CHAPTER 4.8

Cost Estimating for Underground Mines

Scott A. Stebbins

INTRODUCTION

Estimating the costs of mining is often referred to as an art. Unfortunately, this definition turns many would-be evaluators away because of this understandable misconception. Cost estimating, as with any predictive process, requires an evaluator to envision and quantify future events—in other words it requires one to be creative. A better description is that estimating the costs of mining is a creative endeavor. Fortunately in mining, most of the values that an evaluator must predict either stem from measurable entities, such as the configuration of a deposit, or from well-understood and accepted engineering relationships. In actuality, mine cost estimating is a process of matching values obtained through simple engineering calculations with cost data, a process made easier in recent years thanks to readily available printed and electronic information databases.

Mine cost estimating is also referred to as an art because no widely accepted rigorous approach to the process exists. Unlike the process of estimating costs in the building construction industry, in mining, the process varies noticeably from one evaluation to the next, not only in approach but also in scope.

A complete mine cost estimate cannot be fully detailed in the few pages available here. The information presented in this chapter is primarily aimed at minimizing the intimidation felt by many geologists and engineers when they undertake a cost estimate. The basic premise is that anything can be estimated. And the approach detailed here is one in which more or less complete listings of labor, supply, and equipment requirements are based on information about the deposit and the proposed mine. These listings are then used in conjunction with documented salaries, wages, supply costs, and equipment prices to produce estimates of mine capital and operating expenditures. This method, most often referred to as an abbreviated itemized approach, is much easier than it might initially appear. Although there are several other methods available, including parametric equations, factoring, cost models, and scaling, itemized estimates have the advantage of providing thorough documentation of all of the assumptions and calculations on which the estimated costs are based. As a consequence, the results are much easier to evaluate and adjust, and for this reason, they are more useful. Because they rely on much of the same information required to do a proper job using any of the other methods, evaluators are often surprised to find that engineering-based, itemized estimates can be accomplished with some expedience.

Early in any mine cost estimate, long before the evaluator begins to worry about the cost of a scoop tram, the scope of the evaluation must be determined. To accomplish this, the purpose of the estimate must first be defined. If it will be used to select which one of several deposits should be retained for future exploration expenditures, then the estimate will be less thorough than one used to determine the economic feasibility of a proposed mine or one used to obtain funding for development. Coincidently, the level of information available with regard to deposit specifics also plays a part in determining the scope of the estimate. As the level of information increases, so do the scope of the estimate and the reliability of the results.

Accuracy is a measure of predicted (or measured) value versus actual value. It cannot really be quantified until well after the project is under way and the estimated costs can be compared with the actual expenditures. So, cost estimators instead work more in terms of reliability, which is a measure of the confidence in estimated costs. Reliability is determined by the level of effort involved in the evaluation and by the extent of the available deposit information. Simply, the more information that is available (specifically geologic and engineering information), the greater the reliability of the estimated costs. If an evaluator has a firm grasp on the deposit specifics and works diligently to estimate all the costs associated with development and production, then a highly reliable estimate should result.

Estimators determining the potential economic success of developing a mineral deposit must undertake an iterative process of design and evaluation. After settling on an initial target production rate, the process can be broken down into the following four steps:

1. Design the underground workings to the extent necessary for cost estimating.
2. Calculate equipment, labor, and supply cost parameters associated with both preproduction development and daily operations.

Scott A. Stebbins, President, Aventurine Mine Cost Engineering, Spokane, Washington, USA

CHAPTER 4.9

Cost Estimating for Surface Mines

Scott A. Stebbins and Jennifer B. Leinart

INTRODUCTION

It is obvious that costs vary from one mine to the next. So, although it is of interest to know the costs associated with surface mines in general terms, it is also important to understand how to estimate the costs of a proposed operation in a way that considers the unique development and operational parameters, and subsequently costs, of each deposit. Although focusing primarily on how to estimate costs, this chapter also includes general operating expenses for typical surface mine configurations.

