CHAPTER 10.3

Mechanical Extraction, Loading, and Hauling

James D. Humphrey and Joshua D. Wagner

This chapter discusses the characteristics and applications of the following common systems used in modern mining operations to extract, load, and haul waste and ore:

- Dragline systems
- Bucket-wheel excavator systems
- Loader and hauler systems
- Other systems: bottom-dump coal hauler, trolley-assist mining truck, wheel-tractor scraper, and in-pit crushing and conveying system

Whether a mining operation must select a new system or optimize an existing system, it is important to consider far more than just cost or discounted cash flow. Different systems have different operating characteristics and risk exposures. These characteristics and the capability to manage risk rather than cost more often separates the best option from the rest.

An initial evaluation of operational requirements will narrow the choice of systems. However, inevitably the project feasibility comes down to a capital- and ownership-cost evaluation by means of one of several discounted-cash-flow methods. Significantly, two key components of that evaluation, production and operating costs, are likely to be estimates, and the sensitivity of the system to operational variables affects the accuracy of these estimates and hence the project feasibility.

In the sections that follow, the following operating variables are discussed as characteristics of each system:

- Mobility: the capability of the system to relocate
- Flexibility: the capability of the system to change direction or work irregular patterns
- Operational range: the variety of application methods in which the system can be used to address requirements for operational change
- Sensitivity to geologic variance: the impact of unexpected changes in material characteristics or unexpected structural or stratigraphic variations

Each of the systems discussed is a well-established system for good reason, and any can be the best choice for a particular application.

DRAGLINE SYSTEMS

Draglines (Figures 10.3-1 and 10.3-2) are self-contained systems that load and transport material to a dump point. They are highly productive, comparatively low in operating costs and labor requirements, and extremely robust, and subsequently have very long lives, commonly 30 to 40 years.

For purposes of comparison, the dragline system is a high-capital-cost, low-operating-cost system that is moderately flexible and can operate through a moderate range of applications with low sensitivity to geologic variance.

Because of their high productivity and capability of direct disposal of material, draglines are favored for area mining in areas of flat-lying tabular geology with high production requirements. The most common application for large draglines is overburden removal in coal mining. However, because they excavate below their working level and exert very low ground-bearing pressure, they are uniquely suitable for digging very wet materials, occasionally even below water.

A large dragline can operate through a range from about 50 m (170 ft) above to 65 m (210 ft) below its working level. This means that, with advanced techniques, the dragline can handle overburden depths of about 80 m (260 ft).

Although the largest dragline ever built had a 170-m^3 (220-yd^3) bucket, for the last few decades the largest draglines built have had 125-m^3 (160-yd^3) buckets. Draglines of this size are capable of moving 30–35 million BCM (bank cubic meters) (40–45 million BCY [bank cubic yards]) per year.

Dragline applications and operations are generally determined by two major factors: the placement of spoil material in the space available and the three basic bucket controls (hoist, drag, and swing). The patterns of tub positions, dig location, digging sequence, and dump locations are best optimized by a pit design that considers spoil placement and the characteristics and interrelationships of the bucket controls. To a lesser extent, the digging peculiarities of the bucket and the bench space required for operations are also considerations for pit design.

In most basic dragline operations, the dragline removes overburden material to uncover ore that is the most recent in

James D. Humphrey, Market Professional–Mining, Caterpillar, Inc., Global Mining Division, Decatur, Illinois, USA
Joshua D. Wagner, Marketing Product Consultant, Caterpillar, Inc., Global Mining Division, Peoria, Illinois, USA

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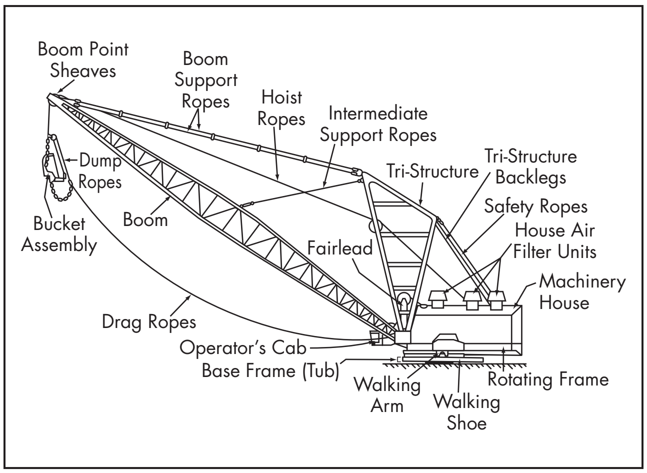
Figure 10.3-1 Typical dragline



Source: Humphrey 1990.

Figure 10.3-2 Dragline design

a series of parallel adjacent pits. Figure 10.3-3 shows a typical dragline pit. Overburden material from the current pit is placed in the previous adjacent pit, from which product has been removed by auxiliary equipment. Pits are narrow and relatively long:

- Pit widths are most commonly 25–60 m (80–200 ft). Widths for rehandle operations tend be on the large side, to reduce the percentage of rehandle. Width is influenced by the maneuverability of the product-removal equipment, depth of the overburden, blasting method, material characteristics, dragline advance rate, and dragline dump radius.
- Pit lengths vary greatly because of the influence of geology, topography, and artificial obstacles. They are most commonly 1,000–2,000 m (3,000–6,000 ft), although some operations have used pits as short as 300 m (1,000 ft) or as long as 3,000 m (10,000 ft). In shorter pits, the sequencing of product removal and blasting becomes complicated and frequent ramp construction is required. In longer pits, power distribution systems become expensive and complex, and dragline propel distances can be excessive.

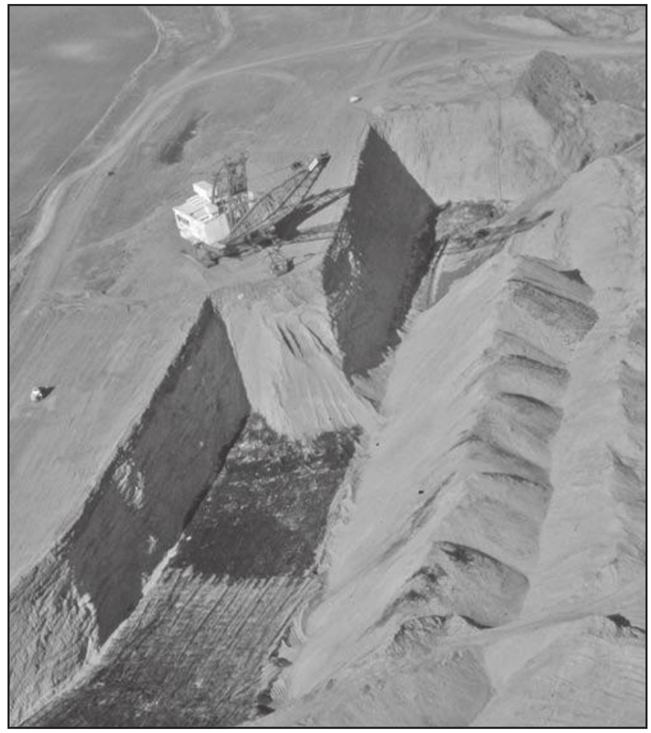


Figure 10.3-3 Typical dragline pit

The pad on which a dragline sits while it works must be clear of hard spots and protruding rocks, and must be relatively level, graded to a slope of ≤2% to provide drainage yet avoid damage due to swing-motor overheating and structural stresses. Modern draglines can propel up and down a ≤10% grade or across a ≤5% grade. When they transition between grades, it is important that they do so gradually, always distributing the load evenly across the tub (the dragline's circular base) and shoes. As a general rule, the rate of grade change should be ≤3% per tub diameter. For example, for a tub 20 m in diameter, the rate of grade change should be ≤3% per 20 m, so a change from 0% to 9% should take at least $9/3 \times 20 = 60$ m. Additionally, when the possibility exists of bridging the shoes, the pad material should be sufficiently compacted to prevent supporting the shoes by the endpoints only.

Draglines are designed to work in soft-underfoot conditions, and as such are designed with tub ground-bearing pressures on the order of 1.2–1.4 kg/cm^2 (17–20 psi). During propel, about 80% of the machine weight is transferred to the shoes, and the remaining weight is carried by the tub edge. This ratio can be changed by carrying the bucket or setting it on the ground. Reducing the tub-edge load by setting the bucket on the ground reduces the probability of pulling a roll under the tub during propel in soft-underfoot conditions.

Rather than remove material from a continuously advancing face as a shovel does, a dragline removes material from a specified length of the pit, called a *set* or *block*. The dragline swings approximately 90° and casts into a pile in the previous pit. Set lengths for larger machines are about 30 m (100 ft), or about 16 steps for the dragline. To remove the overburden from a simple set, a dragline may use from two to four tub positions before retreating to start a new set.

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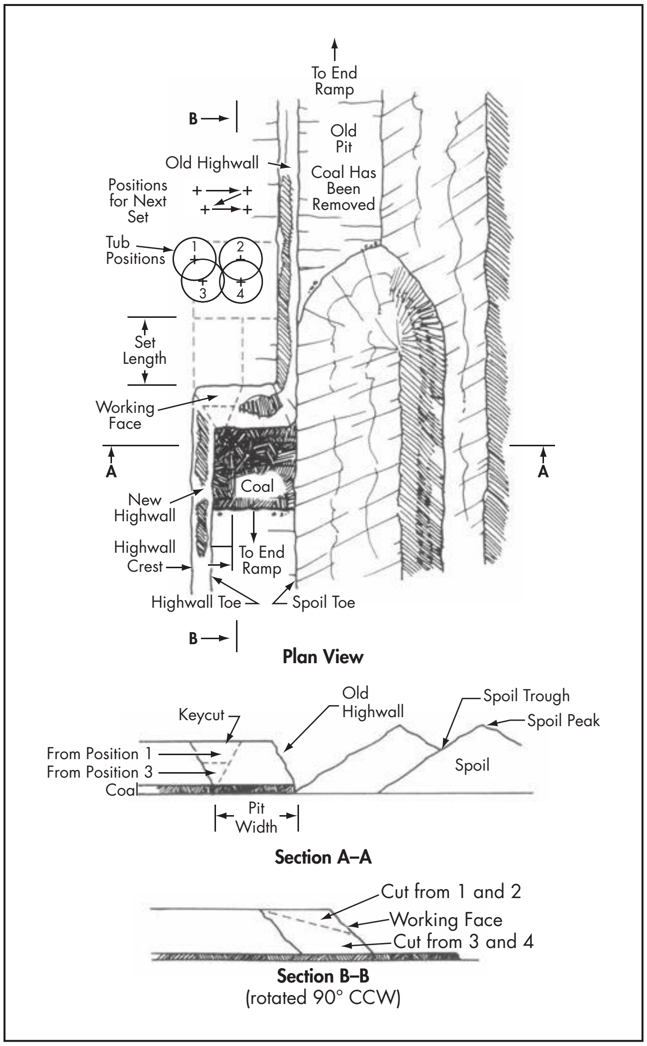

Source: Humphrey 1990.
Figure 10.3-4 Digging positions in a dragline pit

Commonly a dragline follows a pattern of digging positions to excavate a set (Figure 10.3-4). The first two rear positions are set back far enough to ensure that no material is too close to the fairlead to be reached. In shallow pits, these first two positions may suffice to reach the desired depth. In deeper pits, digging may soon reach the point where the drag ropes scour through the crest of the digging face, in which case the dragline must move forward to clear the drag ropes. Thus, from the rear positions, the upper part or *lift* of the set is removed. In the last two positions, the dragline has moved forward to the edge of the digging face to reach down for the lower lift of the set.

The lateral positions in a pit are also significant. From the positions along the highwall (1 and 3 at the bottom of Figure 10.3-4), the dragline can fix the alignment and slope of the new highwall with the *key cut*. This trench-like cut is confined to a bottom width of only a single bucket as it works down. Such a cut allows maximum lateral control of the bucket with minimal lateral strain on the boom. In addition, if the entire dig path is not directly radial to the dragline, production and mechanical availability can suffer. At the bottom of the lift in position 1, the dragline generally moves laterally to position 2. This *plug* position allows the dragline to spoil at maximum range. This move is made before completion of the key to reduce delay caused by hoisting clear of the key before beginning the swing. As excavation progresses, the plug is removed in lifts comprised of a series of cuts to an equal depth (about one-half the bucket height) made in a sweeping pattern. The sweeps normally progress from the spoil side to the key so as to minimize any hoisting required before swinging the bucket to the spoil. Then the dragline moves forward into positions 3 and 4 to excavate the lower lift in much the same fashion.

In Figure 10.3-3, the dragline has stepped back from the face for maintenance but has completed the first three positions of the set. The top of the set has been removed from positions 1 and 2 and the key cut has been finished from position 3. The dragline is now ready to move over closer to the spoil and finish the plug from position 4.

This description, although typical, should be considered general. Set lengths and digging positions will vary depending on operating conditions and machine capabilities.

Dragline Operating Methods

Draglines can operate by means of several operating methods, described in the following paragraphs.

Simple Side Casting

The standard dragline method is simple side casting, used when the dragline has the required reach to move the overburden to its final place. With typical angles of repose and pit widths, the maximum overburden that can be handled by this method is a little less than half the effective radius, discussed later in the "Dragline Selection" section of this chapter. Variations on simple side casting are common.

Advance Benching

Advance benching (Figure 10.3-5A) is useful in areas of uneven terrain or in overburdens where a top layer of unconsolidated material overlays competent rock. The set is split into an upper and lower bench. The lower bench is removed conventionally; the upper bench is removed by *chop cutting*, which typically means digging above the working level but also can mean engaging the bucket at the dump radius. Either way, the bucket is at least partly pulled down a face rather than up or across it. The bucket is usually held in a dump position, teeth down, then lowered onto the face and dragged in. Chop cutting is sometimes used instead of a key cut in spoil-side operations to clean the highwall. However, chop cutting is hard on the rigging, ropes, and bucket, and can increase downtime and repair costs and decrease productivity. Productivity is further decreased by the lower fill factor and increased drag-to-fill time. Advance benching generally requires a longer swing angle as well. Although efficiency varies, typically a reduction of 10%–20% of the conventional rate should be used for initial estimates. When practical, chop cutting should not be done above the height of the fairlead, or else production and maintenance can be significantly impacted.

If the material in the advance bench is extremely unconsolidated, it is sometimes convenient to build a *buckwall* (visible in Figure 10.3-5A) out of dry competent material removed from another area of the set and placed as a retaining wall at the toe of the spoil. The unconsolidated material is then placed and contained behind the buckwall. A buckwall can be used to help stabilize any spoil pile.

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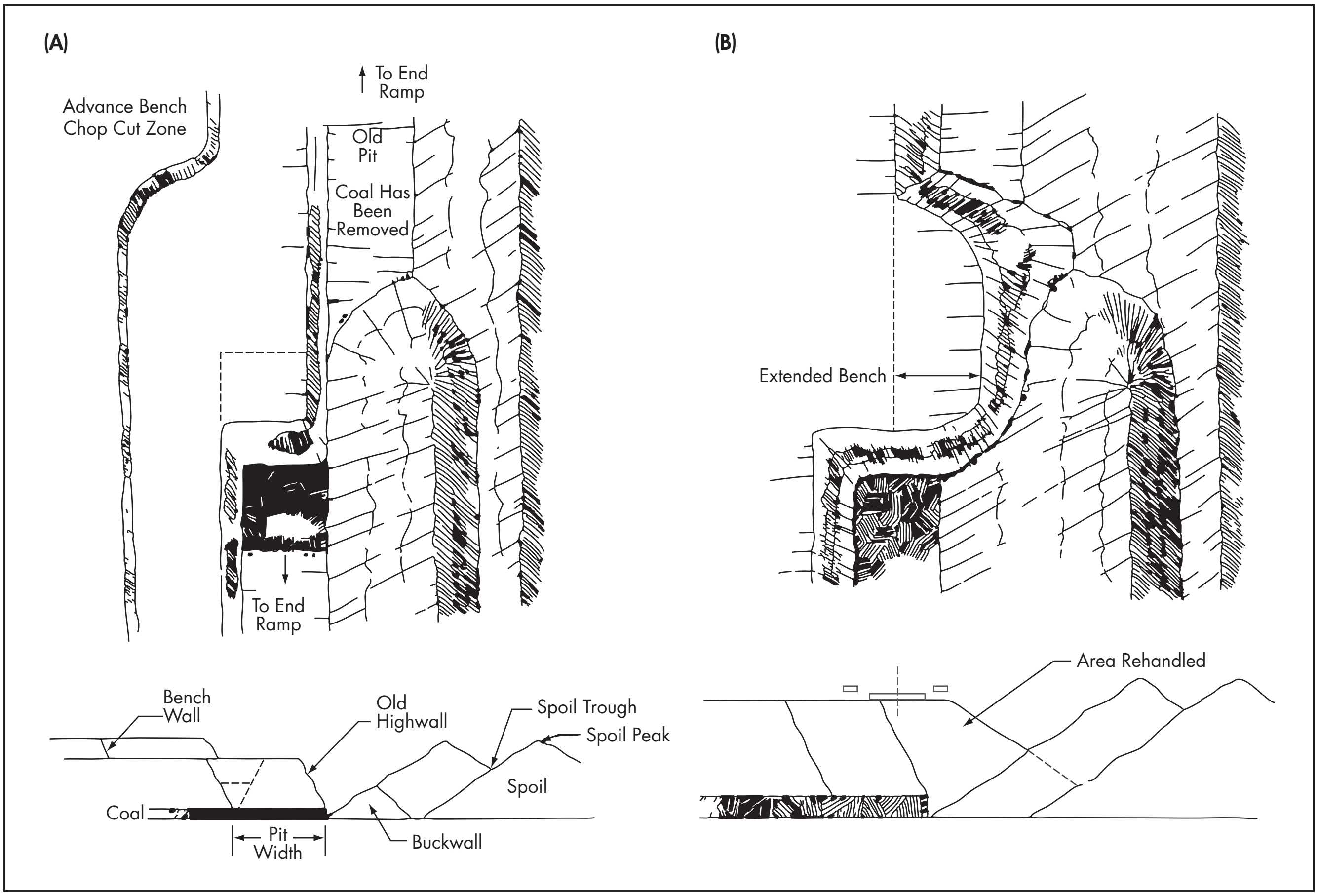

Source: Humphrey 1990.

