09	20.49	21.89	24.29	28.39
14	4 + 10 + 11+ 12	Waste cycle time	min	15.49	19.49	22.09	24.99	27.69	31.29
		Efficiency (50 min/h)							
15		Queue factor	%	95	95	95	95	95	95
16	50 min/13 × 5 × 16	Ore t/h	t/h	237	226	210	197	177	152
17	50 min/14 × 9 × 16	Waste t/h	t/h	278	221	195	172	156	138
18		Training factor	%	80	100	100	100	100	100
19	1/17/19	Operating hours – ore trucks		2639	8862	9512	10 162	11 276	8896
20	2/18/19	Operating hours – waste trucks		49 437	66 953	64 605	75 406	64 915	58 829
21	20/5027 h[a]	Ore trucks	number	0.5	1.8	1.9	2.0	2.2	1.8
22	21/5027 h[a]	Waste trucks	number	9.8	13.3	12.9	15.0	12.9	11.7
23	21 + 22	Total number of trucks	number	10.4	15.1	14.7	17.0	15.2	13.5

a. See Table 7.27.

waste hole parameters and drilling productivities are summarised in Table 7.32.

TABLE 7.32
Drilling parameters.

Parameter	Units	Ore	Waste
Bench height	m	5.0	5.0
Hole depth	m	5.8	5.8
Hole diameter	mm	127	127
Average penetration rate	m/h	31	31

Blasting productivity

It has been assumed that emulsion explosive will be produced and delivered to the hole by a specialist third-party subcontractor. The blasting parameters are given in Table 7.33.

TABLE 7.33
Blasting parameters.

Blasting parameters	Units	Waste	Ore
Hole diameter	m	0.127	0.127
Spacing	m	5.18	5.18
Burden	m	4.50	4.50
Depth	m	5.0	5.0
Rock density	t/m^3	2.80	2.80
Hole depth	m	5.8	5.8
Subgrade	m	0.8	0.8
Stemming length	m	2.4	2.4
Explosive density	t/m^3	1.20	1.20
kg explosive per metre	kg	15.2	15.2
Explosive per hole	kg	51.7	51.7
Rock/hole	t	326	326
Rock/hole	BCM	116	116
Powder factor	kg/t	0.16	0.16
Powder factor	kg/BCM	0.44	0.44

Note: BCM = bank cubic metres.

Ancillary equipment

The ancillary fleet provides services to ensure continued production and perform any general pit and dump work not directly involved in the mining operations. Services include:

- dust suppression
- in-pit refuelling and lubrication
- lighting of work areas
- mobile maintenance
- personnel transport
- road construction and maintenance
- waste dump and in-pit bench maintenance.

Assumptions regarding ancillary equipment fleet requirements are:

- one tracked dozer is assigned to each primary loading tool
- two tracked dozers are assigned to the dumps and general works
- one rubber-tyred dozer provides mobile support
- one grader is assigned to cover the pit, dump and infrastructure roads
- one 90 t water truck covers the pit dump and infrastructure roads.

Grade control

Grade control samples will be collected during blasthole drilling. An allowance has been made for 20 per cent of the waste to be sampled and 100 per cent of the ore at a rate of one sample per metre drilled.

Personnel

Crewing considerations include the roster and the number of tradespeople and operators.

Crew roster

The roster is a continuous three-panel crew roster. The roster parameters provided in Table 7.34 have been based on these assumptions. The roster is 14 days on followed by seven days off.

TABLE 7.34
Personnel roster parameters.

Item	Units	Resulting time
Days on	d/roster	14
Days off	d/roster	7
Maximum days/annum	d/a	365
Weather delays	d/a	13
Annual leave	d/a	20
Sick leave	d/a	10
Funeral leave	d/a	5
Absenteeism	d/a	10
Rostered off	d/a	122
Training	d/a	12
Total worked	d/a	186
Personnel paid/possible shift	person/shift	1.31 (use 1.30)

Operator and trades requirements

Operator numbers are based on effective hours of equipment with labour factor added to cover availability, vacation, sickness and compassionate leave. Absenteeism is unpaid, so is not considered in the cost model.