There are probably as many ways to estimate mining costs as there are cost estimators. Because of the lack of a standardized approach, evaluators are left to estimate costs as best they can, so almost everyone uses a slightly different method. A standardized method that suits every situation would be extremely difficult to develop, given that each proposed mine is unique and conditions can be so variable. Although no such approach exists, many well-documented methods are available. For example, there are the tried-and-true, broad-brush approaches, one of which is the parametric method, where costs are derived from general algorithms (or curves) of the following form:

$$\text{cost} = x(\text{parameter})^y$$

The parameter in these algorithms can be almost anything, but most often it is the production rate. The x and y values are derived through statistical evaluations of known or estimated cost data. The U.S. Bureau of Mines Cost Estimating System, also known as CES (USBM 1987), can be considered a parametric approach, as can methods developed by O'Hara (1980) and Mular (1982).

Another example of a broad-brush method is the factored approach. Usually with this technique, one primary cost (such as the cost of the purchased equipment) is subjected to a series of factors to estimate all the other pertinent costs of the project (Vilbrandt and Dryden 1959). This method has fallen out of general use because it is, in light of subsequent approaches, considered too general.

Evaluators also commonly rely on a comparative approach. With this method, estimators examine costs at similar projects and make adjustments, often through the use of scaling factors (Schumacher and Stebbins 1995), to account for differences in operating parameters. This may be the most comforting of the broad-brush approaches, but it can also be the most misleading. Conditions simply vary too much from one project to the next to rely too heavily on comparative costs. If conditions were the same at every deposit, then assigning costs from a past or similar project would be acceptable, and the approach would be widely used. But it is the differences in the operating parameters from one project to the next that dictate the differences in costs, so these must be fully considered.

Cost models are a form of the comparative approach. These consist of a compilation of cost estimates along with the parameters on which those estimates are based. Evaluators find the example from within the compilation that most closely resembles their project, and they then use the costs associated with the example as an indication of the costs at their project. Example cost models for typical surface mine configurations can be found in Appendix 4.9A.

Significant effort went into the derivation of the specific variations of the aforementioned methods, and each represents an invaluable source of useful, reliable information. In particular, the CES curves (USBM 1987) enable evaluators to estimate costs for a multitude of mining and mineral processing activities for which no other source exists. But, arguably, the concern with each of these approaches is the lack of transparent detail. Evaluators are left to wonder if results truly represent their project. Even though broad-brush methods are often used because much of the information needed for more detailed analyses is difficult to obtain, evaluators still continually strive for more verifiable, and hence reliable, results.

In the past, the broad-brush approaches also maintained their popularity in part because more detailed analyses were time-consuming. Over the past 20 years, however, things have changed. Most evaluators now use a more detailed, engineering-based approach to estimating costs at almost every stage of

Scott A. Stebbins, President, Aventurine Mine Cost Engineering, Spokane, Washington, USA
Jennifer B. Leinart, CostMine Division Manager, InfoMine USA, Spokane Valley, Washington, USA

project evaluation. Two events have led to this eventuality: The first was the development, publication, and distribution of *Mining Cost Service* (InfoMine USA 2009b), along with an increase in the availability of information similar to that contained in *Mining Cost Service* through the Internet. This annually updated document is a comprehensive compilation of current mine and mineral processing cost information. The second event was an improvement in spreadsheet and application-based calculation modeling capabilities, which enabled evaluators to handle the significant increase in the amount of work associated with engineering-based estimates in a timely manner. Evaluators now conduct engineering-based estimates in time frames previously achievable only when they used the broad-brush approaches.

ENGINEERING-BASED, ITEMIZED COST ESTIMATING

The method detailed in the next few paragraphs is best described as an engineering-based, abbreviated, itemized approach. It consists of three major steps, along with a highly variable number of minor steps.

In the first step, estimators design a mine to the maximum extent possible given the available information. For a deposit that can be mined using surface techniques, even a general pit outline, an overall depth, and a delineation of the routes to the processing plant and the waste stockpiles provide a great deal of information pertinent to the cost estimate.

In the next step, evaluators estimate or calculate all the parameters associated with the things that cost money: the workers, the equipment fleet, and the consumable supplies. This step is where an estimator expends the most effort, although the first design step previously outlined is the most important in achieving reliable results.