Figure 10.3-5 (A) Advance benching and (B) extended benching

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Extended Benching

To extend operations in deeper overburdens, the alternative methods of extended benching and spoil-side benching (discussed in the next subsection) can be used to remove material to a depth of about twice that achievable by simple side casting. However, these methods come at the price of increased rehandle, slower cycles, and more complex planning, although the impact on rehandle and cycle time can be reduced by use of auxiliary equipment. These methods can also be used on a temporary or localized basis around ramps, spoil, or highwall slumps, at high spots in the overburden, or inside curves.

In extended benching (Figure 10.3-5B), the dragline places the driest, most competent material from the set against the old highwall. Enough material is placed so that, after leveling by dozers, it forms a bench. The dragline then moves out onto the bench in a position closer to the spoil. As excavation progresses, the bench is removed. This method can be used in two-seam operations as well. A disadvantage of extended benching is that the swing angle is long, lowering total production.

When calculating production requirements, it is important to remember that the rehandle material in the extended bench is loose material, and therefore has a different bucket factor.

For long-term applications, extended benching is frequently combined with cast blasting and push dozing (Figure 10.3-6A), both of which are effective for moving material short distances downhill. Blasting lowers the bench level, decreasing rehandle; it also moves some material to its final place, increasing production. The cast-blasting profile is then leveled by dozers to form the extended bench.

Bench height and width should be designed to take maximum advantage of dragline hoisting. However, although high hoisting can lengthen cycles, the benefit of a lower bench height with its lower rehandle is usually the determining factor in setting bench height. In multiple-seam operations, the bench height may be predetermined by the upper seam (Figure 10.3-6B).

Two positioning issues often arise in extended benching: cleaning the coal toe and cutting the key. For cleaning the coal toe, because the extended bench covers the coal toe, it is clearly advantageous, especially with thicker seams, to position the dragline just outside the edge of the coal, which is the best position from which to clean the spoil toe away from the coal toe and thus minimize rib loss. However, doing so may require pushing the extended bench out a little farther than is necessary to meet dump requirements. For cutting the key, a lower bench without a setback prevents the dragline from being positioned directly over the key. The dragline can be positioned no closer to the new highwall than the rear-end clearance radius. To keep the highwall clean and well defined, a presplit blast is commonly used. The dragline can be positioned out on the extended bench at a distance from the highwall equivalent to the dump radius and then the key can be chop-cut. Alternatively, the key can be developed by

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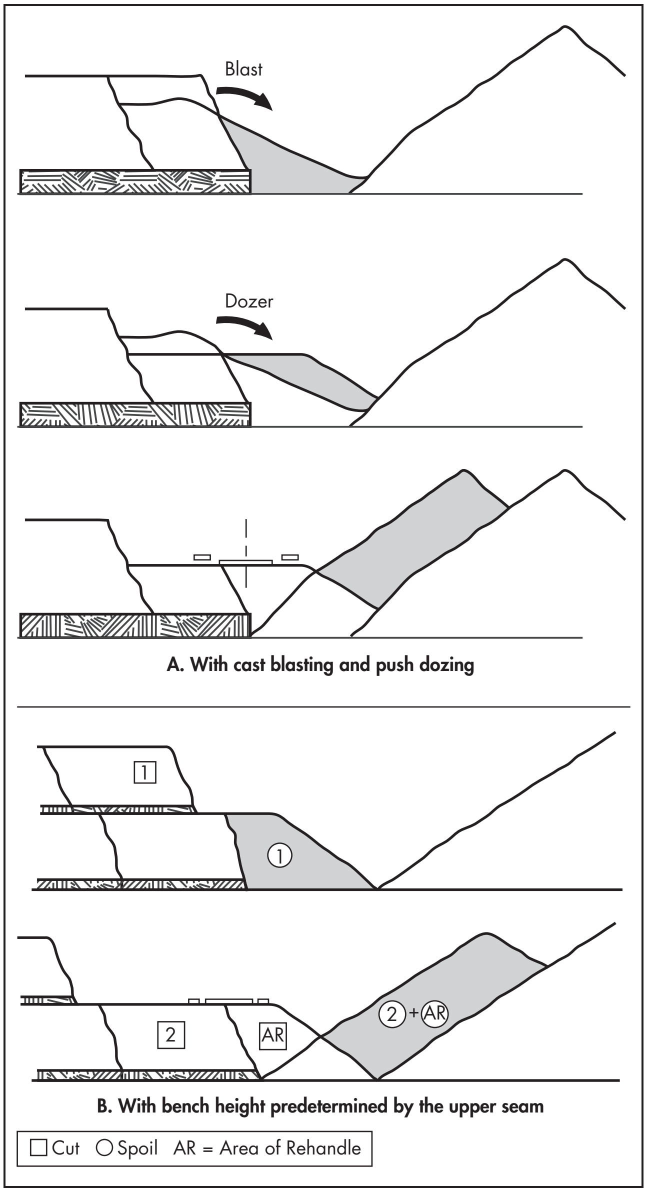


Source: Humphrey 1990.

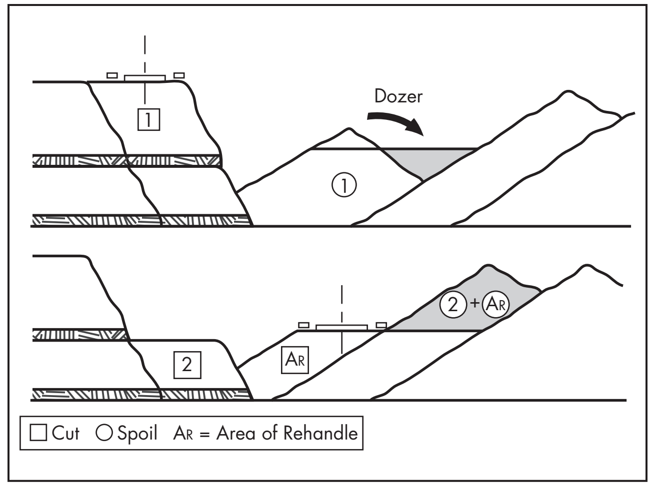
Figure 10.3-6 Combinations of extended benching

auxiliary equipment, typically dozers but sometimes backhoes or smaller draglines (Ingle and Humphrey 2004). In the latter case, it is critical to consider in advance the finer blasting fragmentation required for auxiliary equipment.

Spoil-Side Benching

In spoil-side benching (Figures 10.3-7 and 10.3-1), also called *pull-back*, overburden is removed in two independent passes. This method is common in two-seam operations and is virtually required in three-seam operations.

On the first pass, from the highwall side, material is moved by standard side casting. Spoil from that pass is allowed to ride up the highwall, and then the peak is leveled to form a pad for the second pass. The first pass is generally completed for an extended length of pit, whereupon the dragline bridges across to the spoil bench. On the second pass, the spoil bench can be removed in either direction, depending on pit



Source: Humphrey 1990.

Figure 10.3-7 Spoil-side benching

sequencing, spoil-bench development, and cable layout. On the spoil side, the dragline is positioned so that the key can be chop-cut with the boom perpendicular to the highwall.

The design height of the spoil bench, which also determines its width, is based on the reach requirement to chop-cut the key on the spoil-side pass, the dig depth, and, if the coal toe must be cleaned, the dump-height limitations of the dragline and tub position. If the first pass does not generate enough material to achieve the necessary spoil-bench height, material can be removed from the spoil-side position and placed one or two sets behind the dragline, much like an extended bench.

It is best to develop the spoil bench with auxiliary equipment; however, the amount of material to be moved may justify assistance from the dragline. And since the bench is developed in advance (the dragline is chopping in the direction of travel), it is better to develop a finish grade for only the road width of the dragline along the spoil edge, leaving the outer edge of the bench for the auxiliary equipment to finish-grade.

Spoil-side benching requires carefully managed cable moves and layouts, particularly when raising the bench level from the spoil side. With the cable on the bench and the dragline progressing toward the cable, the bench must be raised in halves. This requires moving the cable from side to side and sometimes also swinging over the cable. Swinging over the cable should be done only with a protective covering and carefully controlled operator technique.

Spoil-side benching enables operation of multiple draglines in a pit (Figure 10.3-1). In a fairly short pit, this can afford a very high production rate. However, it is difficult to schedule multiple draglines, primarily because of the complexity of matching the advance rate of draglines working on separate benches. Although production requirements can be proportioned, short-term variability in production rates invariably causes inefficiencies. Very few operations run tandem draglines in this manner for any length of time.

Dragline Production

The amount of material moved by a dragline is determined by the following basic parameters:

- Bucket capacity: how much material is put in the bucket
- Cycle time: how fast the bucket is cycled

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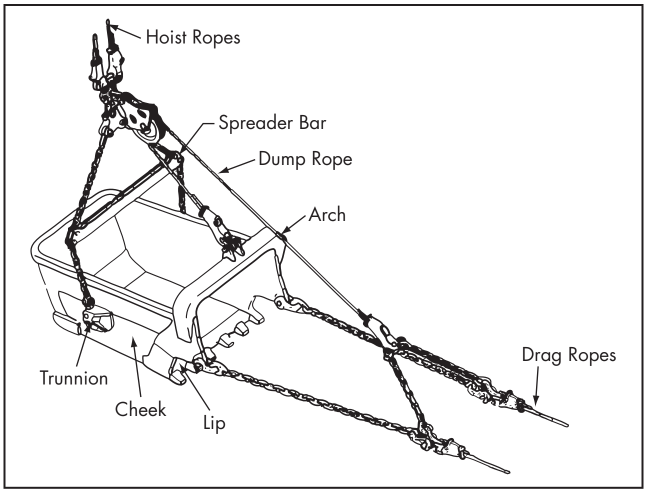

Source: Humphrey 1990.
Figure 10.3-8 Dragline bucket rigging

- Operating hours: how many hours per year the dragline is kept digging

Bucket Capacity

Bucket capacity was historically measured in accordance with SAE standard J67 (1998). This standard calls for a struck-top and front-face calculation and then subtracts 10% of the calculated volume to account for the slope of the front face. It also uses estimating factors to assist with the calculations of the complex curves of a typical bucket. However, the final so-called "rated capacity" in no way represents the behavior of material in the bucket. Rather, it simply provides a uniform method for comparing bucket capacities. For calculating production, the rated capacity must be adjusted for material swell and fill characteristics.

The material in a bucket is loose, so the rated capacity of a bucket can be thought of in terms of loose cubic meters (LCM) or loose cubic yards (LCY). Material swell changes with handling and varies within a bucket or pile of material; within a large pile, it also varies with time. Additionally, it can differ dramatically from the swell in a shovel dipper or truck body.

To account for swell and fill, two factors have been introduced:

1. The *swell factor* F_s is affected most noticeably by fragmentation and material composition, but also by bucket design.
2. The *fill factor* F_f, although more complicated to determine, generally has a larger impact on production variability and should be studied carefully. Fill is affected most noticeably by fragmentation and operator technique, but also by bucket-rigging configuration (Figure 10.3-8) and design.

F_s and F_f depend on material characteristics, digging method, rigging configuration, and bucket design; they do not remain constant if any of these variables change. They are thus best determined from field data, optimally by starting with bucket count and block volume, and then—for a particular material, operating method, and rigging setup—calculating the bank volume moved per bucket.

F_s for material in a dragline bucket is typically about 1.3 loose volume per bank volume. F_f is typically about 0.90 of the SAE J67 rated capacity. F_s and F_f can be combined into a single bucket factor, typically about 0.70, meaning that a bucket rated at 100 m^3 typically carries about 70 BCM. The typical distribution of payloads is about ±10%.

Cycle Time

The digging cycle of a dragline is comprised of five main components: (1) drag to fill, (2) hoist and swing, (3) dump, (4) return swing and lower, and (5) bucket-spot. The time required for each varies depending on a number of factors, most notably dig depth, hoist height, and swing angle. Other variables include material characteristics, dragline performance speeds, and operator proficiency. Because of the diversity of these factors, even machines of the same model and design have different cycle times. Typical designed cycle times for larger machines are in the range 50–60 seconds for a 90° swing with a low dump. A typical cycle is dominated by components 2 and 4; about 70% of a typical cycle is required to get the bucket over to the spoil and back again.

Component 2 of the cycle (hoist and swing) is actually three independent movements: swinging, hoisting, and paying out drag. Each has a specific time requirement. For almost any dump point, one of these movements takes more time than the others. Therefore, two movements are retarded intentionally so that the slower dependent movement has time to coincide at the dump point. However, drag pay speeds are so rapid that they are seldom the dependent movement. It is convenient to think in terms of the curve that the bucket follows at maximum hoist and swing speeds. This swing–hoist coincidental curve represents the points in space that make maximum use of the time available for both functions. Cycles dumped below the curve are *swing dependent*; cycles dumped above the curve are *hoist dependent*. For example, if a particular cycle is a long swing with a short hoist, then it is below the curve and swing dependent. Hoist distance and swing angle can be controlled by pit and digging pattern design, and performance speeds can be affected by operating technique. Thus it becomes obvious that a dragline operator should dump each bucket not at the peak of the spoil but rather near the swing–hoist coincidental curve. Although this is not practical for every cycle, the more coincidental cycles that occur, the more efficient the operation.

Hoist distance and swing angle are minimized by optimizing the bench height and the digging positions of the machine while minimizing the number of relocations (which cause nonproductive propel time). Obviously, machine positioning is limited by the key cut, drag-rope clearance, tail clearance, and reach requirements.

Bucket speed during these independent movements is very dependent on operating technique. Hoist speed is a function of the load in the hoist rope, which is directly related to bucket position. Because hoist acceleration is extremely quick and represents a small proportion of the hoist time, acceleration time does not affect hoist time significantly. To keep the bucket in the carry position, tension must be maintained on the drag rope, which increases the load in the hoist rope. Typically, the hoist load is about 120% of the bucket and payload weight. The closer the bucket is carried to the boom, the greater the hoist load, which ranges from about 110% to 140%. The change in hoist speed is directly proportional to the change in hoist load. So the further out the bucket is carried, the faster it hoists. However, carrying the bucket too close to the dump radius generally causes material to slough off the front of the bucket, reducing the fill factor.

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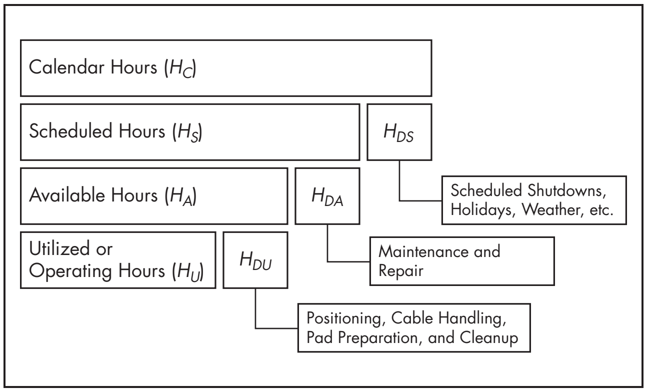

Source: Humphrey 1990.

Figure 10.3-9 Relationship between calendar hours and operating hours

Table 10.3-1 Operating efficiency factor

	Utilization			
Availability	**Excellent**	**Good**	**Fair**	**Poor**
Excellent	0.83	0.77	0.70	0.65
Good	0.78	0.72	0.66	0.61
Fair	0.74	0.68	0.62	0.58
Poor	0.69	0.64	0.59	0.54

In contrast, swing time is very dependent on acceleration. Acceleration to full speed and deceleration to stop requires about 60° of the swing, which is about 85% of the swing time for a swing angle of 90°. Acceleration and deceleration are relative to the rotational inertia, which is a function of the mass times the square of the radius of the center of mass, so it is greatly affected by bucket location. On longer swings, where hoist is not a factor, keeping the bucket in tight as long as possible improves swing time.

Of bucket capacity, cycle time, and operating hours, inefficiency in cycle time is the most difficult to diagnose and improve.

Operating Hours

Ostensibly the easiest performance parameter to measure is time. In reality, there are more ways to categorize delays and define losses than there are mines around the world. The ubiquitous "hour" is possibly the single most misleading term used in mining. The basic goal is to reduce the number of hours we have to work with down to the number of hours actually worked. Clearly understood definitions are critical to production reporting and estimating.

Figure 10.3-9 shows the relationship between calendar hours and operating hours. Although there are about 8,760 calendar hours in a year, draglines typically operate for 6,000–7,000 hours per year. For initial estimating, it is adequate to combine availability and utilization values into a single operating efficiency factor (Table 10.3-1). Typical availability and utilization values are each about 85%, which provides an operating efficiency of about 72% of the scheduled hours. Availability and particularly utilization can be affected significantly by application. The same dragline on a highwall pass will see a significant difference in operating efficiency on a spoil-side pass where it is chop-cutting and pad-building, typically due to increased propel time, decreased dump-rope life, increased bucket maintenance, and so on.