The personnel numbers assume an efficient workforce operating with high levels of multi-skilling and flexibility.

The numbers of tradespeople have been estimated using a ratio of maintenance hours/unit operating hours, as given in Table 7.35.

TABLE 7.35
Trades to operator ratio.

Unit	Trades hours/operating hour
Excavators	0.66
Drill	0.55
Front-end loader	0.45
Trucks	0.44
Water truck	0.44
Dozers tracked	0.40
Rubber-tyred dozer	0.40
Grader	0.30
Other	0.30

Contractor operating costs

A contractor cost model has been developed using a margin of 15 per cent. This scenario assumes that the contractor will carry out all mining, maintenance and supervision.

The operating costs have been estimated to a prefeasibility level of accuracy.

Table 7.36 gives the major cost assumptions used in the study for exchange rates and fuel price.

TABLE 7.36
Fuel price and exchange rates.

Fuel price	A$/L	0.95
Exchange rate	-	A$1.00 = US$1.00

Equipment capital costs

Individual major mining equipment capital cost, estimated equipment life and ownership cost for the selected fleet are provided in Table 7.37.

Operating costs

The cost of operating the loading and hauling fleets makes up the majority of mining operating costs. For this study, the ownership cost was estimated as a lease payment. Equipment capital costs can also be handled by estimating the number of units required, then determining an initial purchase and a replacement schedule based on the expected life of the equipment.

The basic formula and sample calculation for the ownership cost method is given below:

Lease cost per annum = purchase price × lease factor/yearly hours

TABLE 7.37
Capital cost and life of major mobile equipment.

Machine	Capital cost (A$M)	Life (operating hours)
Excavator – Cat 6030	5.9	50 000
Trucks – Cat 777	2.4	48 000
Fuel truck	2.2	80 000
Front-end loader – Cat 988	1.7	48 000
Wheel dozer – Cat 854	1.4	48 000
Dozer – Cat D9	1.2	40 000
Drills	1.0	60 000
Grader – Cat 16M	0.9	36 000
Sump pumps	0.12	20 000
Light vehicles	0.07	7500
Lighting plants	0.06	25 000

$$\text{Lease factor} = i\,(1 + i)^N/((1 + i)^N - 1)$$

where:

i interest rate

N life or payment period in years

Example

Purchase price	= $1 800 000
Life	= 8 years
Usage per annum	= 5000 hours
Interest rate	= 15%
Salvage value	= 10%
Lease factor	= $0.15\,(1.15)^8/((1.15)^8 - 1)$
	= 0.2229
Discounted salvage value	= $(1\,800\,000 \times 0.1)/(1.15)^8$
	= $58 842
Yearly lease cost	= (1 800 000 - 58 842) × 0.2229
	= $388 104
Average lease cost/hour	= $388 104/5000
	= $77.62/h

Table 7.38 gives the lease assumptions used in this study.

TABLE 7.38
Lease cost assumptions.

Item	Value (%)
Interest rate	8
Salvage value	10

Equipment operating costs on a $/h basis for the main and ancillary fleets are summarised in Table 7.39 and Table 7.40. Total costs by year for the main and ancillary fleets are summarised in Table 7.41 and Table 7.42. These costs are exclusive of labour costs.

TABLE 7.39
Summary of main equipment operating costs – excluding labour ($/h).