The final step is the simplest, thanks to publications such as *Mining Cost Service*. Evaluators need only apply known unit costs for labor, equipment operation, and supplies to the projected and calculated development and operating parameters to arrive at estimates of the operating costs (in addition to estimates of many of the preproduction development costs). They then need to apply equipment purchase prices along with the costs of some common mine facilities to the previously determined parameters to arrive at the primary components of a capital cost estimate.

The advantages of the engineering-based, itemized approach are many. It can be applied at almost any stage of a project evaluation, from the initial phases when information is scarce to the final stages when almost all pertinent resource and project characteristics have been established. It is reliable in that it concerns itself almost exclusively with parameters specific to one deposit. It lends itself well to computerization because so much of the work involves simple calculations (albeit a lot of them) that are easily encoded on a spreadsheet or a Windows-based application. It is easily adjusted and updated as more information becomes available. As such, the reliability of the estimate increases as the information base expands. And when the evaluation stage is complete, the final computerized product is (in essence) a dynamic cost model that engineers can use to examine operational alternatives throughout the life of the mine.

Traditionally, and logically, evaluators have kept the level of detail in their cost estimates comparable with the amount of information available for the deposit. Unfortunately, it is sometimes tempting to reduce the level of detail in an effort to reduce the amount of time spent on the estimate. Ignoring detail by procedures such as averaging site parameters or combining cost components can reduce the representativeness of the estimate. For instance, if haul distances and gradients for individual haul segments can be gleaned from maps and plans, the cycle time associated with the haul may be significantly different than the cycle time for a more convenient but less reliable overall distance and average gradient (over the entire distance). Example 4 (presented later in this chapter) helps to illustrate this point.

Just as significantly, combined cost values, such as those presented for equipment operation in various publications, can also lead to estimates that are not fully representative. If such costs are broken down into individual components, (i.e., fuel, lubricants, repair parts, tires, and wear parts), then each component can be adjusted individually to suit conditions. For instance, in a situation where a mobile loader is used to collect extremely abrasive rock, the evaluator might adjust the tire and wear part consumption rates upward. If these components were not treated separately, the evaluator might simply adjust the entire composited operating cost upward. The significance of avoiding such an approach is this: If you increase the tire consumption rate by 100% (i.e., multiply the tire operating cost by 2) and the tire cost is initially 10% of the overall operating cost, then the impact on the overall machine operating cost is minimal, as would be any error in the evaluator's assumption of the increase. And, because equipment operating costs may only represent 25% of the overall operating cost, the impact of any error would be even less. In essence, a 100% error in a cost component that comprises only 2.5% of the overall cost is much less significant than a similar error in a cost component that comprises 25% of the overall cost.

GETTING STARTED

Often, where to start is the question. It is sometimes a difficult question to answer when an estimator is trying to figure out how much a deposit will cost to mine. However, when evaluators begin the process of approximating the costs of a mining project, they soon notice a synergy. As one parameter is determined, the value of another is often defined. For instance, as the number of trucks needed to haul the ore is determined, the number of drivers required to operate the trucks and the number of mechanics needed to maintain them are also determined. An evaluator can then use those values to begin the process of estimating the sizes of the shop, the parking lot, the living quarters (if needed), and the workers' changehouse.

A few things must always be known to estimate the costs of a surface mine. The first four things to look for are a target production rate, a stripping ratio, the ore and waste haul profiles, and an estimated powder factor. Just those four items provide a good start.

The target production rate is most often based primarily on the overall size of the resource, although the assets of the operator play a secondary role. In very general terms (from a strictly economic perspective), the more revenues that the project generates early in its life, the better. To that end, operators in a perfect world prefer to maximize the production rate and initiate operations as soon as possible. On the other hand, operators often rely on revenues from the project to fund further development and expansion and thereby minimize early expenditures and the associated economic risks. For the purposes of early-stage feasibility analyses, estimators often aim for somewhere in the middle. A variation of Taylor's rule (Hoskins 1977) provides a reasonable value and is expressed as follows:

$$\text{capacity, t/yr} = (\text{metric tons resource}^{0.75}) \div 70$$

In comparison to production rates at active mines, results from this equation (which was in use as far back as the 1970s) are now conservative. Operators now try to drive economic conditions in their favor by taking advantage of economies of scale. Doing so also shortens the duration of their projects. As an added economic benefit, revenues (and hopefully profits) are maximized as early as possible in the project. A current and more representative equation that works in a manner similar to the variation of Taylor's rule just mentioned is as follows:

capacity, t/yr = (metric tons resource$^{0.69}$) ÷ 20.12

For the other items, a sketch or two can be of great help. A plan view of a proposed pit, the surrounding terrain, and the location of the mill and waste dump sites along with a few cross sections through the pit and along the main haul routes furnish the information needed to roughly estimate the stripping ratio and to define the haul-route profiles.