Apart from the obvious minimization of maintenance and utilization delays, for maximum production it is key to use the available hours efficiently. Most dragline operations can realize significant production gains by reducing cycles that do not contribute directly to ore production. The most significant contributors to inefficient operation are rehandle, nonproductive cycles, and poor planning. To address these contributors, the following are important:

- **Effective management of rehandle.** The use of auxiliary equipment and careful planning are the best tools to manage rehandle. The cost of rehandle should be thought of in terms of incremental costs of ore production, rather than the difference in cost per unit of production for dragline operation compared to that for auxiliary system operation. In addition, in certain applications, a slight increase in rehandle (e.g., changing a bench level or pushing out an extended bench) can improve production due to the effect on cycle and propel time.
- **Productive digging.** Nonproductive cycles are often difficult to measure but can decrease production significantly. There probably isn't an operation at work today that couldn't reduce the amount of time the dragline does pad preparation and cleanup work by better use of auxiliary equipment. Draglines should not waste time heeling the bucket to level the pad or pushing down the roll. In addition, efficient coordination of the ground crew during cable layout and pad preparation can minimize move times.
- **Quality operation planning and coordination.** The amount of detailed planning required to operate a dragline most efficiently should not be underestimated, particularly for nonroutine digging. Specific tub positions, bench levels, and material placement need engineered plans. It is important to involve dragline crews in planning to ensure their understanding and consensus. In addition, it is a mistake to assume that everyone can easily create, from a two-dimensional drawing, a three-dimensional (3-D) operation. Rather, it is wise to use 3-D software or sandbox models to work through operations in advance, especially complex operations such as ramp crossings. In complex operations, the use of "playbooks," with a diagram for every tub position to describe the digging and dumping points, can be extremely valuable.

Dragline Selection

An important concept to keep in mind when sizing or selecting a dragline is to select the dragline for the mine plan, not the mine plan for the dragline. Draglines are engineered systems, generally customized for an application; even when purchased used, they can be modified during reassembly to better fit an application. The two major parameters used to select a dragline are dump radius and allowable load. Occasionally, other parameters such as ground-bearing pressure of the tub, rear-end swing clearance, dump height, and dig depth may also affect selection.

Dump radius R_d is the horizontal distance from a machine's center of rotation to the hoist rope when the bucket is vertically suspended. Part of this radius is consumed by the stand-off distance S_o (Figure 10.3-10), which is the distance from the center of rotation to the crest of the old highwall. The

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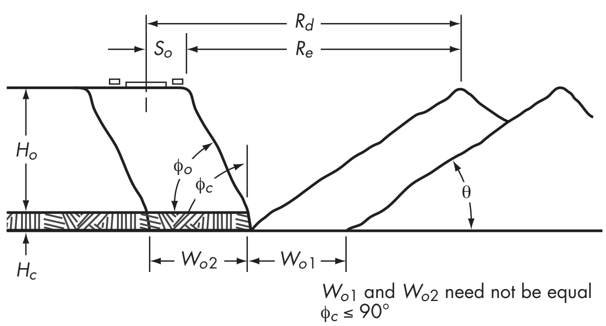

$$R_e = \frac{H_o}{\operatorname{Tan} \phi_o} + \frac{H_c}{\operatorname{Tan} \phi_c} + \frac{W_{o2}\, H_o\, F_s}{W_{o1} \operatorname{Tan} \theta} + \frac{W_{o1}}{4}$$

$$H_o = \frac{\left(R_e - \dfrac{W_{o1}}{4} - \dfrac{H_o}{\operatorname{Tan} \phi_c}\right)}{\left(\dfrac{1}{\operatorname{Tan} \phi_o} + \dfrac{W_{o2}\, F_s}{W_{o1} \operatorname{Tan} \theta}\right)}$$

Source: Humphrey 1990.
Figure 10.3-10 Dragline geometry

remaining dump radius is the effective radius R_e; thus $R_d - S_o = R_e$. Stand-off distance varies depending on machine size, operational history, and overburden conditions. In the absence of field data, for planning purposes the minimum stand-off distance is commonly considered to be 50% of the width of the dragline from the outside of the shoes or 75% of the tub diameter.

The allowable load, sometimes called the maximum suspended load or rated suspended load, is the maximum weight of bucket, rigging, and material for which the dragline is designed to provide optimum performance. Standard-duty buckets including rigging weigh about 1.2 t (metric tons) per rated m^3 (2,000 lb per rated yd^3) of capacity, although they may vary from 1.1 to 1.4 t (1,800 to 2,300 lb), depending on the application. Overburden densities are site specific, but 1.8 t/LCM (3,000 lb/LCY) is a common approximation. The combined weight of a standard bucket and material load is then about 3.0 t per rated m^3 (5,000 lb per rated yd^3). Therefore, an operation that requires a dragline with a 46-m^3 (60-yd^3) bucket requires the dragline to have about a 138-t (300,000-lb) allowable load. Allowable load is calculated for a 100% full bucket (peak bucket load), even though the average fill used for production calculations is less (e.g., 90%).

For a given dragline model, or more correctly a given frame size, the allowable load can be varied by changing the dump radius. Basically, a shorter reach means a larger bucket. This design change is limited by the hoist load that the gearing and motors can handle (maximum allowable load) or, conversely, the longest boom that the frame can handle. The general rule is: the longer the reach, the deeper the depth that can be handled without rehandle and the less rehandle. However, the best choice is almost always the largest bucket at a shorter reach, despite the higher rehandle. The larger bucket more than makes up for the additional rehandle with additional production.

By calculating bucket requirements using a standard range of production values—8,400 scheduled hours, 60-second cycles, 72% operating efficiency, and a 70% bucket factor—it can be calculated that a dragline produces about 250,000 BCM/yr per m^3 (250,000 BCY/yr per yd^3) of rated bucket capacity. This unit of production per unit of bucket capacity is a convenient general factor called the production factor or digging index (Humphrey 1990), and can be used to quickly estimate the annual capacity of a given bucket, or conversely the bucket required for a given production.

The production factor is commonly used to compare differences in dragline operations, either one application vs. another or one dragline vs. another. With historical data, production factors for specific applications can be developed over time and used for planning and forecasting purposes. Operating mines commonly have production factors of 200,000–300,000, depending on application, cycle time, and efficiency. Production factors are often calculated on an hourly basis, with care given, of course, to which kind of hour is used. The hourly production factor can be useful for measuring the efficiency of a specific operation within an application (Kennedy 1990).

The application of production factor presumes that overburden production requirements are known and have been adjusted for any ore losses in the pit and plant and any rehandle expected for the scenario.

Caution should be used with volumetric terminology. Some operations, mostly outside of North America, report production in terms of *prime* (also called *virgin* or *in-situ*) volumes but label them "bank" volumes. The prime volume is the actual volume of overburden above the coal that was uncovered and does not include rehandle. In fact, most mines typically experience 5%–10% of additional operation rehandle for ramps, bench fill, and the like. A production factor based on prime volume is thus smaller than can be expected in actual operation. Prime volume is adequate for comparing machines using the same method at the same mine; however, it is not a true measure of individual dragline productivity. It is advisable to use the terminology *total (including rehandle) bank cubic meters* (TBCM) and *prime (not including rehandle) bank cubic meters* (PBCM), or for cubic yards, TBCY and PBCY.

Draglines as Loaders

Although draglines are normally used to direct-cast material, they are also sometimes used as loaders. In that case, the normally imprecise dumping technique of the dragline must be altered to something as precise as is used when spotting the bucket in the dig. Most experienced operators have little difficulty adapting to point dumping, and the method has been used to load hoppers, trucks, and barges. Generally, it is most useful when the pit bottom is unsuitable for truck traffic and too deep for a hydraulic excavator. For example, it is common in Florida phosphates for a dragline to mine the phosphate matrix and dump it into a slurry sump on the highwall. The sump is about the size of a large mining truck body.

Contrary to the initial mental image of this method, the bucket does not swing over the target. Rather, the swing is stopped with the bucket in the carry position and the drag is payed out to dump. Trucks are positioned facing away from the dragline so that the bucket enters over the tailgate. The

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Figure 10.3-11 Typical bucket-wheel excavator

precision required to determine the truck position (i.e., to spot the truck) is no different than is required for any other loading tool. However, because the spot cannot be marked by the bucket as in shovel operation, position markers may have to be provided. As the dump target is generally on the highwall side, it is easy to locate the target so that the dragline has a very short swing angle and a short cycle time.

To reduce the dump distance and place the material in a smaller area, the dump ropes can be shortened. This has the adverse effect of increasing the load in the dump rope and decreasing the dump-rope life. Alternatively, the dump target can be elevated (such as to an advance-bench level) to shorten the dump distance. Of course, elevating the dump target much above the operator's line of sight compromises hoist time and visibility.

Typical dragline operations are viewable by satellite-image software in the vicinity of these coordinates:

- Simple side casting in a lignite mine: 47° 5.19′ N 101° 19.63′ W
- Cast blasting in a thick-seam coal mine: 43° 41.36′ N 105° 18.80′ W
- Cast blasting with spoil side in a multiple-seam coal mine: 21° 28.85′ S 148° 22.60′ E

BUCKET-WHEEL EXCAVATOR SYSTEMS

The bucket-wheel excavator (BWE) (Figure 10.3-11) is one of the grand machines of the mining industry and traces its origins to drawings by Leonardo da Vinci. The original concept from the late 1800s was technologically challenged by advances in the steam shovel, had its practical beginnings in the early 1900s, but had its first real mining applications in German lignite mines during World War I (Rasper 1975).

For purposes of comparison, the BWE system is a high-capital-cost, low-operating-cost system that has limited flexibility and can operate through a limited range of applications with sensitivity to geologic variance.

BWEs are highly customized and vary in design more than do any other mining machines, to the extent that nearly every machine is not just unique but almost dissimilar. The machines are very robust in design and consequently very long lived. Their most common mining applications are mining unconsolidated overburdens and lignite, handling bulk materials such as stockpiles and load-out facilities, and heap leaching pad construction and removal.

Three major types of BWE exist in mining operations:

1. Systems that direct-feed into a shiftable conveyor system that connects to a series of other conveyors and a discharge system; these machines can weigh up to about 12,500 t (14,000 st) and cut material from a bank height of more than 50 m (160 ft) at rates of more than 10,000 m^3/h (13,000 yd^3/h)
2. Systems with long discharge booms that direct-place material into the spoil
3. Compact systems, generally two crawler machines used for bench heights of about 10–15 m (30–50 ft) with production rates of 1,000–2,000 m^3/h (1,300–2,600 yd^3/h)

When the BWE is used to reclaim stockpiles for rapid loading of trains and ships, it is often rail-mounted. Because the shape of the stockpiles is known, this application is ideal for semiautonomous control.

The numbering system for BWE models is probably the most logical of all mining-equipment numbering systems. The format of a typical German BWE model number is (ThyssenKrupp Fördertechnik 2005)

$$\text{Sch Rs}_{\frac{1{,}600}{2.5}-28\times 12}$$

where

Sch = *Schaufelradbagger* (or simply *bagger*), the German name for the BWE; German manufacturers thus lead their model numbers with S, SH, SR, or Sch
R = *auf Raupen*, meaning that the system is on crawlers
s = *schwenkbar*, meaning slewable
1,600 = bucket size, L
2.5 = cutting depth, m
28 = cutting height, m
12 = crowd length, m

Other countries may order the components differently, as, for example, in the Russian ER$_{1{,}600\frac{28}{2.5}\times 12}$. Compact or nonstandard BWEs generally have simpler model numbers, such as C500, which is a compact BWE with a 500-L bucket.

BWE productivity is highly specific to design and application. Cutting forces, wheel-rotation speed, number of buckets, slewing speed, and material characteristics all affect the production rate. As a (very rough) general rule, the peak capacity in cubic meters per hour is from 3 to 6 but usually about 4 times the capacity of a single bucket in liters. So, for example, the German "Sch Rs" example above has a bucket size of 1,600 L so would have a capacity of about 4,900 m^3/h (6,400 yd^3/h), where hours are operating hours, not scheduled hours (for more information, see the "Operating Hours" section).

BWEs are used primarily for unconsolidated materials. Even the largest-capacity BWE has a relatively small bucket, ≤5 m^3 (3.5 yd^3), and so has relatively limited capacity for rocky materials. Additionally, because of the associated conveyors, BWEs require linear, flat-floored mining faces that advance in straight or radial patterns. Thus their main application has historically been large lignite mines, although their largest U.S. application was in the 1960s and 1970s in the Illinois (United States) coal basin where they were used to direct-spoil glacial till covering the sedimentary overburdens.

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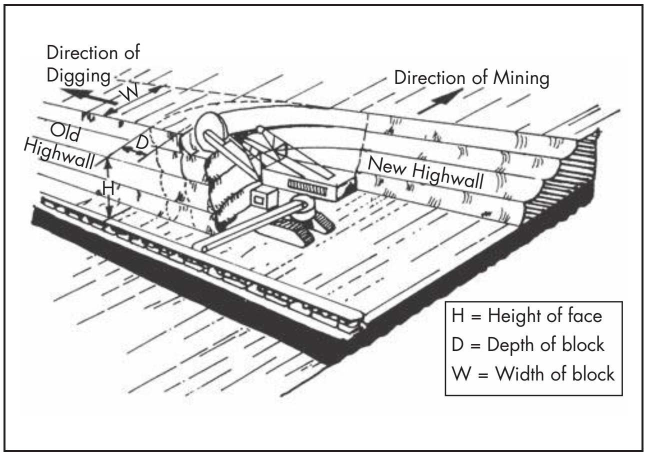
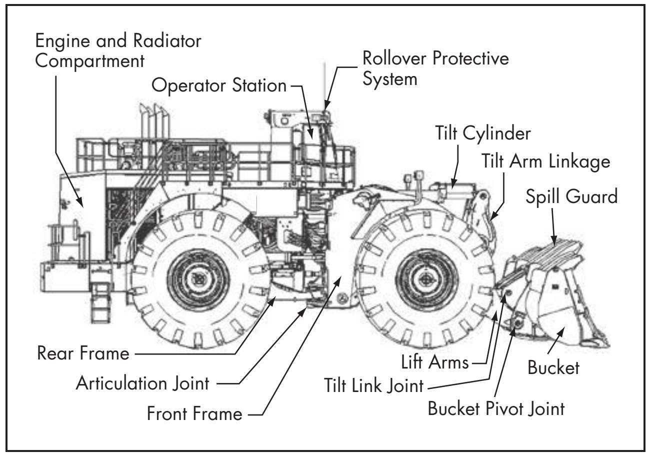


Source: Atkinson 1992.

Figure 10.3-12 Bucket-wheel excavator used for block digging a lateral terrace cut



Courtesy of Caterpillar, Inc.

Figure 10.3-13 Wheel loader (smallest-capacity bucket)

BWEs can operate in linear pits with a spoiling process operating in parallel to fill the void a short distance away, or in multiple benches in deep pit mines with outside spoil dumps. Pit-face lengths vary from 800 to 4,000 m (2,500 to 13,000 ft), but are usually 1,500 to 2,500 m (5,000 to 8,000 ft). The BWE advances along the face, taking a cut width consistent with its design, up to 90 m (300 ft). The digging action is either a lateral terrace cut (Figure 10.3-12) or a vertical dropping cut, depending on machine design. At the end of the pit, the BWE can cut perpendicularly to the face conveyor for a short distance by means of its slewable discharge boom and or a belt wagon. This plunge cut enables the BWE to establish a face to work in the opposite direction, eliminating the need to deadhead.

Crawler systems for BWEs have as many as 12 crawler sets, commonly set up in an asymmetric arrangement with independent steering to enable larger machines to steer. Systems for use in soft-underfoot conditions can exert ground-bearing pressures much lower than for shovels, about 1.0–1.5 kg/cm^2 (14–21 psi). Tramming speeds are 4–12 m/min (13–40 ft/min) and a typical turning radius for a larger machine is 50–100 m (150–300 ft).

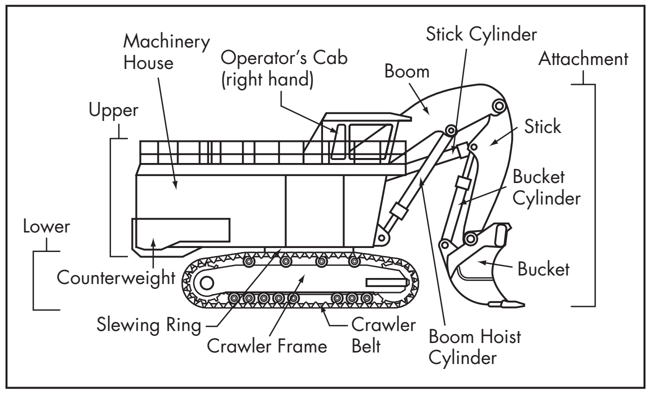
Typical BWE operations are viewable by satellite-image software in the vicinity of these coordinates:

- Compact BWE with multiple benches in advance: 39° 46′ 26″ N 111° 15′ 25″ E
- Large BWE prestripping in advance of dragline: 26° 43′ 14″ S 27° 57′ 37″ E
- Large BWE in large open pit: 50° 54′ 38″ N 6° 30′ 17″ E

LOADER AND HAULER SYSTEMS

More material is moved by loaders and truck haulers than by all other excavation systems combined. The deciding factors in the selection of this system are typically the qualitative characteristics of flexibility and the probability of achieving production and cost targets.