Machine	Model	Capital cost ($M)	Operating hours/a (h/a)	Life (op.h)	Ownership costs ($/op.h)	Fuel + lube ($/op.h)	Parts, workshop ($/op.h)	Overhaul ($/op.h)	Tyres, tracks ($/op.h)	GET, body, bucket, etc ($/op.h)	Opex excl labour ($/op.h)	Capex + Opex excl labour ($/op.h)
Excavator	15 m^3	5.9	5745	50 000	159.68	248.86	65.61	58.92	51.46	100.96	525.82	685.50
Front-end loader	6 m^3	1.7	5069	48 000	49.34	44.84	4.62	3.57	23.27	1.36	77.66	127.01
Truck	90 t	2.4	5027	48 000	69.87	84.08	9.78	6.73	34.52	15.31	150.42	220.29
Drill	152 mm dia	1.0	4731	60 000	26.11	67.26	3.90	2.36	11.92	0.93	86.37	112.48
Grader	4.3 m blade	0.9	4752	36 000	32.38	33.63	4.91	3.30	16.27	9.16	67.27	99.65
Track dozer	300 kW	1.2	4752	40 000	40.15	67.26	6.19	4.42	30.60	15.30	123.77	163.93
Wheel dozer	340 kW	1.4	4752	40 000	46.84	78.47	5.31	3.90	13.26	1.46	102.40	149.24
Water truck	90 t	2.4	5027	48 000	69.87	84.08	9.78	6.73	34.52	15.31	150.42	220.29

Note: GET = ground engineering tools.

TABLE 7.40
Summary of ancillary equipment operating costs – excluding labour ($/h).

Machine	Capital cost ($M)	Operating hours/a (h/a)	Life (years)	Ownership costs ($/a)	Fuel ($/a)	Lube ($/a)	Repairs and maintenance ($/a)	Opex excluding labour ($/a)	Capex + Opex excluding labour ($/a)
Shift change bus	0.150	3000	5.0	35 012	4750	713	6000	11 463	46 474
Light vehicle	0.070	3600	3.0	25 006	2850	428	2800	6078	31 084
Person haul vehicles	0.100	2000	5.0	23 341	4750	713	4000	9463	32 804
Ambulance	0.100	500	10.0	14 213	950	143	4000	5093	19 305
Fire truck	0.250	500	10.0	35 532	1425	214	10 000	11 639	47 170
Fuel/lube truck	0.300	3000	10.0	42 638	3800	570	12 000	16 370	59 008
Maintenance truck	0.125	3000	5.0	29 176	3800	570	5000	9370	38 546
Lighting plant	0.060	3590	10.0	8 528	5700	855	2400	8955	17 483
Sump pump	0.120	3000	10.0	17 055	5700	855	4800	11 355	28 410
Tyre handler/ fork-lift	0.200	3000	10.0	28 425	950	143	8000	9093	37 518
Compactor	0.285	1500	10.0	40 506	19 000	2850	11 400	33 250	73 756
Crane	0.400	3000	20.0	39 867	2850	428	16 000	19 278	59 144

TABLE 7.41
Summary of main fleet costs – excluding labour ($/h).

Machine	Operating cost ($/h)	Year	2015	2016	2017	2018	2019	2020	Total
Excavator – ore + waste	685.50	Hours	7083	7760	6744	6929	5589	4365	38 469
		Cost ($'000)	4855	5320	4623	4750	3831	2992	26 371
Front-end loader	127.01	Hours	5069	5069	5069	5069	5069	5069	30 413
		Cost ($'000)	644	831	831	831	831	831	4798
Trucks – ore + waste	220.29	Hours	52 076	75 815	74 117	85 568	76 191	67 725	431 492
		Cost ($'000)	11 472	16 701	16 327	18 850	16 784	14 919	95 052
Drill	112.48	Hours	5370	8966	9091	9340	7534	5884	46 185
		Cost ($'000)	604	1009	1023	1051	847	662	5195
Grader	99.65	Hours	4752	4752	4752	4752	4752	4752	28 512
		Cost ($'000)	474	474	474	474	474	474	2841
Tracked dozer	163.93	Hours	15 363	15 923	15 083	15 235	14 127	13 115	88 846
		Cost ($'000)	2518	2610	2472	2497	2316	2150	14 564
Wheel dozer	149.24	Hours	4752	4752	4752	4752	4752	4752	28 512
		Cost ($'000)	709	709	709	709	709	709	4255
Water truck	220.29	Hours	3379	3379	3379	3379	3379	3379	20 275
		Cost ($'000)	744	744	744	744	744	744	4466
								Total cost ($'000)	157 543

TABLE 7.42
Summary of ancillary fleet costs – excluding labour ($/h).