DRILL AND BLAST

To estimate the costs of drilling and blasting, engineers can glean a great deal of information from just a powder factor. Such a factor (which is most often reported in terms of kilograms of explosive per metric ton blasted) of course differs from one project to the next and is typically determined through experimentation, observation, and adjustment over time at an active operation. Consequently, the value will not be known ahead of time. But reported powder factor values are plentiful in books such as this handbook, in case studies contained in periodicals, and in publications such as the *Mining Source Book* (Scales 2009).

A powder factor from a mine in rock similar to that of a proposed project should supply an initial value that is within reason. From this one value, engineers can of course estimate the cost of explosives in terms of dollars per metric ton of ore. But in addition, they can also estimate how much (in terms of meters) to drill each day, which in turn provides the number of blastholes that must be drilled each day, and that value in turn furnishes the number of caps and boosters consumed each day.

With the daily drilling requirements in hand, estimators can approximate values for daily drill use (in terms of hours per day), drill bit and steel consumption, and (with all this previous information) they can proceed to gauge the required number of drillers and blasters. All of this is a lot to derive from just a powder factor and it is important to remember that for an early-stage cost estimate, precise values are not necessary (nor can they be expected). Reasonable, representative values are required, but highly precise values are simply not obtainable at the early stages of a cost estimate unless the information needed for such precision exists.

To illustrate the process, Example 1 works from a powder factor to estimate consumption rates (and subsequently costs) for explosives, caps, detonation cord, and drill bits and steel. From there, estimated values are further used to suggest drill use (in terms of hours per day) as well as labor requirements.

Example 1

Consider the case where the following have been determined:

- Production rate = 5,000 t/d
- Stripping ratio = 2.5:1 t waste to t ore
- Ore powder factor = 0.305 kg/t ore
- Waste powder factor = 0.331 kg/t waste
- Explosive (ANFO) specific gravity = 0.80
- Hole diameter = 15.24 cm
- Bench height = 12.20 m
- Subdrilling = 1.43 m
- Stemming = 4.27 m
- Drill bit penetration rate = 1.10 m/min
- Drill bit consumption = 2,500 m/bit
- Worker efficiency = 83%
- Drill relocation and setup = 2 min/hole

Based on this information, the following can be calculated.

1. Explosive consumption:

   Ore: 5,000 t/day × 0.305 kg/t ore = 1,525 kg/d

   Waste: 5,000 t ore/d × 2.5 t waste/t ore
   × 0.331 kg/t waste = 4,138 kg/d

   total = 1,525 kg/d (ore) + 4,138 kg/d (waste)
   = 5,663 kg/d

2. Daily drill-hole volume:

   5,663 kg/d × (0.80 × 1,000 kg/m^3)) = 7.08 m^3/d

   unit volume of blasthole = [π × (15.24 cm
   × 100 cm/m)^2] × 4
   = 0.01824 m^3 per meter of depth

3. Daily drilling requirements:

   total drilling (explosives only) = 7.08 cm^3/d
   ÷ 0.01824 m^3/m drilled = 388 m/d

   hole loading factor = ((12.20 m + 1.43 m) – 4.27 m)
   ÷ (12.20 m + 1.43 m) = 0.687