Loaders and truck haulers excel in flexibility. They are not dimensionally constrained by operating method, and so are able to move in any direction for any distance. They can thus work in constrained or irregular geology and terrain, and can be added incrementally, both of which make them virtually the only choice for use in very deep pit mines. Their flexibility also enables a mining operation to adapt quickly to changes in commodity prices, geology, and other influences that cause the original mine plan to change, as it inevitably does.



Courtesy of P&H Mining Equipment, Inc.

Figure 10.3-14 Hydraulic shovel (middle-capacity bucket)

Because haulers always operate in parallel—and, in large systems, loaders do so as well—the impact of adverse performance on the part of any individual unit is minimized. Additionally, the system can operate through a large range of geologies and climates. This all provides for a dependable system with little unpredictable variation in efficiency, production, or cost.

For purposes of comparison, the loader and hauler system is a low-capital-cost, high-operating-cost system that is very flexible and can operate through a broad range of applications with low sensitivity to geologic variance.

Loading Tools

Loading tools are a specific class of excavator that depend on a separate independent haulage system. The most common types of loading tool, from small to large capacity, are wheel loader (Figure 10.3-13), hydraulic shovel (Figure 10.3-14), and mining shovel (Figure 10.3-15). In mining they are used in conjunction with haulers, most often off-highway mining trucks.

In the past 30 years, and particularly the past 10 years, the development of larger, more reliable wheel loaders and

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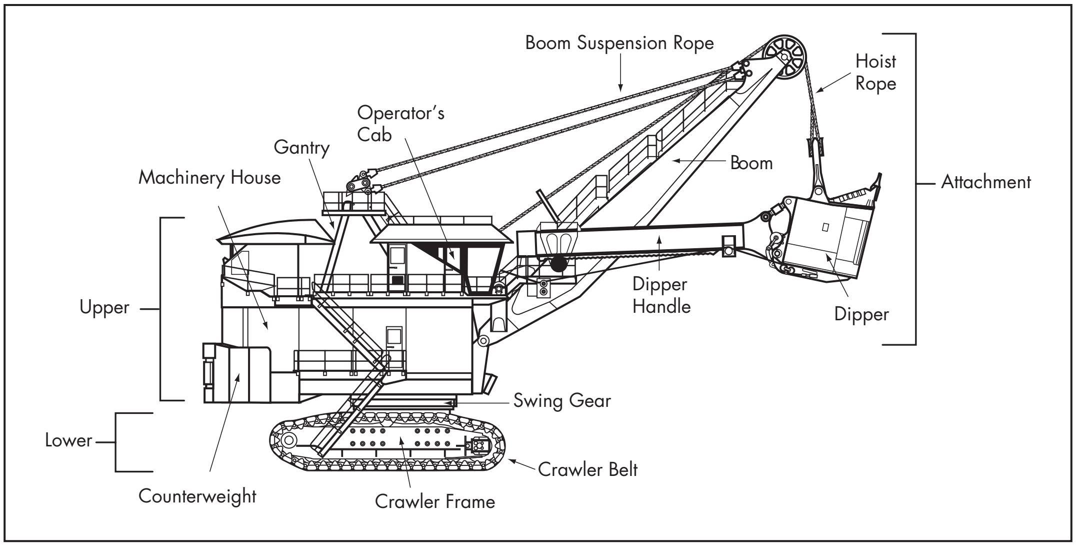

Courtesy of P&H Mining Equipment, Inc.

Figure 10.3-15 Mining shovel (largest-capacity bucket)

hydraulic shovels has encroached into what was previously the exclusive territory of mining shovels: the loading of large off-highway trucks. These trucks, also called large mining trucks, generally have capacities of >135 t (150 st). The growth in wheel loaders and hydraulic shovels has lead to a new delineation in the loader market, with wheel loaders predominant at the lower end, hydraulic shovels in the middle range, and mining shovels at the upper end of bucket capacity (Figure 10.3-16).

Selection

Because all loading tools perform the same basic function (i.e., they load trucks), the differences among them lie in other characteristics. Likewise, due to design differences and resulting differences in capital costs and operating costs per unit of production, evaluations of net present value can prove useless. Deciding factors are more likely the qualitative characteristics of capacity, mobility, flexibility, life, and support requirements.

Capacity. The range in capacity of the three types of loading tool is a differentiating characteristic. For operation with large mining trucks, their capacities are as follows:

- Wheel loader: 27–45 t (30–50 st)
- Hydraulic shovel: 27–81 t (30–90 st)
- Mining shovel: 54–110 t (60–120 st)

These differences in capacity spread even further when compared on an annual production basis, because of cycle-time and operating-hour differences. Generally, when comparing machines of similar size, the mining shovel cycles more times per year than does a hydraulic shovel, which in turn cycles more times than does a wheel loader. This is due primarily to differences in operating hours but also, to a degree, in cycle time. Production is most heavily influenced by the degree of utilization (i.e., the extent to which a tool is kept in use when it is mechanically available). It is also influenced by consistency and efficiency of application, with problems arising when the pit layout is poor, resulting in long swing angles, excessive moves, and other workarounds. Table 10.3-2 shows the influence of swing angle on cycle time and hence production level of a loading tool.

The optimal production factors for loading tools are approximately as follows (the number in units of BCY per year per cubic yard is the same as the number in units of BCM per year per cubic meter; production factors are discussed in the "Dragline Selection" section):

- Wheel loader: 330,000 BCM/yr per m^3 of dipper capacity
- Hydraulic shovel: 350,000 BCM/yr per m^3 of dipper capacity
- Mining shovel: 400,000 BCM/yr per m^3 of dipper capacity

Mobility. If mobility is critical to an operation, the best choice is usually a wheel loader. In operations having multiple faces that require frequent relocations or want a backup unit for multiple loaders, the wheel loader is uniquely capable of rapid relocation. However, recent developments in larger low-boy or float transporters has extended the capability of medium to large hydraulic shovels to rapidly relocate.

Flexibility. The capability to work in faces of different heights or to dig at different levels of a face is an advantage. Wheel loaders are most productive at face heights of at least three times the bucket height, although at shorter face heights they can drive forward through unconsolidated or well-broken material with only slight impact on productivity. In less-consolidated material, hydraulic shovels can penetrate a bank at different levels to separate material at the face with only slight impact on productivity.

Most hydraulic shovels can also be configured as backhoes or mass excavators for digging below grade and loading trucks below or at operating level. Loading trucks below operating level allows use of a number of spotting techniques and shortens cycle times. Loading trucks at operating level, although slower, is obviously desirable when the pit bottom is wet. Backhoes are generally limited to use where face heights are about equal to the stick length.

Mining shovels require a face height of about 50% of their point sheave height, basically about the height of the teeth when the dipper stick is horizontal. However, they can operate at higher bench heights than can wheel loaders and hydraulic shovels, which reduces other operating costs. A higher face

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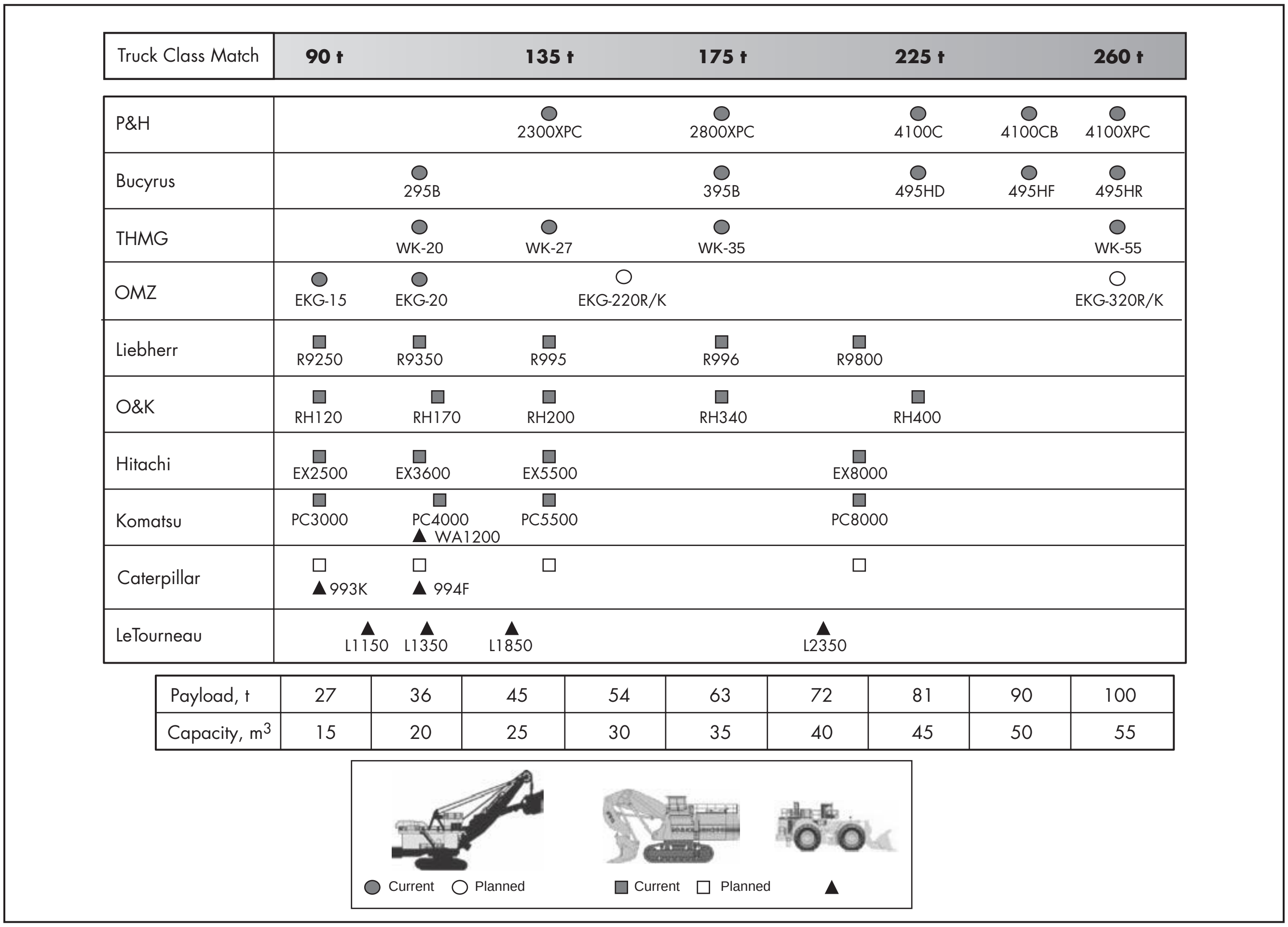

Courtesy of Caterpillar, Inc.

Figure 10.3-16 Loading tool industry offerings by manufacturer and class

Table 10.3-2 Impact of swing angle on production level for a loading tool

Swing Angle, degrees	Impact on Production, %
45	126
50	116
60	107
70	100
100	88
130	77
180	70

Source: Adapted from P&H Mining Equipment, Inc. 2006.

means fewer benches, fewer relocations, and lower drill-and-blast costs.

Life. Economic lifetimes for loading tools are generally as follows:

- Wheel loader: 5 to 7 years
- Hydraulic shovel: 7 to 10 years
- Mining shovel: ≥15 years

Of course, with enough replacement components, the life of any unit can be extended. Longer life is arguably a desirable feature: it is certainly necessary to justify a higher capital cost, but a system with a shorter life but lower operating costs and high resale value can be an equally good or better choice.

Support requirements. Several factors affect support requirements for a loading tool:

- **Drive system.** Large mining shovels are currently available only with electric drives. Wheel loaders and hydraulic shovels generally have diesel drives, although very few wheel loaders and some hydraulic shovels are optionally available with electric drives. Electric drives have lower and more consistent operating costs but require in-pit electrical reticulation systems comprised of electrical substations and power distribution cables involving specialized support equipment, personnel, planning, and operations. Diesel drives require fuel transport for refueling in the field but the equipment involved is usually common to mining operations using large equipment.
- **Digging profile.** Wheel loaders and hydraulic shovels, more so than mining shovels, can flat-pass and thus require minimal cleanup assistance. However, mining shovels, with their greater reach, can stand back farther from the face and so keep the truck back farther from the toe—although one could argue that this class of loading tool should not waste time doing cleanup work that is better left to auxiliary equipment.

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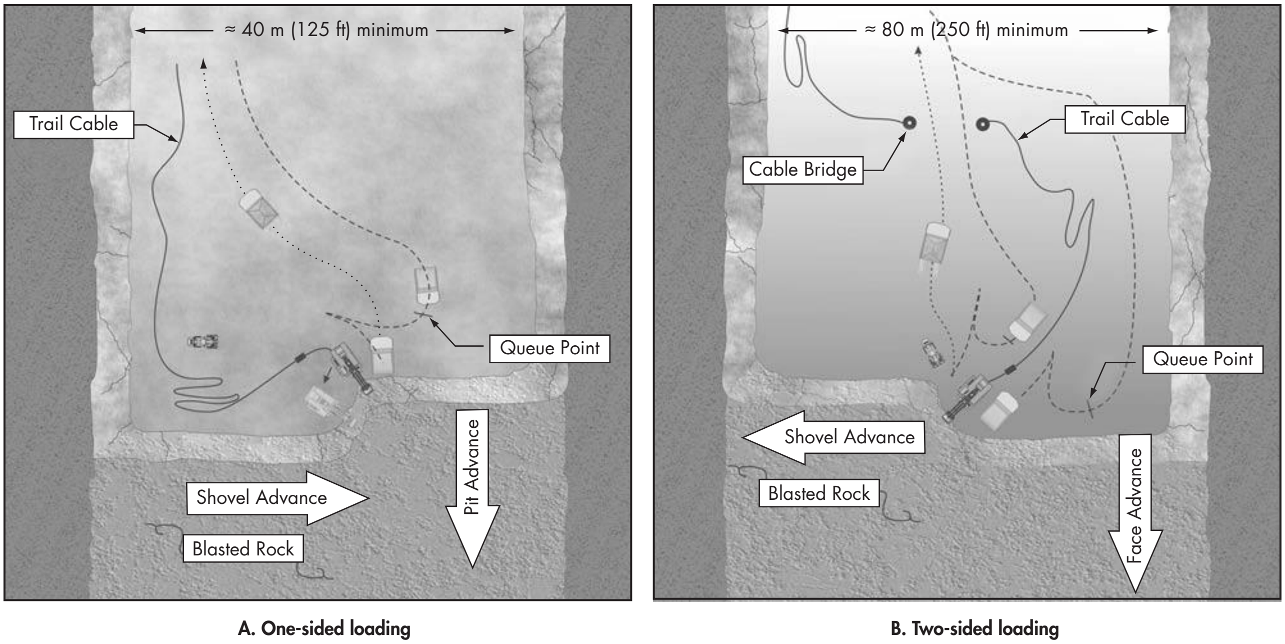

Figure 10.3-17 Loading-tool operating methods

- **Maintenance.** Larger hydraulic excavators and mining shovels have limited mobility and transportability, so all maintenance on these machines must be conducted in the field.
- **Material conditions.** The condition of the floor and bank affects the different types of loading tool differently. A wet or soft floor causes traction problems for wheel loaders, which can significantly impact tire costs and productivity. For soft floors, track shoes can lower the ground-bearing pressure exerted by a shovel. For very soft floors, extra-wide track shoes can minimize the ground-bearing pressure, although they reduce maneuverability due to the increased turning forces required.

Operating Methods

The most common loading-tool operating method is one-sided loading (Figure 10.3-17A), also called single-sided loading. This method requires minimal pit-support coordination and minimizes variations in traffic patterns and truck movements, the latter of which often significantly influences safety. The method also has a relatively small footprint, so it can easily be implemented in benches only 30–40 m (100–125 ft) wide. In the most common variation of the method, the truck stops or queues in a position where it can observe the loading operation, which allows the truck operator to ensure that the area is clear after the previous truck has been loaded and moved off. The location of the queuing and reversing points is generally left up to the truck operator, as it changes so frequently. However, choice of location can significantly influence cycle time, so training on efficient techniques is important. Road constrictions, pit obstacles, poor spot selection, cleanup, and cable handling are common causes of unnecessary delay. Traffic patterns are best designed to allow the truck to reverse with the shovel on the truck operator's left and thus always visible to the operator either directly or in the left side mirror.

One-sided loading has several disadvantages. Mining shovels and hydraulic shovels cycle in 30–35 seconds and spotting takes 45–60 seconds, so the shovel must wait for the truck, reducing production levels. In addition, the time required for a cleanup dozer to work in the spot area can delay operation.

These disadvantages can be addressed to some extent by two-sided loading (Figure 10.3-17B), also called double-sided loading, which reduces the shovel delay between trucks. Although this method appears to be just a doubled-up single-sided loading method, it actually adds some complexity. The road circuit is more complex because it requires a Y intersection, a much larger working face, up to three queuing points, and twice the cleanup and road-maintenance support. Electric-powered loaders also require a cable bridge and have more confined turnaround areas. All of these factors dictate a larger footprint for the loading area, so the method is more suitable for benches at least 80 m (250 ft) wide. Furthermore, near the ends of the working face, the work area becomes restricted and the shovel generally must revert to single-sided loading, so two-sided loading favors wide working faces. Two-sided loading can increase production by 5%–10%, with the general caveat that the more passes per truck, the less the benefit, since improvements come from reducing time between trucks.