Ancillary plant	Annual cost ($/a)	Year	2015	2016	2017	2018	2019	2020	Total
Shift change bus	46 474	No units	4	4	4	4	4	4	
		Cost ($'000)	186	186	186	186	186	186	1115
Light vehicles	31 084	No units	20	20	20	20	20	20	
		Cost ($'000)	622	622	622	622	622	622	3730
Person haul vehicles	32 804	No units	6	6	6	6	6	6	
		Cost ($'000)	197	197	197	197	197	197	1181
Ambulance	19 305	No units	1	1	1	1	1	1	
		Cost ($'000)	19	19	19	19	19	19	116
Fire truck	47 170	No units	1	1	1	1	1	1	
		Cost ($'000)	47	47	47	47	47	47	283
Fuel/lube truck	59 008	No units	1	1	1	1	1	1	
		Cost ($'000)	59	59	59	59	59	59	354
Maintenance truck	38 546	No units	1	1	1	1	1	1	
		Cost ($'000)	39	39	39	39	39	39	231
Lighting plants	17 483	No units	5	5	5	5	5	5	
		Cost ($'000)	87	87	87	87	87	87	524
Sump pumps	28 410	No units	2	2	2	2	2	2	
		Cost ($'000)	57	57	57	57	57	57	341

TABLE 7.42 CONT ...

Ancillary plant	Annual cost ($/a)	Year	2015	2016	2017	2018	2019	2020	Total
Tyre handler/fork-lift	37 518	No units	1	1	1	1	1	1	
		Cost ($'000)	38	38	38	38	38	38	225
Compactor	73 756	No units	1	1	1	1	1	1	
		Cost ($'000)	74	74	74	74	74	74	443
Crane	59 144	No units	1	1	1	1	1	1	
		Cost ($'000)	59	59	59	59	59	59	355
								Total cost ($'000)	8898

Remuneration

The base pay remuneration, inclusive of 28 per cent on-costs, airfares at $750 per return trip and accommodation and messing at $120/d, are summarised according to personnel level in Table 7.43.

Blasting costs

It has been assumed that a specialist third-party subcontractor will produce and deliver emulsion explosive to the hole. The blasting parameters and unit cost estimates are given in Table 7.44.

TABLE 7.43
Remuneration according to personnel level.

Position	Salary + on-costs (A$/a)	2015	2016	2017	2018	2019	2020
Management and technical services							
General manager	455 000	1	1	1	1	1	1
Mine manager	350 000	1	1	1	1	1	1
Maintenance manager	250 000	1	1	1	1	1	1
Technical services manager	350 000	1	1	1	1	1	1
Admin superintendent	350 000	1	1	1	1	1	1
Chief geologist	350 000	1	1	1	1	1	1
Senior mine engineers	350 000	1	1	1	1	1	1
Chief surveyor	275 000	1	1	1	1	1	1
Surveyors	175 000	1	1	1	1	1	1
Mine engineers	175 000	2	2	2	2	2	2
Geologist	156 250	4	4	4	4	4	4
Surveyor assistant	125 000	1	1	1	1	1	1
Field assistant	125 000	4	4	4	4	4	4
Senior safety and training officer	187 500	1	1	1	1	1	1
Safety and training officer	156 250	1	1	1	1	1	1
Human resources officer	145 000	1	1	1	1	1	1
Secretary	115 000	2	2	2	2	2	2
Clerks	105 000	2	2	2	2	2	2
Systems analyst	125 000	1	1	1	1	1	1
Assistants/trainees	95 000	2	2	2	2	2	2
Total		30	30	30	30	30	30

TABLE 7.43 CONT...