   total drilling requirement = 388 m ÷ 0.687
   = 565 m/d

   holes drilled each day = 565 m ÷ (12.20 m
   + 1.43 m) ≈ 42 holes

4. Drill use:

   daily drill use = (565 m ÷ 1.1 m/min) ÷ 60 min/h
   ≈ 8.56 h/d

5. Worker requirements for drilling:

   daily drilling = 8.56 h/d + ((2 min × 42 holes)
   ÷ 60 min/h) = 9.96 h/d

   worker requirement = 9.96 h/d
   ÷ 0.83 (worker efficiency) = 12.0 h/d

6. Worker requirements for blasting:

   blasthole loading = (4 min/hole × 42 holes)
   ÷ 60 min/h = 2.80 h/d

This example points out the one real difficulty of engineering-based itemized cost estimates—values for several parameters are not always readily apparent or available. Powder factors, drill penetration rates, drill bit consumption rates, and bench heights have yet to be determined in the early stages of project evaluation, and subdrilling depths and stemming requirements have yet to be calculated. In the engineering-based itemized approach, evaluators determine most such parameters using one of two processes: statistical analysis of reported data or calculations based on established engineering relationships.

In Example 1, various mine operating information sources are searched to compile a series of data points. As mentioned earlier, these sources include case studies printed in periodicals, data in publications such as the *Mining Source Book* (Scales 2009), and compilations in handbooks such as this. For example, if engineers need to estimate a powder factor, they might first collect and compile as many reported values as possible. At the same time, they would identify a parameter related to the powder factor and gather associated, representative data points. Powder factors are often listed along with a rock type, for instance, and rock types can be roughly related to compressive strengths (discussed in other chapters of this handbook). With these two data strings (powder factors and rock compressive strengths), evaluators can develop an empirical relationship that they can then use to estimate a powder factor based on the rock type (as represented by an estimate of the compressive strength of the rock). Typically, such a relationship must be derived through geometric regression of the data. One such compilation (based on information primarily from the *Mining Source Book* and this handbook) provided the following algorithm:

$$\text{powder factor, kg/t} = 0.0240 \times (\text{compressive strength, MPa})^{0.4935}$$

Geometric regression analysis is outside the scope of this discussion but is detailed in most statistical analysis textbooks.

For a specific deposit, values for the compressive strength of the rock become clearer as more information becomes available and actual testing begins, but the value gained through the regression should provide a reasonable basis for early estimates. And, while for illustrative purposes in this chapter, the powder factor is related to rock strength, it may be more closely tied to other factors such as the production rate, explosive type, or rock quality. When a relationship such as this has been derived, it can be used to estimate values at other properties.

Evaluators also rely on standard, established engineering relationships to determine some of the project cost parameters. In Example 1, for instance, if the bench height is 15 m, an engineer can rely on well-established blasting design algorithms (Olofsson 1997) to determine subdrilling and stemming requirements as shown in Example 2.

Example 2

1. Maximum blasting burden:

   $$\text{maximum burden} = 1.36 \times (L_b^{0.5}) \times R_1$$

   where

   L_b = charge concentration ~ 14.5 kg/m (Olofsson 1997)

   R_1 = correction for vertical drilling ~ 0.95 (Olofsson 1997)

   $$\text{maximum burden} = 1.36 \times (14.5^{0.5}) \times 0.95 = 4.92 \text{ m}$$

2. Subdrilling:

   $$\text{subdrilling} = 0.3 \times 4.92 \text{ m (maximum burden)} = 1.48 \text{ m}$$

3. Error in drilling:

   $$\text{error in drilling} = [152 \text{ mm (blasthole diameter)} \div 1{,}000] + (0.03 \times (15 \text{ m} + 1.48 \text{ m})) = 0.65 \text{ m}$$

4. Adjusted burden:

   $$\text{burden} = 4.92 \text{ m (maximum burden)} - 0.65 \text{ m} = 4.27 \text{ m}$$

5. Stemming:

   $$\text{stemming} = 4.27 \text{ m (equivalent to burden)}$$

Although Example 2 relies on algorithms from Olofsson (1997), there are several such sources for drilling and blasting engineering calculations, and each may be more or less detailed in its approach. It is only important that, in the early stages of an evaluation, estimators arrive at reasonable, reliable numbers.

As is evident, both of these values (stemming and subdrilling depths) are needed to provide a reasonable estimate of daily drilling requirements (in terms of meters drilled). When an evaluator analyzes these values in conjunction with the bench height, the result is the number of holes that must be drilled each day and, subsequently, the number of caps and boosters consumed on a daily basis.