Wheel loaders spot trucks somewhat differently than do the other types of loading tool (Figure 10.3-18). From a wheel loader, trucks are best spotted at a 45° angle, which allows the loader to approach the face and the truck at right angles with minimal turning between the two. A good setup requires less than one tire rotation of travel distance in each direction to load the truck.

A less-common method, drive-by loading (Figure 10.3-19), is used with bottom-dump trailers or backhoe loaders. It does not require reversing; rather, the truck merely drives alongside the loader. It requires a very narrow face with a long, clear

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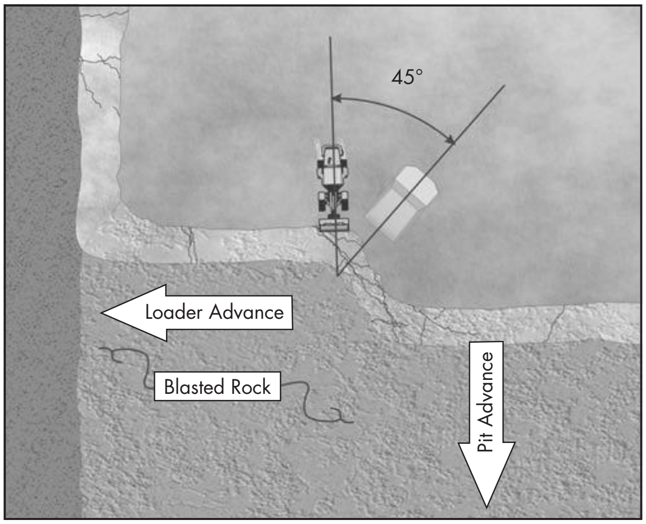
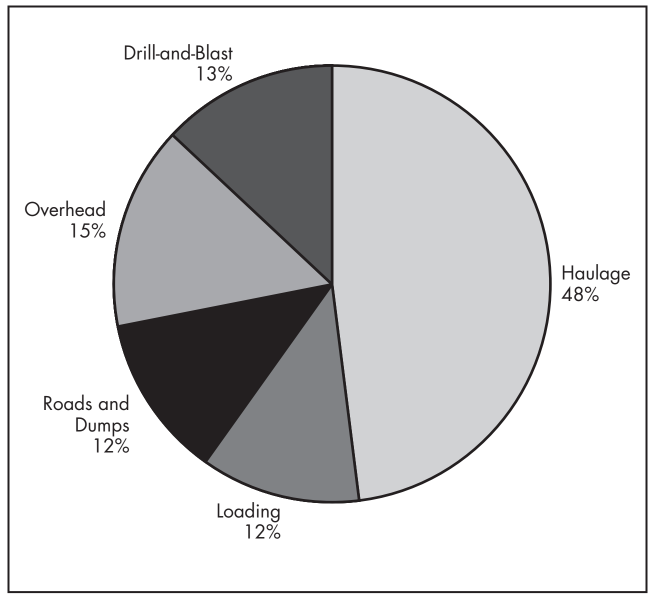


Figure 10.3-18 Wheel loader spotting a truck

Figure 10.3-19 Drive-by loading



Courtesy of Caterpillar, Inc.

Figure 10.3-20 Relative influence of haulers on open-pit production costs

bench adjacent so that the truck can approach along the bench and the loader need never move far from the bench.

Backhoes offer a few variations for loading trucks on the floor because the truck can back to the face at virtually any angle. The truck spot can be set up to bring the material over the side or through the tailgate, depending on the bench configuration. Some hydraulic shovels without independent pumps for hoist and swing can benefit from positions that emphasize either hoist or swing rather than working both together.

Matching Loading Tools to Haulers

Selecting the best loading-tool size to hauler size involves analyzing the number of passes required to load a truck and the number of trucks needed to match the shovel. The primary goal is to optimize the total loading and haulage cost.

Maximum production is determined by the loading tool, not the hauler. However, pit production costs are most significantly influenced by the hauler (Figure 10.3-20). Specifically, haulers account for nearly 50% of the total system cost, and loading tools only about 10%. Thus the following bears repeating: in general, *the loading tool drives production and the hauler drives cost.*

The strategy in matching loading-tool size to hauler size is to consider but not be ruled by the concept of matching passes and minimizing the number of passes per truck. The variety of loaders and trucks available make it virtually impossible to always achieve a perfect match. Another complication is that variations in material density and bucket fill ensure that no two dipper loads will be exactly the same; in fact, the distribution of dipper and truck-load sizes is nearly normal. This distribution pattern is considered in loader design, and some latitude exists in matching dipper loads to truck loads. Target payload should be changed only after consultation with the manufacturer.

Given that spot time cannot be less than shovel cycle time, it follows that the more passes per truck, the more the shovel (and therefore the system) produces. The downside of this premise is that it favors selection of smaller trucks, which of course means higher operating costs and more congestion. The premise is therefore best considered as suggesting that truck size, not shovel size, determines the number of loading passes.

In reality, variations in production or fleet costs caused by pass match are not as significant as the consequences of undertrucking or overtrucking a fleet. Undertrucking a typical fleet by only one truck offsets the production advantage of the additional pass per truck; likewise, overtrucking a fleet by only one truck offsets the cost advantage of a larger truck with fewer passes per truck.

Track Dozers

Large track dozers (Figures 10.3-21 and 10.3-22) are extremely common in all mining operations. They are designed to move the greatest amount of material in the most efficient way, and generally used for both utility and production work. Utility work includes tasks that support a mine's main production fleet, such as dump-site preparation and cleanup, bench preparation, road creation, stockpile work, and reclamation. However, the focus of this section is their use for production

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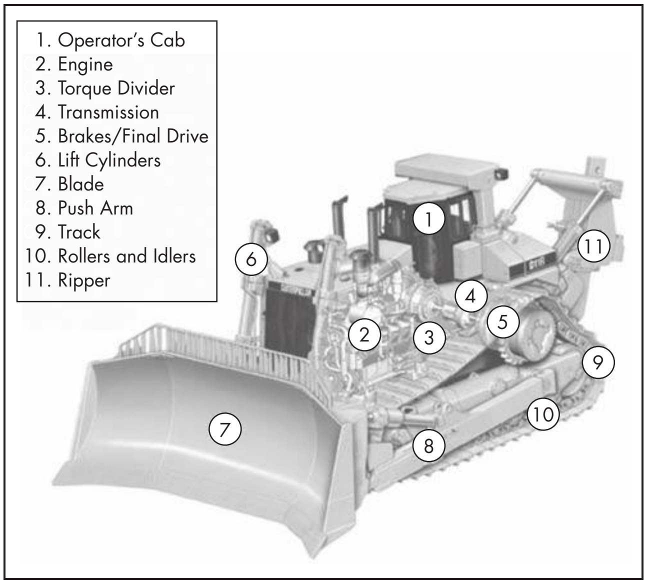


Courtesy of Caterpillar, Inc.
Figure 10.3-21 Track dozer

Courtesy of Eric C. Orlemann.
Figure 10.3-22 Track dozers working in a pit

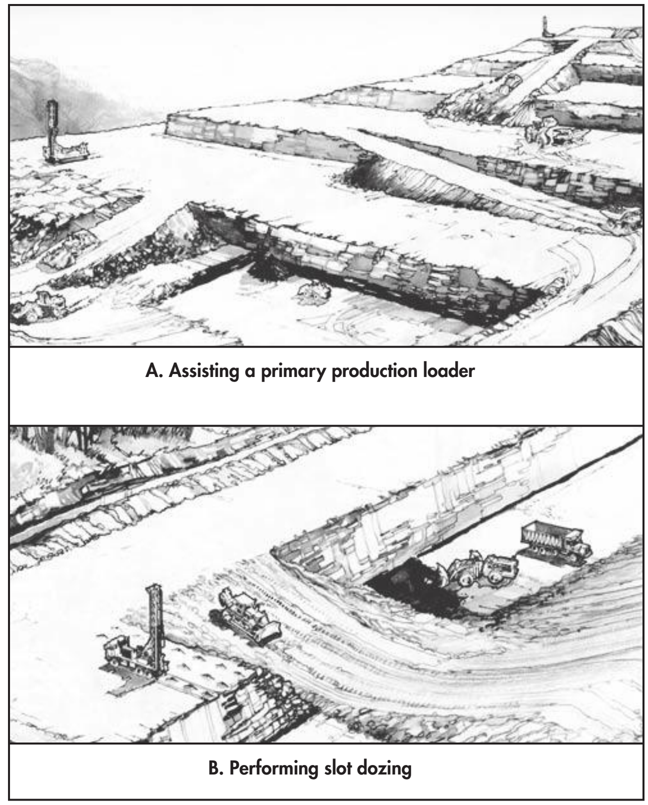
work, specifically mass excavation, for which they excavate, push, and rip in-situ or blasted material from one area to another. Examples of this use include assisting primary production loaders and slot dozing (Figure 10.3-23).

For purposes of comparison, the track dozer is a low-capital-cost, high-operating-cost system that has moderate flexibility and can operate through a limited range of applications with moderate sensitivity to geologic variance.

Currently, two major manufacturers build track dozers for the mining industry: Caterpillar and Komatsu. Table 10.3-3 compares their specifications for typical large track dozers used for mining production.

The specifications are very similar between manufacturers, except for the Komatsu D575 Super Dozer (SD), whose high-end size class is an anomaly. Most of the industry uses the smaller size classes because of their lower operation costs and flexibility.

Track dozers are complex machines, because of their variety of mechanical, electrical, and hydraulic systems all fitted into a compact design that protects against the elements.



A. Assisting a primary production loader

B. Performing slot dozing

Courtesy of Skelly and Loy, Inc.
Figure 10.3-23 Track dozers

Operating conditions are unlike those for other material-excavation equipment that loads statically and advances as the face moves. Rather, slot-dozing and rip-and-push operations to assist large loaders require the dozer to push material at varying distances and gradients in poor ground conditions. Therefore, machine design must be very robust. To this end, the mainframe is rigid and consists of multiple fabrications and castings. All major components and systems are mounted to the frame. The radiator, engine, torque divider, transmission, brakes, and final drives are housed in the frame and body, and in most cases are modular for ease of removal and installation.

The components that typically require extensive forethought for maintenance are those that engage the ground, commonly referred to as ground-engaging tools (GETs). GETs include blades and wear plates, rippers (if fitted), rollers and idlers, and tracks. All dozer manufacturers provide a large variety of options for these components, except for rollers and idlers. The blade and associated wear plates are customized for each application; for instance, Caterpillar provides five types of blade (semiuniversal, universal, reclamation, coal, and carry-dozer). Two types of ripper (single-shank and multiple-shank) are available. And both regular-size and wide track shoes are available.

The goal is to maximize GET life and productivity by matching the machine configuration and options to the site characteristics. Thorough site analysis by the manufacturer is required to evaluate the production cycle and material characteristics. The longer the push distances, potentially the higher the replacement frequency. Material characteristics such as

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Table 10.3-3 Track-dozer specifications by manufacturer and model*

	Caterpillar			Komatsu			
Specification	**D10T**	**D11T**	**D11T CD**	**D375**	**D475**	**D475 SD**	**D575 SD**
Operating weight, kg (lb)	66,451 (146,500)	104,590 (230,580)	113,000 (249,120)	71,640 (157,940)	108,390 (238,960)	113,198 (249,560)	152,597 (336,420)
Flywheel engine, kW (hp)	433 (580)	634 (850)	634 (850)	455 (610)	664 (890)	664 (890)	858 (1150)
Blade, m^3 (yd^3)	18.5–22 (24.2–28.7)	27.2–34.4 (35.5–45)	43.6 (57)	18.5–22 (24.2–28.8)	27.2–34.4 (35.6–45.0)	45 (58.9)	68.8 (90)

*Specifications vary by manufacturer.

Table 10.3-4 Dozer case study of larger loads slower versus smaller loads faster

Parameter	Dozer 1 (larger loads slower)	Dozer 2 (smaller loads faster)
Average doze time, min	0.95	0.83
Average doze speed, km/h (mph)	2.4 (1.5)	2.9 (1.8)
Average return time, min	0.40	0.38
Average return speed, km/h (mph)	5.9 (3.7)	6.1 (3.8)
Average cycle time, min	1.35	1.21
Cycles per hour	44.4	49.6
Average blade load, LCM (LCY)	34.4 (45)	27.5 (36)
Average push distance, m (ft)	40 (130)	40 (130)
Production, LCM (LCY)	1,492 (1,952)	1,418 (1,855)
Productivity difference, %	+5	Base

fragmentation (size), abrasiveness, and cohesiveness must be evaluated to determine a dozer's replacement life.

Dozer Productivity

The first step in determining track-dozer productivity is to calculate how much material a particular dozer can push. This narrows the number of suitable size classes and defines some viable configurations before detailed productivity calculations are made.

The following factors are involved in determining dozer capability:

- **Weight.** The dozer cannot push more than its weight.
- **Coefficient of traction.** This is the percentage of the dozer's weight that can be pushed for a given material before the track shears or slips. For most materials, this value averages 60%. For loose sandy material, it can be as low as 30%. Multiplying the coefficient of traction by the dozer's weight gives the weight that a dozer can push.
- **Material density.** The denser the material, the smaller the volume of material that the dozer can push. Blade capacity and selection are directly related to material density.
- **Carry-force ratio.** This is the energy needed to compensate for friction or drag and push material across itself. It is usually about 10% and is included in calculations of drawbar pull, which provides the corresponding machine speed.
- **Slope.** This is the percent grade downhill or uphill. It is included in calculations of drawbar pull, which provides the corresponding machine speed.
- **Push distance.** This is the same as the return distance, although push and return are traveled at different speeds. Both have a large impact on cycle time.

In production dozing, it is critical to plan dozer size and configuration so as to maximize productivity. A common tagline in the industry is "big load slow," which describes how to manage a machine for maximum productivity. Productivity is defined as work done over a unit of time. The industry continues to debate whether dozer productivity is higher when pushing larger loads slower or smaller loads faster. Table 10.3-4 shows a case study of two identical dozer configurations for a slot-dozing application. Cycle conditions are identical and the average push distance and gradient are approximately the same. The average cycle time is about 10% slower for dozer 1 than for dozer 2, in large part because the blade load is 25% larger. The end result is that productivity is 5% higher for dozer 1, pushing larger loads slower.

Among the factors that can improve the operating efficiency of a dozer are technique and technology. Technique has to do with how an operator approaches a particular job. The following technique tips reflect best practices for a variety of dozer operations related to production dozing and ripping, where the highest productivity gains can be made. In some cases, these adjustments can increase productivity by up to 25%.

- Slot: Front-to-back technique (most efficient)
  - Operator works the cut from front to back.
  - Push distance increases with each pass.
  - Efficiency is optimal due to downhill blade loading.
  - Creates the slot and uses it throughout the cut.
- Slot: Back-to-front technique (less efficient)
  - Operator works the cut from back to front.
  - Push distance decreases with each pass.
  - Efficiency suffers from uphill blade loading.
  - Does not fully use the slot throughout the cut.
- Slot: Back-each-pass technique (less efficient)
  - Operator starts each pass at the back of the cut.
  - Each pass uses the entire length of the cut at a uniform depth.

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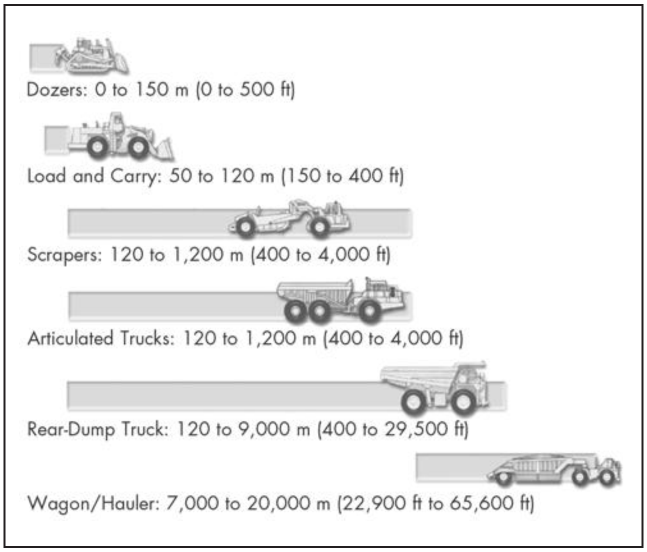

Courtesy of Caterpillar, Inc.

Figure 10.3-24 Haul distances for various types of surface haulers

  - Efficiency and productivity suffer because the machine travels the entire length of the cut in both directions with each pass.
- Berm: Criss-cross removal (most efficient for removing center berms)
  - Operator works the cut from back to front.
  - Push distance decreases with each pass.
  - Existing slots are used to hold in material and increase blade load.
- Berm: Management
  - Berm should not exceed blade height and should be high enough only to trap material for optimum loading.
  - Center berm width should be one-third the blade width.
  - For optimum productivity, the smaller the berm, the easier the disposal.
- Ripping
  - Operator should rip downhill when possible.
  - Operator should reduce speed in shock and impact conditions.
  - Operator should try cross-ripping if material does not free up.
  - Operator should pull the ripper tip forward after penetration.

The final piece to improving dozer efficiency concerns technology. Dozer manufacturers have developed software features that increase safety and efficiency. These features provide automatic control or even real-time data to allow the operator to adjust to conditions. One such feature is Caterpillar's AutoCarry (Caterpillar 2008), which automatically optimizes the blade lift-and-lower functions during the carry segment of push dozing by monitoring, calculating, and integrating data on power-train output, ground speed, track slip, and tractor attitude. Another such feature is an onboard Global Positioning System (GPS) that links directly to survey data in the mine office. The operator obtains needed data via a cab-mounted display rather than by reading surveying maps or looking for grade stakes. GPSs are available from the major manufacturers and from third-party suppliers.