Position	Salary + on-costs (A$/a)	2015	2016	2017	2018	2019	2020
Operations							
Superintendent	390 000	1	1	1	1	1	1
Foreman	325 000	3	3	3	3	3	3
Shift supervisors	325 000	3	3	3	3	3	3
Shovel operator	215 000	4	4	4	4	3	2
Truck operator	195 000	26	38	38	43	39	34
Loader operator	215 000	3	3	3	3	3	3
Dozer operator	202 000	10	10	10	10	9	9
Grader operator	202 000	2	2	2	2	2	2
Drill operator	202 000	3	5	5	5	4	3
Water truck operator	202 000	3	3	3	3	3	3
Explo/RC drill operators	202 000	0	0	0	0	0	0
Shot crew	195 000	5	5	5	5	5	5
Sample collectors/drill assistant	202 000	3	3	3	3	3	3
Allowance for sick leave/training, etc	202 000	15	18	18	20	18	16
Helpers/trainees	150 000	0	0	0	0	0	0
Total		81	98	98	105	96	87
Maintenance							
Maintenance superintendent	299 000	1	1	1	1	1	1
Shift supervisor	240 500	3	3	3	3	3	3
Fitter	240 500	8	10	10	10	10	9
Electrician	240 500	3	3	3	3	3	3
Welder	155 000	8	10	10	10	10	9
Serviceman	135 000	8	10	10	10	10	9
Maintenance planner	187 500	1	1	1	1	1	1
Senior mechanical engineer	200 000	1	1	1	1	1	1
Allowance for sick/training, etc	187 500	7	8	8	8	8	8
Helpers/trainees	95 000	0	0	0	0	0	0
Total		40	47	47	47	47	44
Grand total		151	176	176	182	173	161

The total annual blasting cost estimates given in Table 7.45 were generated using the blasting parameters and unit costs (Table 7.44) and the mining schedule (Table 7.26).

Drilling costs

Drilling costs are a function of the operating hours incurred; an estimate of drill operating hours is given in Table 7.46.

Grade control

It has been assumed that grade control samples will be collected during the blasthole drilling process. It has been estimated that 100 per cent of ore and 20 per cent of waste will be sampled at a rate of one sample per blasthole metre. A cost of A$30 has been estimated for collecting and assaying each sample. A grade control cost estimate is provided in Table 7.47.

TABLE 7.44
Blasting parameter and unit cost estimate.

Explosive cost	Units	Ore costs	Waste cost
Emulsion	$/t	1120.00	1120.00
Emulsion	$/hole	57.89	57.89
Fixed cost	$/hole	2.00	2.00
Downhole detonators and nonel	$/hole	13.63	13.63
Boosters	$/hole	10.69	10.69
Surface delay	$/hole	9.41	9.41
Blasting total	$	93.62	93.62
Cost/tonne	$/t	$0.29	$0.29

TABLE 7.45
Estimated blasting costs.

Blasting	Units	2015	2016	2017	2018	2019	2020	Total
Tonnes/annum ore	Mt/a	0.5	2.0	2.0	2.0	2.0	1.4	9.9
Tonnes/annum waste	Mt/a	11.0	14.8	12.6	13.0	10.1	8.1	69.6
Explosives required	t	1823	2663	2315	2378	1918	1498	12 595
Explosives required/day (assuming 352 days operation)	kg/day	5179	7566	6575	6755	5449	4256	35 781
Blasting cost	$'000	3302	4824	4192	4307	3373	2714	22 712

TABLE 7.46
Drill operating hours.

Parameter	Units	2015	2016	2017	2018	2019	2020	Total
Holes drilled ore	number	1534	6134	6134	6134	6134	4141	30 211
Holes drilled waste	number	33 740	45 395	38 647	39 874	30 979	24 845	213 480
Metres drilled (including 5% re-drill, etc)	m	214 815	313 816	272 721	280 193	226 023	176 522	1 484 090
Training factor	%	0.80	1.00	1.00	1.00	1.00	1.00	
Mechanical availability	%	0.80	0.80	0.80	0.80	0.80	0.80	
Use of availability	%	0.70	0.70	0.70	0.70	0.70	0.70	
Drilling hours/annum	h/a	5370	8966	9091	9340	7534	5884	46 185
Number of rigs	number	1.4	1.9	1.9	2.0	1.6	1.2	

TABLE 7.47
Grade control cost estimate.