EXCAVATE AND HAUL

Estimators find that most of the expense of any surface mine is attributable to excavating the rock, loading it into some sort of conveyance, hauling it somewhere (either a mineral processing plant or a stockpile), and then dumping it. Consequently, a representative estimate hinges on the reliability of the excavating and hauling costs.

As with the cost estimates of all the other surface-mining tasks, the basis for the costs of excavating and hauling begins with the design. It is crucial to know the routes over which the ore and waste will be hauled. The more that is known about these routes, the more reliable the estimates will be. Distances and gradients are the key components. And while average gradients over total haul distances can be used, much more reliable results are achieved if the routes are split into segments at each significant change in gradient. The importance of carefully defining the distances and gradients of each segment increases with the stripping ratio. Evaluators find that large projects with high stripping ratios can become, in essence, waste bound, in that the space needed to stack and store waste is at a premium. At such deposits, operating costs are more sensitive to waste haul distances and gradients than to any other factor.

To estimate excavating and hauling costs, evaluators must first determine cycle times for both the excavators and the haul trucks. Evaluators use these cycle times in conjunction with respective machine capacities to gauge the size of the required fleet and to eventually estimate operating costs and purchase prices. If the purpose of an evaluation is to estimate the average costs of production for the project, then the haul profiles should be defined at a point halfway through production. In other words, they should be based on the pit profile at that point in time when about half the resource has been extracted. When engineers structure the cost-estimating process on a spreadsheet or through a Windows application (or any number of other computerized approaches), it is entirely possible for them to estimate the costs associated with haul profiles from any bench, in fact from any point on any bench, in the pit. This is, of course, pertinent when an evaluator is optimizing a resource with software that asks for production costs from various benches as part of the optimization process.

Cycle times for excavators are, for the most part, fixed and related to machine size. Wheel loaders are the exception in that they are sometimes called on to travel a short distance from the active face to the loading point. Most tracked excavators simply pivot after they collect a load of broken rock to transfer that load to the truck. Truck cycle times are more complicated. Although some of the time components are fixed (spot, load, dump, and turn), travel times typically represent the largest component of a truck's cycle. It is also the component that typically has the greatest impact in distinguishing costs at one project from those at another.

Engineers attempt to achieve the following goals as they design the excavator and hauler segments of their mine plan:

- Three to six loader cycles should completely fill the truck bed.
- Loader bucket capacities should be selected so that, whatever the number of cycles, the truck is full or close to full after loading is complete. For instance, a 7.0-m^3 bucket could be used to fill a 21.0-m^3-capacity truck, but it would be inefficient if used to fill a 17.0-m^3-capacity truck. Two loads would not fill the 17.0-m^3-capacity truck completely, but three loads would overfill it.
- The number of trucks and the number of loaders should be determined to minimize both the amount of time that any loader must wait for a truck and the amount of time that any truck must wait in a queue to be loaded.

To meet these goals, engineers rely on a multistep process. First, they estimate the loader cycle time and use it in conjunction with the loader's bucket capacity to determine the number needed to meet production goals. This first step is straightforward and proceeds as shown in the following example.

Example 3

Estimate daily excavator and truck use for the following situation:

- Shift length ≈ 8 h
- Production schedule ≈ 2 shifts/d
- Waste production capacity ≈ 18,000 t/d
- Front-end-loader bucket capacity (volume) ≈ 11.5 m^3
- Front-end-loader bucket capacity (weight) ≈ 21.7 t
- Average bucket fill factor ≈ 90%
- Material weight ≈ 2,400 kg/m^3
- Material swell ≈ 55%
- Cycle time
  - Load ≈ 12 s
  - Lift and swing time ≈ 12 s
  - Dump time ≈ 8 s
  - Return and lower time ≈ 10 s
- Rolling resistance ≈ 3%
- Haul profile
  - *Segment 1:* From working face across pit floor, 200 m at 0% gradient
  - *Segment 2:* From pit floor to pit entrance, 1,200 m at 12% gradient
  - *Segment 3:* From pit entrance to waste stockpile, 1,600 m at –6% gradient
  - *Segment 4:* From base of stockpile to top of stockpile floor, 800 m at 12% gradient
  - *Segment 5:* From top of stockpile across to dump point, 200 m at 0% gradient
- Operator efficiency ≈ 83%