Haulers

The hauling of material (such as coal, ore, sand, gravel, or topsoil) from one point to another in a safe, efficient, and cost-effective manner is critical for mining operations. Selecting the type of surface hauler requires a thorough understanding of the selected mining method and its associated advantages and disadvantages along with the machinery available to the industry.

A number of haulage options are available for mining, all of them with unique characteristics that can be optimal for a particular mine site and haul distance. Figure 10.3-24 compares haul distances for dozers, front-end loaders, wheel-tractor scrapers, articulated dump trucks, off-highway rigid-frame trucks, and belly-dump haulers, each of which has an economic advantage at certain haul distances, with some overlap.

On-Highway Trucks

On-highway trucks are not widely used in mining operations because of their lack of capacity, which is only 6–12 m^3 (8–15 yd^3), and their limited performance capability. However, some mine sites use these trucks for hauling coal (eastern United States), prestripping ttopsoil in contract surface mining operations (Australia), and hauling construction aggregates for road building. Their capability to travel long distances at relatively low cost while meeting local on-highway regulations and their overall flexibility in mining operations where landscape is limited provide a unique hauling alternative.

Three types of on-highway truck are commonly used for mining operations (DumpTrucksGuide.com 2006):

1. **Standard dump truck.** This truck typically has a two- or three- (1 front, 1–2 drive) axle truck chassis with a dump body mounted on the frame. The dump body is hoisted hydraulically by cylinders mounted between the cab and the front wall of the body. The small size of the truck allows for exceptional maneuverability in tight loading areas.
2. **Semitrailer rear-dump truck.** This tractor–trailer combination typically has a three-axle tractor and a two-axle trailer. The trailer body is hoisted hydraulically. Key advantages compared to a standard dump truck are faster unloading and increased payload.
3. **Semitrailer belly-dump truck.** This tractor–trailer combination typically has a three-axle tractor and a two-axle trailer with a C-shaped dump gate. The dump gate, mounted on the trailer, is hoisted hydraulically. A key advantage is the capability to unload material as a wind row.

The advantages of on-highway trucks in typical mining operations are few, but there are applications for which they can provide an adequate hauling alternative. Their small size allows for flexibility in operations where loading areas are small, as for eastern U.S. contour coal-mining operations. Their low fleet cost is advantageous when mining operations need additional hauling capacity for short-term use. They require substantially less investment in up-front capital and subsequent operating cost than do large off-highway mining trucks. They can also be used as a secondary fleet for special

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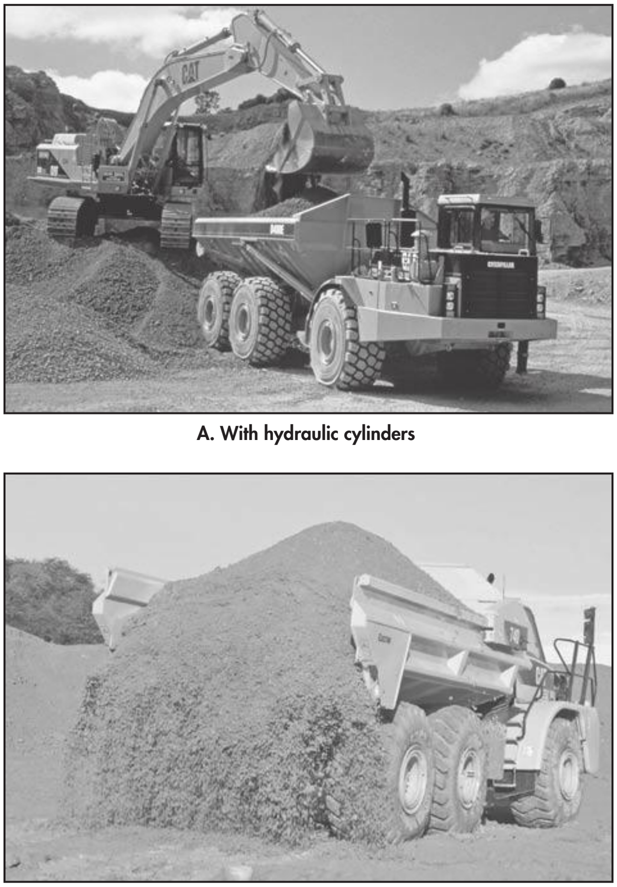

A. With hydraulic cylinders

B. With ejector body

Courtesy of Caterpillar, Inc.

Figure 10.3-25 Off-highway articulated dump trucks

projects such as road construction and prestripping, which are typically contracted from local construction firms.

The disadvantages of on-highway trucks are based primarily on their relative performance compared to that of large off-highway trucks. Payload capacity is an obvious difference; on-highway trucks are smaller and thus have higher overall fleet costs per ton of material due to their lower productivity and the larger number of trucks required. Additionally, they are not designed for use in rigorous mining conditions. Their structural design, electrical and hydraulic systems, brake and steering performance, and power train are designed for highway use, not for 24/7 mining applications. In mining operations, haul road gradients plus rolling resistance can be as high as 20%. On-highway trucks can have difficulty in deteriorating road conditions, to the point where support equipment is needed to recover them, reducing fleet productivity. Thus, they are not considered to be a primary haulage solution for the mining industry, although they will always have limited use in mining applications.

Off-Highway Articulated Trucks

Off-highway articulated trucks are a hauling alternative primarily for middle-to-large-scale construction projects. They are often used for prestripping, road construction, and material hauling for ground preparation for buildings and other infrastructure. When used with small hydraulic excavators or wheel loaders, they can constitute an effective loader and hauler fleet for a mass excavation project. They are widely used for pre- and postmining construction in soft-underfoot conditions, in small loading and dump areas, or on steep (10%–15%) grades.

Table 10.3-5 Articulated dump truck specifications*

Specification	Size Class, t (st) 23 (25)	27 (30)	32 (35)	36 (40)
Gross engine, kW (hp)	224 (300)	261 (350)	298 (400)	335 (450)
Net engine, kW (hp)	212 (285)	250 (335)	287 (385)	324 (435)
Top speed, km/h (mph)	56 (35)	56 (35)	56 (35)	56 (35)
Tire size	23.5R25	23.5R25	26.5R25	29.5R25

*Specifications vary by manufacturer.

An articulated dump truck (ADT) (Figure 10.3-25A) is a three-axle machine with an articulation point between the front axle and the two rear axles. The articulation, which is unique to this truck type, is useful where there is limited area in which to operate. The three axles all provide power to ground. This all-wheel-drive capability provides an advantage over on-highway or off-highway rigid trucks whose one or more rear axles are the only source of power to ground. It also allows the ADT to operate well in soft-underfoot conditions, defined as rolling resistances of 10%–20% (Caterpillar 2000). An ADT can vary power to the wheels according to road and haul conditions. For example, a Caterpillar ADT has three operating modes: a standard mode (40/60 split between front and two rear axles), a low mode (50/50 split), and a high four-wheel-drive mode (all three axles have equivalent power to ground).

The dump body is mounted on the rear frame, with traditionally two options to dump material. The first option is similar to that for on-highway dump trucks: two hydraulic cylinders hoist the body, dumping material rearward. The second option is an ejector body (Figure 10.3-25B) with a hydraulically moveable front wall that runs on a rail system fixed to the side wall; the front wall pushes material back and dumps it rearward. Despite its added design complexity, an ejector body increases productivity by decreasing both dump times and the amount of carryback per load. It also allows the machine to safely dump in steep inclines or side slopes.

A number of major global construction and mining manufacturers provide ADTs for the mining industry, including Caterpillar, John Deere, Komatsu, Terex, and Volvo. In addition, a number of regional manufacturers in China, India, and elsewhere have offerings. ADTs generally range from 23 to 36 t (25 to 40 st) in four size classes that increment every 4.5 t (5 st). Table 10.3-5 highlights their specifications by size class (payload).

ADTs providing the surface mining industry a hauling solution that is adaptable to tough hauling conditions. However, because of their limited hauling capacity, they are not viable as primary production machines.

Off-Highway Rigid-Frame Trucks

The primary hauling machine in mining is the large off-highway rigid-frame truck (Figure 10.3-26). In the 1950s and

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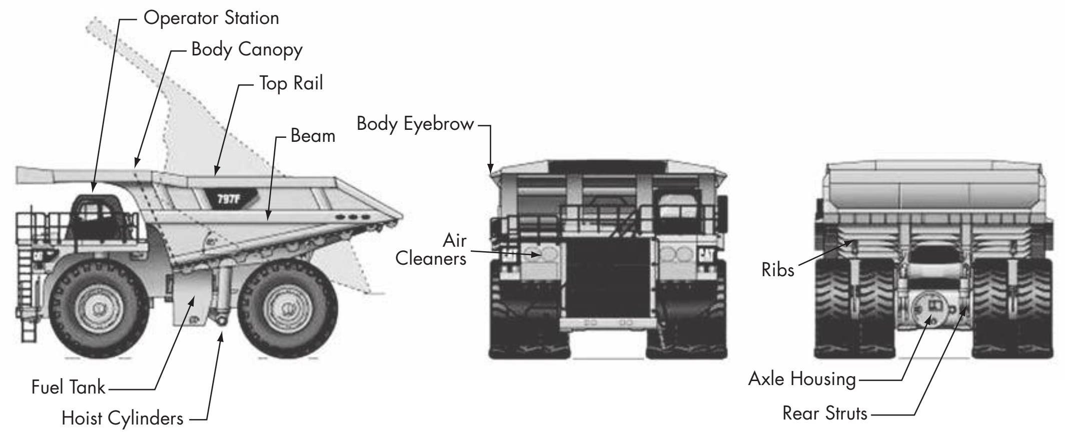

Courtesy of Caterpillar, Inc.
Figure 10.3-26 Off-highway rigid-frame truck

1960s, it was used as an alternative to locomotives and small dump trucks. Over the years, it has proven to be cost-effective, flexible across a variety of applications, and capable of handling the rigors of 24/7 operation. It continues to be a frequent hauling solution of choice for surface mining.

An off-highway rigid-frame truck has a rigid (unarticulated) frame constructed of multiple steel fabrications and castings. To this frame are mounted the truck cab, body, diesel engine, power train, front and rear suspension, front and rear wheels, tires, and more, all interconnected by sophisticated mechanical and electrical hardware and software. The two most distinguishing characteristics of the various makes and models are payload capacity and power-train type.

Payload capacity. The size or payload of an off-highway rigid-frame mining truck plays a significant role in determining the viability of a mining operation. For example, for a given mine plan, one could elect to operate either fifty 90-t (100-st) trucks or twenty-five 180-t (200-st) trucks. At first glance, assuming that machine performance is equal, the second choice, at half the fleet size, would seem to reduce costs dramatically. However, each fleet carries a different capital- and operating-cost footprint, and these footprints should not be assumed to be linear.

Trucks for surface mining currently have payload capacities of 90–360 t (100–400 st). These values have evolved over time, driven by the mining industry's desire to go larger in order to maintain or increase production while decreasing fleet size and operating costs. Within this payload range, there are five distinct classes, designated according to size:

1. 90-t (100-st) class
2. 135-t (150-st) class
3. 180-t (200-st) class
4. 220-t (250-st) class
5. >290-t (320-st) ultra-class truck (UCT)

Figure 10.3-27 shows current mining truck models by size class and manufacturer. Some manufacturers are not included because information was not available.

Product strategies regarding payload capacity clearly vary by manufacturer. Most mining truck manufacturers work closely with the industry to determine the appropriate size for an application, typically matching the rated payload capacity to the current and expected future loading tools, using three to five passes as the optimal level. For example, for a truck with a rated payload capacity of 220 t (250 st), an electric cable shovel with a capacity of 46 m^3 (60 yd^3), assuming a 90% fill factor, could load 1,780 kg/LCM (3,000 lb/LCY) of material in approximately three passes.

As important as payload is to a mining operation, the following points continue to be debated:

- Can we get more payload?
- Are our truck payloads at optimal levels?
- Should we upsize?

Some of these questions can be answered by considering the loaders and associated practices, but in some cases the trucks should be considered as well. Thus, after a decision is made about size class, the next decision should concern the type of truck body. There are now a myriad of truck body designs for any type of mining truck, and the choice of design depends on the material characteristics of the mine, which differ from country to country. Truck bodies today can be customized for each operation so as to maximize payload, reliability, and durability. For the purposes of such customization, a mine operation creates a profile listing the following information:

- Material type
- Material density (lightest), needed to ensure that payload is met, regardless of any fluctuation in material density
- Material fragmentation (size), needed to determine liner-plate thickness and dumping characteristics
- Material abrasiveness, needed to predict wear characteristics for determining the appropriate liner package
- Material cohesiveness, also needed to determine the appropriate liner package
- Loading tools (model and type, percent utilization, bucket size)
- Body/mining truck dimension limitations, needed to determine maintenance facilities, hoppers/crushers, loading tool dump heights, and more

Another point of discussion is payload management. The constant push to increase payload for an existing truck fleet is not a bad thing if trucks consistently perform, on average, at under their rated levels. Because pushing trucks beyond their

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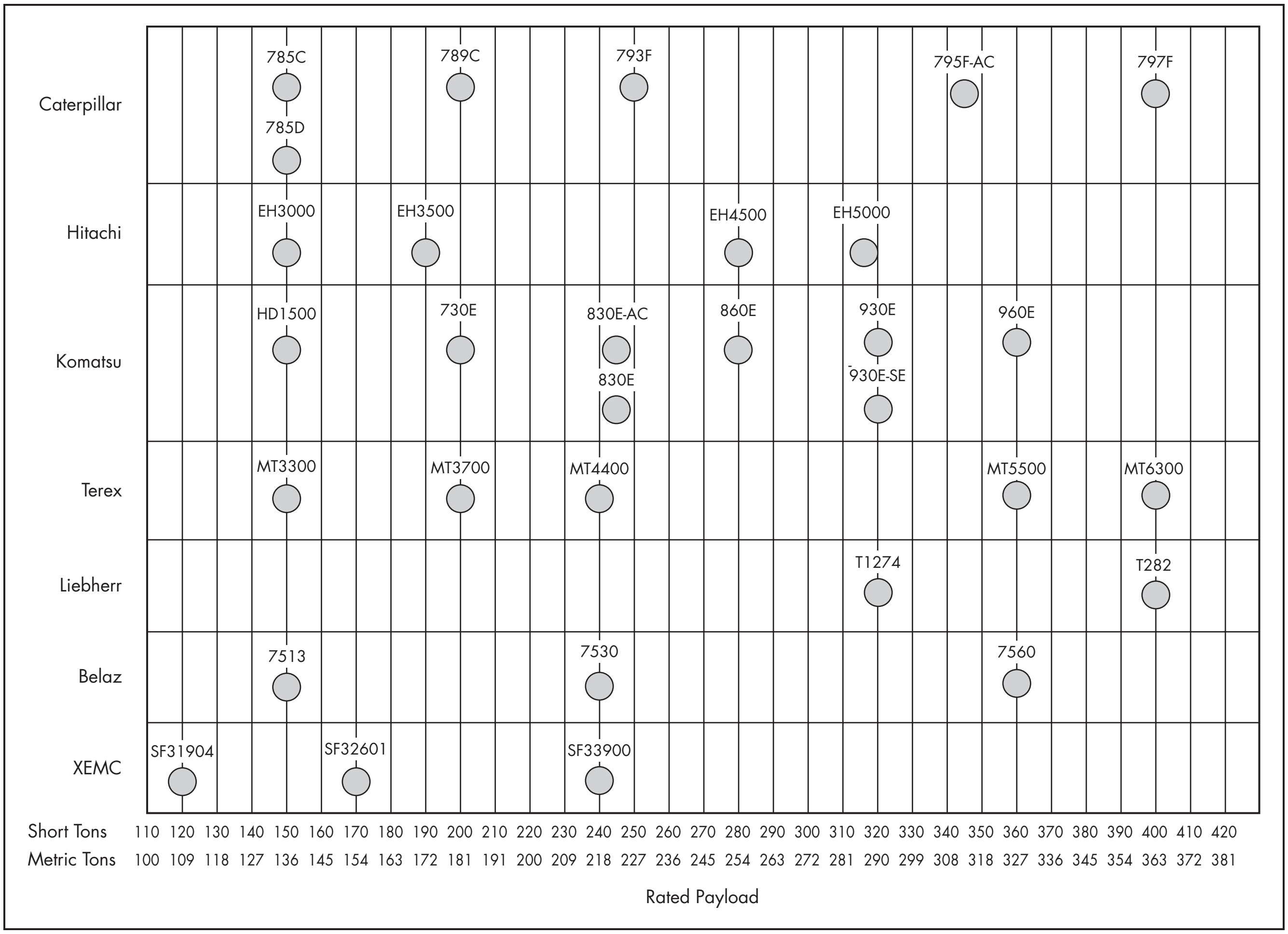

Courtesy of Caterpillar, Inc.

Figure 10.3-27 Off-highway rigid-frame truck models by size class and manufacturer

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rated levels can be detrimental, most manufacturers have a payload policy that outlines the levels at which a truck can perform within the certification or design envelope of the machine. Standard practice today is a so-called 10/10/20 payload policy (Figure 10.3-28) that distributes truck payloads over a set period of time to address risks associated with overloading. The highest risk is overloading beyond 120% of target payload.