Grade control	Year	2015	2016	2017	2018	2019	2020	Total
Ore metre drilled	m	8895	30 672	30 672	30 672	30 672	20 704	152 287
Waste metres drilled	m	39 138	45 395	38 647	39 874	38 034	24 845	225 933
Total	m	48 033	76 068	69 320	70 547	68 706	45 549	378 223
Samples	number	16 723	39 752	38 402	38 647	36 868	25 673	196 065
Cost of assaying	$/sample	30	30	30	30	30	30	30
Total cost	$'000	502	1193	1152	1159	1106	770	5882

Other overheads

Other overhead costs are given in Table 7.48.

Miscellaneous costs

Allowances for miscellaneous costs have been included in the estimate; they are given on a yearly basis in Table 7.49.

Unallocated maintenance expenses have been estimated at two per cent of the fleet operating costs and unallocated operating expenses are estimated to be 0.5 per cent of fleet operating costs.

Cost summary

The cost summary for all components of the estimate is given in Table 7.50. This would normally form one input into a cost model for the entire operation.

TABLE 7.48
Other overhead costs.

Ancillary plant	Annual cost (A$)	Ancillary plant	Annual cost (A$)	Ancillary plant	Annual cost (A$)
Service truck	708 313	Maintenance truck	38 546	Ambulance	19 305
Lowbed and prime mover	61 919	Tyre handler/fork-lift	37 518	Fire truck	47 170
Person haul vehicles	32 804	Shift change bus	46 474	Crane	59 144
Compactor	73 756	Lighting plant	17 483	Light vehicles	31 084
Tipper – 6 t	14 752	Pit dewatering pump	28 410	Fuel/lube truck	59 008

TABLE 7.49
Miscellaneous costs.

Cost	Units	2015	2016	2017	2018	2019	2020	Total
Consultants	$'000	250	250	250	250	250	250	1500
Contractors – other	$'000	225	225	225	225	225	225	1350
Technical services	$'000	275	275	275	275	275	275	1650
Head office	$'000	0	0	0	0	0	0	0
Dewatering	$'000	250	250	250	250	250	250	1500
Mobilisation/establishment	$'000	60 000	0	0	0	0	0	60 000
Unallocated maintenance expenses	$'000	317	408	391	428	380	336	2260
Unallocated operating expenses	$'000	79	102	98	107	95	84	565
Total miscellaneous	$'000	61 397	1510	1488	1535	1474	1420	68 824

TABLE 7.50
Cost summary.

Mine operating costs	Units	2015	2016	2017	2018	2019	2020	Total
Labour	$M	30.23	35.03	34.77	36.00	33.83	32.10	202.0
Fleet	$M	23.50	29.88	28.69	31.39	28.02	24.96	166.4
Blasting	$M	3.30	4.82	4.19	4.31	3.47	2.71	22.8
Miscellaneous	$M	61.40	1.51	1.49	1.53	1.47	1.42	68.8
Grade control	$M	0.50	1.19	1.15	1.16	1.11	0.77	5.9
Subtotal	**$M**	**118.94**	**72.43**	**70.29**	**74.39**	**67.91**	**61.97**	**465.9**
Contingencies	$M	0	0	0	0	0	0	0
Profit	$M	17.84	10.87	10.54	11.16	10.19	9.29	69.9
Total	**$M**	**136.78**	**83.30**	**80.84**	**85.55**	**78.09**	**71.26**	**535.8**
Unit cost/t rock	**$/t**	**11.89**	**4.96**	**5.54**	**5.70**	**6.45**	**7.54**	**6.74**
Unit cost/t ore	**$/t**	**273.56**	**41.65**	**40.42**	**42.78**	**39.05**	**52.79**	**54.40**

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