1. Bucket load:

   2,400 kg/m^3 ÷ [1 + (55% swell ÷ 100)] = 1,550 kg/m^3

   [11.5 m^3 × 1,550 kg/m^3 × 0.90 (fill factor)] ÷ 1,000 kg/t = 16.0 t

2. Total cycle requirement:

   18,000 t/d ÷ 16.0 t/cycle = 1,125 cycles/d

   [1,125 cycles/d × (12 s + 12 s + 8 s + 10 s)] ÷ 60 s/min = 787.5 min/d

3. Loader operators:

   [787.5 min/d ÷ 0.83 (efficiency)] ÷ 60 min/h = 15.8 h/d

   15.8 h/d ÷ 8 h/shift ≈ 2 operators

Next, an average truck cycle time is determined (see Example 4). Evaluators base both haul and return times (travel times) on the length of the haul and the average gradient over that length. Gradient is defined as the change in elevation divided by the length over which that change takes place. A downhill gradient is typically reported as a negative value, and an uphill gradient is reported as a positive value.

Engineers typically obtain the speeds of the trucks over these haul distances through rimpull/speed/gradeability curves and retarder curves, which are specific for each vehicle. Examples of these curves can be found in the *Caterpillar Performance Handbook* (Caterpillar 2009). Travel speeds (with the vehicle either loaded or empty) for down-gradient segments can be gleaned from the retarder curves, and travel speeds for up-gradient segments (loaded or empty) are taken from the rimpull/speed/gradeability curves.

On examination, it is apparent that the data in these curves can also be subjected to geometric regression analyses to provide relationships that estimators can then use to approximate speeds based on the specific gradients. These relationships do not provide precise results, but they are more than adequate for early-stage cost-estimating purposes. Estimators typically ignore increases in travel times due to acceleration or deceleration over haul lengths of any significance. Over very short hauls, these need to be considered.

Evaluators also need to adjust the travel gradients for rolling resistance. Because of the flexibility inherent in roadbeds and the weight on the tires, trucks always "sink" into the road surface just a little as they travel along. One way to visualize rolling resistance is to view it as the gradient that the tire must continually overcome to drive out of the slight depression that it creates in the roadbed because of the weight that it carries.

Example 4

Consider the following situation:

- Bed capacity (volume) ≈ 60 m^3
- Bed capacity (weight) ≈ 90 t
- Material weight ≈ 2,400 kg/m^3
- Material swell ≈ 55%
- Turn and spot time ≈ 15 s
- Dump time ≈ 8 s
- Return and lower time ≈ 12 s

First, the time to load the truck is estimated from the previous example.

1. Load time:

   60 m^3 bed capacity ÷ [11.5 m^3 bucket capacity × 0.85 (fill factor)] ≈ 6 cycles to load

   [6 cycles × (12 s + 12 s + 8 s + 10 s)] ÷ 60 s/min ≈ 4.20 min/truck

2. Travel time—fully loaded:

   Segment 1: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 43 km/h] × 60 min/h = 0.54 min

   Segment 2: 12% gradient resistance + 3% rolling resistance = 15% total resistance

   [(1,200 m ÷ 1,000 km/m) ÷ 9 km/h] × 60 min/h = 8.00 min

   Segment 3: –6% gradient resistance + 3% rolling resistance = –3% total resistance

   [(1,600 m ÷ 1,000 km/m) ÷ 64 km/h] × 60 min/h = 1.50 min

   Segment 4: 12% gradient resistance + 3% rolling resistance = 15% total resistance

   [(800 m ÷ 1,000 km/m) ÷ 9 km/h] ÷ 60 min/h = 5.33 min

   Segment 5: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 43 km/h] × 60 min/h = 0.54 min

   total haul time loaded = 15.37 min

3. Turn and dump time = 1.20 min.
4. Travel time—return empty:

   Segment 5: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 63 km/h] × 60 min/h = 0.38 min

   Segment 4: –12% gradient resistance + 3% rolling resistance = 9% total resistance