All mining trucks are designed to meet certain Society of Automotive Engineers (SAE) and International Organization for Standardization (ISO) design standards that address not only component and system functionality but also safety. Two particular standards are related to payload: ISO 3450 (1996) concerning brake certification and ISO 5010 (1992) concerning steering certification. The requirement for certification is that a truck loaded to its maximum payload (an overloaded state at the upper end of the normal distribution of payloads) should stay within requirements. Overloading beyond the payload policy risks, among other things, machine durability, as the structural components are designed to certain life targets and, if overloaded more frequently than recommended, become fatigued and prone to fail. Early failure increases maintenance and repair costs dramatically due to unplanned rebuilds and repairs. Payload management can be difficult to apply in practice, but it plays a large role in meeting production requirements and keeping a truck fleet operating safely and efficiently. It is important to check with the manufacturer for the tested limit specific to a truck.

Finally, when should a truck fleet be upsized to a new size class? Moving to a higher size class has many benefits and at least a few barriers, which become more daunting with increasing truck size. The following is a list of considerations (Caterpillar 2009):

- Mine operations
  - Mine design: Larger mining trucks can require changes in haul road design and load and dump area, especially when upsizing to UCTs.
  - Loading tools: It is important to have the proper loaders to meet production requirements with the new fleet.
  - Support equipment: Larger mining trucks place a stronger demand for support equipment to maintain haul roads and loading/dump areas.
  - Operator training: Curriculum and training tools must be changed.
- Mine maintenance
  - Facilities: Maintenance facilities (such as shop, lube islands, and parts and component storage) may need to be upgraded to handle the larger machines.

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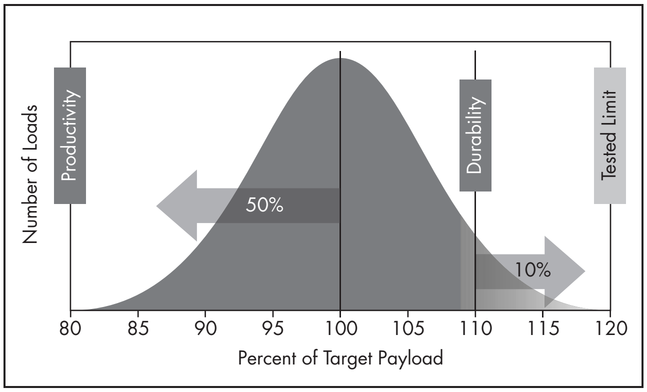


Courtesy of Caterpillar, Inc.
Figure 10.3-28 10/10/20 Payload policy

– Tooling: Tooling requirements must be upgraded to handle larger components and any specialized tools.
– Training: Significant training may be required when upsizing, regardless of system commonality.

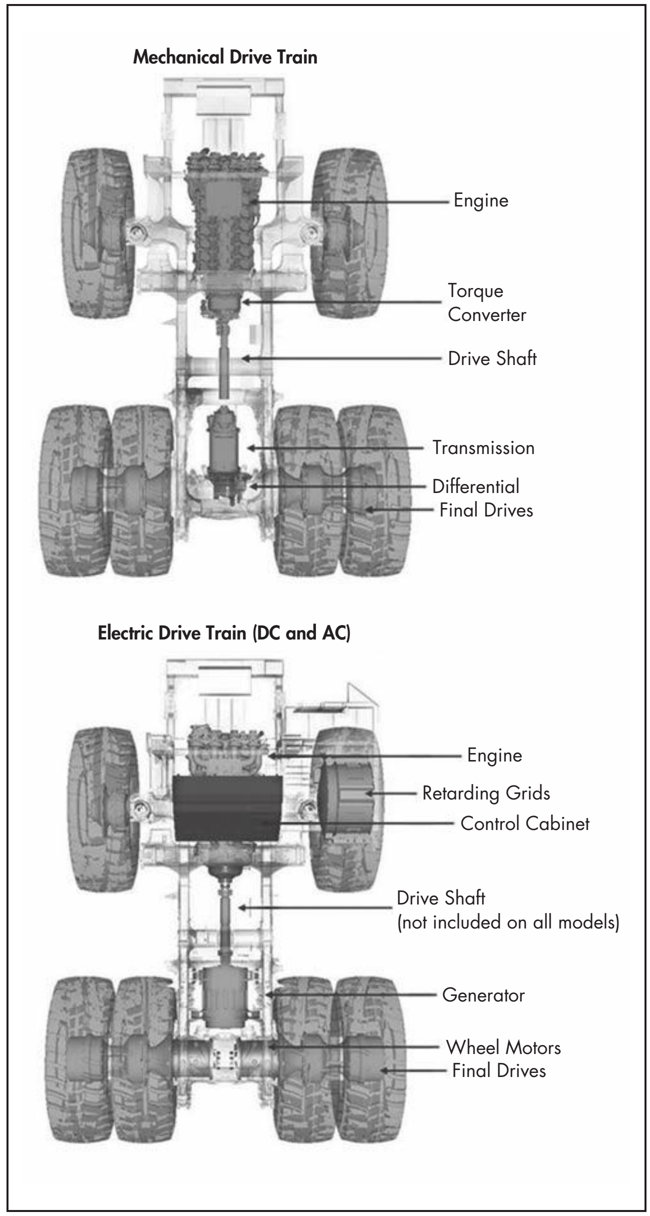
Drive-train type. Truck performance is the next key piece of the puzzle. Three types of drive train are currently in use:

1. Mechanical drive, similar to those used in on-highway automobiles and trucks
2. Direct-current (DC) electric drive
3. Alternating-current (AC) electric drive

Figure 10.3-29 shows a side-by-side comparison of the mechanical and electric drive trains.

The mechanical drive train contains five major components: engine, torque converter, transmission, differential, and planetary gear sets (wheels). The power source is the diesel engine plus torque converter; the latter transmits rotational power from the engine to the main driveshaft. The transmission controls machine torque and speed during operation. The differential transfers output torque to the wheels. Two hydraulic brake packs are mounted on each of the axle shafts on the two rear wheels. The entire system is activated by means of a variety of electronic control modules and hydraulic control systems.

DC and AC electric drive trains contain six major components: engine, generator, power converter, wheel motors, planetary gear sets (wheels), and retarding grid. The power source is the diesel engine plus generator; the latter converts mechanical power from the engine into electric power. AC current from the generator is then converted into useable form. In a DC-drive truck, a rectifier converts it into DC power; in an AC-drive truck, a rectifier converts it into DC power and inverters convert it back to a controllable version of AC power suitable for managing the amperes, volts, and frequencies of the wheel motors in order to create machine speed and torque. The DC or AC wheel motors receive the electric power and feed it mechanically to the planetary gear sets (wheels). The retarding grid—a bank of resistor elements—provides braking force by turning the wheel motors into generators, creating power rather than receiving it. This power is sent through a control cabinet and on to the resistor elements. The resistors impede the flow of the electric power, which causes the wheel motors to slow rotation. Heat generated is cooled by an electric fan.



Courtesy of Caterpillar, Inc.
Figure 10.3-29 Mechanical and electrical drive trains

The three types of drive train have unique performance characteristics that impact productivity and operating costs. The major points to compare are the following, examined in more detail in Table 10.3-6:

- System limitation on grade
- Speed on grade (propelling)
- Speed on grade (retarding)
- Top speed
- Fuel consumption
- Maintenance and repair costs

OTHER SYSTEMS

These additional systems deserve serious consideration:

- Bottom-dump coal hauler
- Trolley-assist mining truck

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Table 10.3-6 Points of comparison for three types of drive train

Point of Comparison		Mechanical	Electric, DC	Electric, AC
System limitation on grade	Define	Stall torque gradeability (STG) is the maximum percent grade that a loaded mining truck can pull out at zero speed. Trucks can stall in load and dump areas with poor underfoot conditions, requiring them to be pulled out. In addition, open-pit mines continue to go deeper. To make doing so economically viable, ramp gradients have increased from 8% to ≥10%. When rolling resistance is added, mining trucks must overcome ≥12% total effective grades, causing unexpected wear on a number of power-train components.		
	Compare	Is best in class. The maximum STG is about 28%–30%. The design accommodates all mining operations that continually operate at gradients up to 15%.	Is the least effective. The standard system has a peak STG of 23% but can be modified to reach 25%. Has thermal limitations during long uphill hauls on gradients of >8%. A thorough site evaluation must be conducted.	Is superior to the DC electric drive system due to the higher power density of the wheel motor. STG is ≤26%, depending on configuration. Has thermal limitations during long uphill hauls on gradients of >10%. A thorough site evaluation must be conducted.
Speed on grade (propelling)*	Define	Performance is dictated by truck weight, engine horsepower, and power train efficiency, and can vary from one model to another depending on a manufacturer's design strategy. However, power train efficiency is a point of differentiation and is the cumulative efficiency of all of the major components. The higher the efficiency, the lower the required engine horsepower. Most surface mines have haul-profile segments at a positive grade; thus the higher the speed, potentially the higher the productivity.		
	Compare	Has the highest gross power train efficiency at its peak, 88%. At an engine horsepower equivalent to the other power trains, has a higher speed-on-grade performance.	Has the lowest gross power train efficiency at its peak, 81%.	Has a gross power train efficiency at its peak of 83%.
Speed on grade (retarding)*	Define	Performance is dictated by truck weight and braking/retarding power. In loaded downhill applications, this is a key performance metric. As long as a truck can travel safely at its optimum level, this point of comparison can optimize productivity.		
	Compare	Is significantly improved since its inception due to addition of another gear.	Has the lowest speed on grade in retarding. However, changes in retarding systems have closed the gap.	Is best in class. The wheel motor power is capable of high levels depending on truck class, and thus has high-speed capability in retarding.
Top speed*	Define	Top speed, both empty and loaded. This is typically a point of differentiation with applications that have long flat hauls. However, except for DC-drive trucks, the systems do not differ significantly.		
	Compare	Depending on truck model, either 61–66 km/h (38–41 mph) or 55 km/h (34 mph).	55 km/h (34 mph)	Is best in class, at 64 km/h (40 mph) regardless of truck-size class.
Fuel consumption	Define	Fuel consumption is one of the most debated performance characteristics of mining trucks. So much depends on the application, engine type and fuel efficiency, condition of the engine and other machine systems, and even how the operator handles the machine. Thus, for a proper evaluation, a site study is required. However, the functionality of a power train is focused on reducing fuel consumption.		
	Compare	Power train efficiency permits engine horstepower to be less than that of an equivalent electric-drive model. Less horsepower equals less fuel consumed for an equivalent speed on grade. Also has a feature that allows the truck to consume zero fuel while retarding (Caterpillar 2006). When the truck approaches a downgrade and the operator's foot is removed from the throttle, gravity takes over. With a mechanical linkage between the wheels and engine, no combustion (and thus no fuel) is needed to maintain engine rotation.	Possesses unique operating characteristics that save fuel. The engine does not cycle during acceleration, as does a truck with a mechanical drive train. Rather, it runs at constant speed, allowing control software to manage speed and torque. Also, the engine actively manages horsepower demand from the engine as it cycles through the engine power curve, matching requirements to operating conditions. Finally, the engine applies a partial-power feature on minimal grades, reducing power draw from the engine to the drive system.	
Maintenance and repair costs	Define	The second most debated performance characteristic is the cost for maintenance and repair, defined as the life-cycle operating costs associated with replacing parts and components, along with labor required for removal, installation, and repair. This characteristic is difficult to compare, as much depends on the severity of the application, maintenance practices, on-site support infrastructure, and more. A fleet-selection process should provide site-specific comparisons of the impact of these costs on the operating cost of a drive system.		

*Refer to the manufacturer's performance handbook for rimpull and retarding curves.

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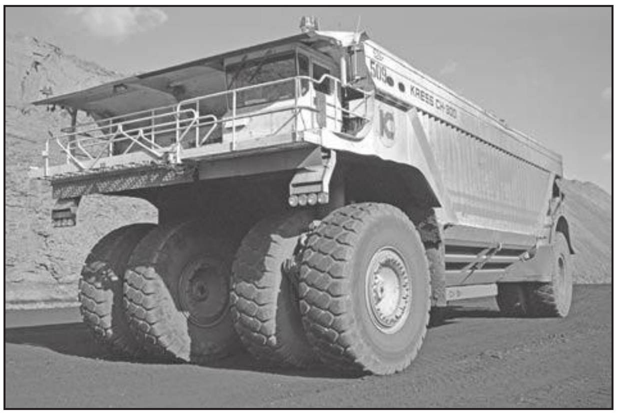
A. Truck–trailer configuration



B. Unibody configuration

Courtesy of Eric C. Orlemann.

Figure 10.3-30 Bottom-dump coal haulers

- Wheel-tractor scraper
- In-pit crushing and conveying system

Bottom-Dump Coal Hauler

A semipopular haulage solution for surface coal is the bottom-dump coal hauler. It has become a staple in thermal coal operations because of the need to haul coal from the pit to a nearby power plant.

The truck–trailer configuration (Figure 10.3-30A) of the bottom-dump coal hauler has a mining truck chassis as the tractor, modified with a hitch assembly to receive a trailer. The most common size class is the 90–136 t (100–150-st) standard truck chassis. The high-volume trailer has typically 1.5 to 1.7 times the payload capacity of the corresponding truck fitted with a rear-dump body. This high capacity is well suited to long hauls with few high-gradient segments, and the bottom-dump coal hauler provides a higher production rate than does a traditional mining truck, thus lowering haulage costs per ton for the truck fleet.

The main players in this market segment for the chassis are Caterpillar (777 and 785 models), Hitachi (CH120, CH135, and CH150 models), and Komatsu (785 and HD1500 models), all of whom provide the necessary chassis modifications from the factory. The trailers are designed and manufactured by smaller specialty firms such as Kador Engineering (Australia), Maxter-Atlas (Canada), and Mega (United States). These firms provide the trailer and hitch assembly, whereas truck OEMs provide the additional axle to be fitted to the trailer, in the interest of product consistency. One firm provides the complete package: Rimpull (United States) provides an entire lineup of tractor–trailer options with their CW160, CW180, and CW200 models.

The truck–trailer configuration of the bottom-dump coal hauler has limited application due to the design specifications of the chassis. A fully loaded rear-dump truck has a continuous rating that allows for total effective gradients of 10%–15%. When the additional weight of a trailer is added and payload is increased, performance drops, limiting the machine to effective gradients of just 5%–10%. This is suitable for a number of surface coal operations, especially where the coal seam is relatively shallow and there is a limited amount of cover or overburden. Another point of limitation is haul distance. The truck–trailer configuration is economically viable only for haul distances of ≥3.2 km (2 mi). For shorter distances, this configuration, with its added costs associated with the trailer, tires, higher fuel consumption due to an increase in cycles, and lower performance characteristics does not compete well against a traditional truck configuration. However, for one-way haul distances of 8–16+ km (5–10+ mi), all of the benefits associated with this configuration outweigh those for the rear-dump truck.

A variant of the truck–trailer version of the bottom-dump coal hauler is the unibody configuration, or unibody coal hauler (Figure 10.3-30B), currently manufactured only by Kress (United States). This machine has a built-in bottom-dump body (i.e., it is unitized). Among its advantages compared to the truck–trailer configuration, it has a significantly higher payload-to-weight ratio, a higher horsepower-to-weight ratio, and a 50-t- (55-st-) lower empty weight. It also has higher fuel efficiency (fuel consumed per ton) and lower metric-ton kilometers per hour (ton miles per hour), which improve tire life. Its higher horsepower-to-weight ratio can enable it to, for example, operate at higher effective gradients to increase productivity. In addition, its drive train is capable of achieving higher top speeds—typically up to 30% higher—than is the drive train for the tractor–trailer configuration, which is limited to the speed capability of the chassis. This can increase productivity measurably on long hauls.

Trolley-Assist Mining Truck

A trolley-assist mining truck (Figure 10.3-31) is a unique application for mining trucks, and strictly exclusive to electric-drive models. Its use for large-scale material transport dates back to the late 1930s in Italy's full-trolley systems; its use for mining began during the energy crisis of the 1980s. With upgrades in technology, it still has relevance in the mining industry. Currently, two suppliers offer a trolley assist on their mining trucks: Hitachi and Komatsu. Four operations use it today, all in Africa.

A trolley-assist mining truck draws its power from overhead power lines that are run on haul segments where the largest benefits can accrue, such as where the loaded truck operates on a positive grade. The truck is fitted with a pantograph that acts as a conduit between the line and the truck's electric-drive distribution system. As the truck approaches the line, the operator lifts the pantograph until it contacts the line. When the two engage, the operator removes his or her foot from the throttle and continues to steer while the truck draws power from the line. Power is fed to the wheel motors, temporarily replacing the diesel engine and generator.

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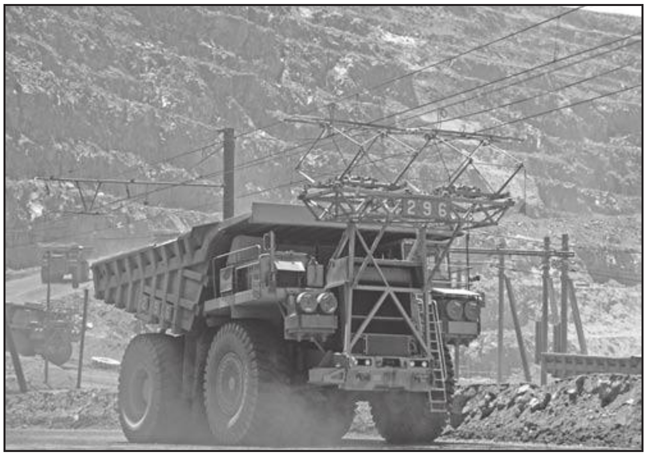

Figure 10.3-31 Trolley-assist mining truck

Trolley assist has several benefits:

- Decreased fuel consumption, achieved by running the engine at idle for the length of the line. Depending on the length of the ramp, fuel savings can be as high as 50%.
- Increased productivity per cycle, achieved by using the excess power capacity in the wheel motors. The power rating of the wheel motor is almost twice the engine gross horsepower, in order to meet the technical requirements for continuous operation under diesel power. That is the primary reason for the significant speed-on-grade performance in retarding, when the wheel motors use their full potential. The same principle applies during trolley assist, when power from the diesel engine is replaced by power from the overhead line. The result is an increase of up to 80% in speed-on-grade performance. Depending on the haul cycle, this can translate into an increase in production of up to 10%.
- Increased diesel engine lifetime. The heaviest toll on an engine in a haul cycle normally occurs when the truck is fully loaded on a grade—the very point at which trolley assist kicks in. The life of a diesel engine is calculated in terms of the total quantity of fuel consumed during the design life. With the engine operating at idle on grade rather than at maximum, the life can be extended significantly, potentially eliminating one complete engine rebuild over the life of the truck.

Trolley assist also involves additional operational costs and constraints, including the following:

- **Relative costs of electric power and diesel fuel.** This is one of the single largest variables to consider when evaluating trolley assist. The cost for diesel fuel can be enormous for a medium to large fleet, but the ultimate question is whether the savings in fuel can offset the cost of electric power.
- **Capital cost of trolley wayside equipment.** This consists of mine power distribution, substations, masts, and wire.
- **Capital cost of truck trolley equipment.** This consists of a pantograph, auxiliary cooling, and truck controls.
- **Mine plan.** Trolley assist does not allow for operational flexibility. After the equipment is in place, it typically is not moved until doing so makes economic sense, often 5 to 10 years from the initial installment. Therefore, it is critical to evaluate the long-term mine plan and determine whether or not a permanent main haul road is possible.
- **Haul profiles.** Determining which haul cycle benefits from trolley assist is one of the most critical pieces to the evaluation. A long haul segment with a grade that the truck travels loaded is the best choice.
- **Capital cost for additional mine-support equipment.** Costs associated with additional motor graders or wheel dozers may need to be included. Haul roads where trolley assist is used must be kept in pristine condition, since spillage and rutting can cause the truck to lose connection with the overhead line.

The industry will continue to support trolley assist. Technology improvements now under consideration include concepts such as auto-control when ascending a grade and regeneration of power when retarding during a return cycle.

Wheel-Tractor Scraper

One of the oldest concepts of bulk material handling is the wheel-tractor scraper (WTS) (Figure 10.3-32), which traces its roots back to horse-drawn slip scrapers in the late 1800s (ASME 1991). Today the WTS is the only machine that can self-load, haul, and dump with a single operator. For purposes of comparison, the WTS is a low-capital-cost, high-operating-cost system that is very flexible and can operate through a limited range of applications with high sensitivity to geologic variance.

Mobility and flexibility are key characteristics of the WTS, which makes it ideal for small, short-life mining projects. Its capability to remove and place material in controlled lifts makes it the machine of choice for topsoil relocation in reclamation operations.

WTSs are available with three types of loading: pan, elevator, and auger. The pan uses the motion of the machine to force material in to the bowl; the elevator and auger have mechanical apparatus that assist the material into the bowl. The pan is slower to load but is better suited for blocky materials. The largest units have a bowl capacity of ≤34 m^3 (44 yd^3) for earth and rock densities, and can be larger for lighter materials such as coal.

WTSs are either single-engine or twin-engine systems that can be pushed, usually by a dozer, to assist with loading. Larger units can also configured in a push–pull arrangement for connecting two scrapers during loading, thus putting the horsepower of two machines on one cutting edge. Because of its cutting mechanics, the WTS best suited to unconsolidated materials.

Production is obviously affected by haulage distance, but for a moderate haul of 450 m (1,500 ft) a large WTS can produce at a rate of about 300 BCM/h (400 BCY/h). It is capable of speeds >50 km/h (30 mph). Optimal one-way haul lengths are 200–1,200 m (400–4,000 ft). The basic haul cycle is similar to that for a truck: load, haul, dump, and return. Because the load and dump components of the cycle can be 50–100 m (150–300 ft) long, it is most efficient to set up the haulage route so that the loaded haul is shorter than the return. The effect of haul distance on production rate is not quite 1:1; for example, doubling a mid-range haul distance decreases the production rate by about 40%. Properly designed and well-maintained roads are as critical to a WTS as to any other hauler, although this fact is often overlooked, possibly because the load and dump areas are relatively rough. A smooth haul

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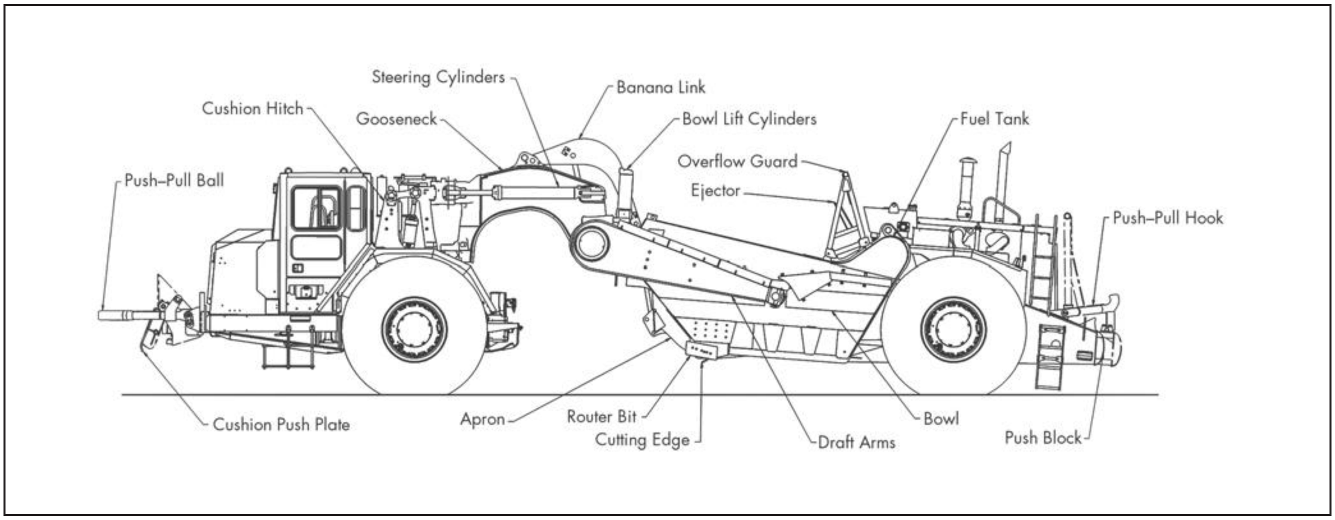
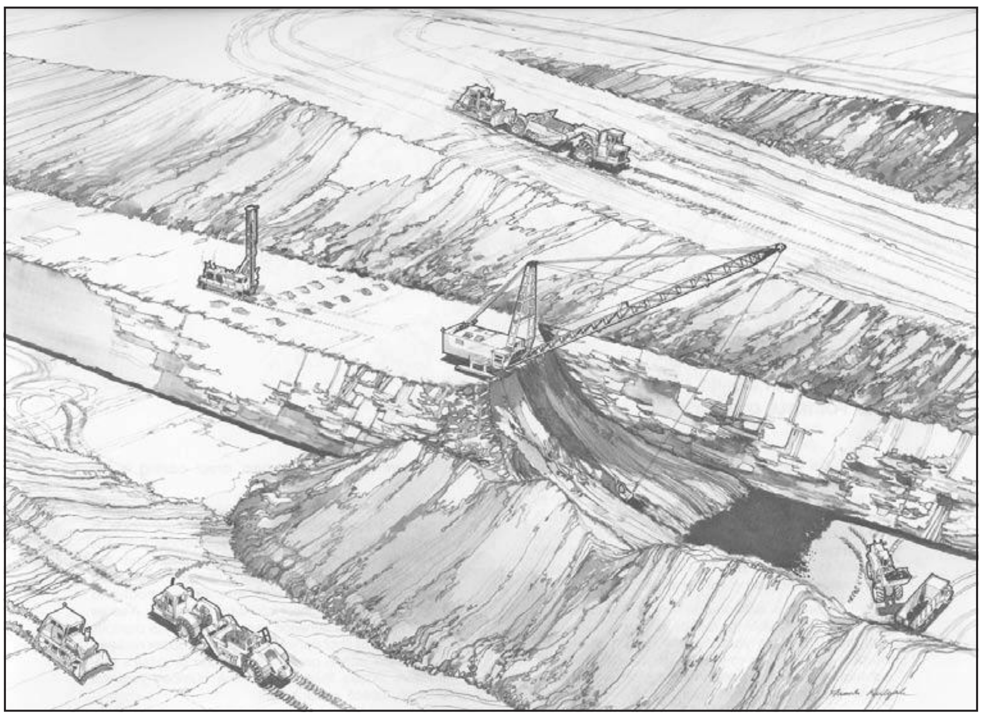


Courtesy of Caterpillar, Inc.
Figure 10.3-32 Wheel-tractor scraper



Courtesy of Skelly and Loy, Inc.
Figure 10.3-33 Wheel-tractor scrapers performing typical operations, including dozer-assisted downhill loading

not only lowers rolling resistance, it also makes for a smooth ride with reduced loping and therefore higher speeds.

Under most conditions, WTS load times are in the range of 30–60 seconds. Production can be enhanced by downhill or assisted loading (Figure 10.3-33). A dozer assist or a push–pull system delivers additional horsepower, increasing the production rate by about 10%. Having a ripper-equipped dozer handy can help to loosen hard-packed material between scraper loads. Loading is accomplished by lowering the bowl until the material flows steadily. Cutting too deep may take longer, resulting in higher fuel consumption and spinning tires with their associated costs.

WTS dump or spread time is about 20–30 seconds and the total fill time is 35–45 seconds (Caterpillar 1998). A significant feature of the WTS is its capability to spread a load in a controlled manner. Laying material down in thin lifts improves compaction on multiple lifts or allows material to be spread to a specified thickness.

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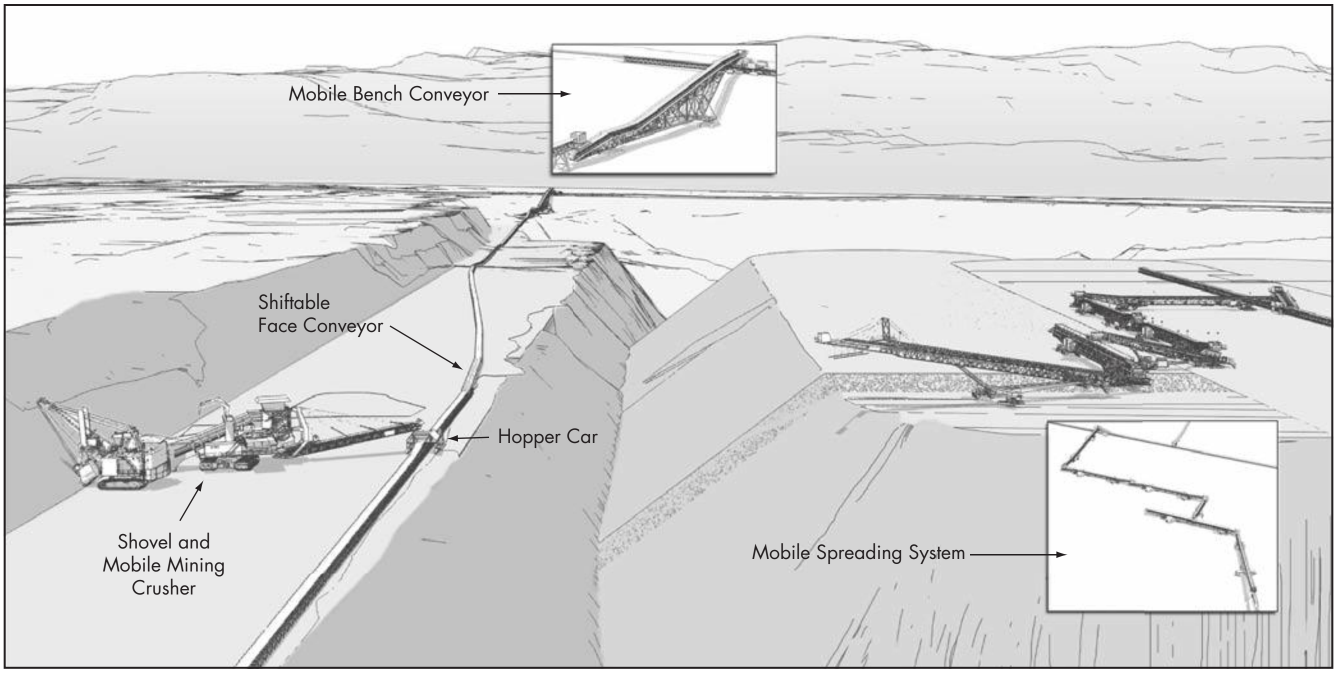

Courtesy of P&H Mining Equipment, Inc.

Figure 10.3-34 In-pit crushing and conveying system

In-Pit Crushing and Conveying System

A relatively recent and increasingly popular entry into large surface mining systems is the in-pit crushing and conveying (IPCC) system (Figure 10.3-34). This system uses a crusher/sizer unit to process material from a cyclic loader to a size that is suitable for conveyor transport, extending the application of around-the-pit conveyor systems to include consolidated waste and overburden.

For purposes of comparison, the IPCC is a high-capital-cost, low-operating-cost system that has limited flexibility and can operate through a limited range of applications with moderate sensitivity to geologic variance.

In-pit or near-pit crushing of ore has always been common, primarily because the location of the crusher station has only a very small impact on the total comminution costs. Crushing of waste, on the other hand, until the advent of this technology, has been difficult to justify on the basis merely of enabling greater use of conveyor transport.

Although IPCCs have been used since the 1950s, the first large-scale mining systems capable of working with feed direct from a mining shovel were implemented only in the 1980s. With the implementation of compact twin-roller sizers, in about 2002, the system finally gained wider acceptance.

The IPCC is a mobile crusher/sizer machine. Historically "mobile" meant relocatable, usually with significant cost, infrastructure, and time requirements. However, the IPCC is self-propelled or easily transportable with no fixed infrastructure requirements and travels with its own loading tool.

For crushing, the IPCC crushes material to a size suitable for feeding to a conveyor belt: in all three dimensions, the material can be no more than about 30% of the width of the belt, and a significant percentage of the material must be smaller than that, in order to cushion the belt and prevent damage to it. Although the IPCC's crushers and sizers can handle materials with extreme rock strengths, to date, the IPCC (for waste) is most cost-effective for use with materials whose compressive strengths are 50–90 MPa (7,000–13,000 psi) and somewhat less cost-effective for harder or softer materials.

For conveying, the IPCC relies on a series of components that feed material from one to the next. A cyclic loading tool such as mining shovel feeds into a mobile sizer that follows along with the shovel. The sizer feeds into a face conveyor, much as for a BWE. The face conveyor feeds into a series of other conveyors, eventually leading to a discharge conveyor at the dump.

Because of the multiple conveyors, the IPCC is suitable for pit geometries that favor use of draglines and BWEs (i.e., those with long linear faces). Because its loading tool is a shovel, it can handle materials with rock strengths greater than a BWE can handle. At the time of this writing, however, IPCC systems are still finding their niche application but show favor for waste transportation in linear pits with geometries that require long hauls with relatively limited elevation change.

The digging face of the IPCC has a large footprint, similar to the case for the BWE, which needs to be considered when planning other pit operations such as blasting or pit access. Conversely, the conveyor route has a relatively small footprint, which can reduce costs for ramps, earthen bridges, and the like. Likewise, the dump area can be configured with some flexibility and material can be placed in final form with very little rework required for rehabilitation.

The IPCC has an electric drive system. Although this type of system requires electrical infrastructure in the pit, it does not depend on diesel fuel, whose costs tend to fluctuate independent of the product market. It also does not have tires; certainly the tire shortage of 2006–2008 had a significant influence on a number of IPCC purchases.

The IPCC generally requires minimal operating labor per unit of production. Operational and maintenance labor requirements are cyclic, with significantly higher demand during

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conveyor relocation. These cycle demands are relatively easy to manage where maintenance contractors are available. The operating method lends itself to semiautonomous operation of some components, further decreasing labor demand.

The newest IPCCs have been matched to the large mining shovels with peak capacities of about 10,000 t/h (11,000 st/h). Because the system components are arranged in series, overall efficiencies are similar to those for a BWE system, about 55%–60%, resulting in an average production rate of about 6,000 t/h (6,600 st/h).

Typical IPCC operations are (or in the second case will be) viewable by satellite-image software in the vicinity of these coordinates:

- IPCC prestripping in advance of dragline: 21° 43′ 33″ S 147° 59′ 23″ E
- IPCC open-pit application with truck and shovel: 22° 43′ 23″ S 147° 38′ 08″ E

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Information in the "Bottom-Dump Coal Haulers" and "Trolley-Assist Mining Truck" sections is taken from Moore 2007 and Hutnyak Consulting 2004, respectively.

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