   [(800 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 1.50 min

   Segment 3: 6% gradient resistance + 3% rolling resistance = 9% total resistance

   [(1,600 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 3.00 min

   Segment 2: –12% gradient resistance + 3% rolling resistance = 9% total resistance

   [(1,200 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 2.25 min

   Segment 1: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 63 km/h] × 60 min/h = 0.38 min

   total haul time for return = 7.51 min

5. Turn and spot to load = 0.80 min
6. Total cycle time:
   - Load = 4.20 min
   - Travel loaded = 15.37 min
   - Turn and dump = 1.20 min
   - Return time = 7.51 min
   - Turn and spot to load = 0.80 min
   - Total cycle time = 29.08 min
7. Required number of trucks:

   6 cycles/truck × 11.5 m^3/cycle × 0.9 (fill factor) × 1,550 kg/ m^3 = 96,225 kg/load

   18,000 t/d ÷ (96,225 kg/load ÷ 1,000 kg/t) = 187.1 loads/d

   187.1 loads/d × 29.08 min/load = 5,440.9 min/d

   5,440.9 min/d ÷ (2 shifts/d × 8 h/shift × 60 min/h) ≈ 6 trucks

8. Time spent in queue:

   29.08 min/cycle ÷ 6 trucks = 4.85 min available to load truck

   Because 4.85 minutes > 4.20 min/load, then

   time the loader spends waiting for a truck = 4.85 min – 4.20 min = 0.65 min

   Had the time that the loader spent waiting for a truck been negative (i.e., trucks have to wait for the loader), it would have been necessary to increase the size of the loader.
9. Truck drivers:

   5,440.9 min/d ÷ 0.83 (efficiency) ÷ 60 min/h = 109.3 h/d

   109.3 h/d ÷ 8 h/shift ≈ 14 operators

Because of the efficiency of the truck drivers, it is apparent in these estimates that at least one (and probably two) utility operators will be needed to drive the trucks while the regular drivers take their breaks. And if this estimate is to represent the costs at a point in time halfway through the project, then more trucks may be needed later in the project as the pit deepens and haul distances increase. The six trucks indicated at this point are operating close to their maximum capacity.

If, instead of estimating the travel times over each haul segment, evaluators had calculated an average haul gradient over the entire haul distance, the results would have been noticeably different because the average gradient is about 5.0%. The significantly slower segment where the gradients are 12% would not have the same impact on the analysis and the haul-cycle time would be reduced by more than 1.5 minutes. This is one example where a high level of detail, even at a very early stage of an analysis when information is scarce, has a significant impact on the reliability of the results.

Most of the machine specifications stated in the excavator and truck examples were taken from the *Caterpillar Performance Handbook* (Caterpillar 2009). Machine specifications are very often available from equipment manufacturers. Similar information can also be found in *Mining Cost Service* (InfoMine 2009b).

ANCILLARY SYSTEMS

To produce a complete cost estimate, much work remains for the evaluator even after the drilling, blasting, excavating, and hauling costs have been determined. However, these previously determined costs (along with the parameters derived during the estimation process) do provide a basis for estimates of the remaining costs.

To begin with, the costs associated with many of the machines typically found at any surface mine have yet to be estimated. For most such projects, costs for bulldozers, graders, dust suppressant tankers, equipment-maintenance trucks, pumps, lighting plants, personnel movers, and, in some cases, generators, crushers, and conveyors, may all need to be estimated and included.

For each of these machine types, the determining factors that provide the basis for the estimated costs are (as with the drills, excavators, and haulers) the capacity of the machine and how many hours it must operate each day. Consequently, the techniques that evaluators rely on to estimate the capacity and daily use parameters for each are similar to those they use to gauge the same parameters for the drills, excavators, and haulers.

For instance, at almost every surface mine, a fleet of bulldozers manages blasted rock at the working faces and dumped waste rock at the stockpiles (in addition to performing a host of other tasks). The process that estimators use to determine the number and operating requirements of these machines is rarely as straightforward as the process that they use to determine the excavator and loader needs, but it is still based on a very similar approach.

At all but the smallest operations, bulldozers work continuously at each dump site. They also often work at each active face, moving scattered, broken rock to the excavator. The size requirements for these machines are based on the amount of material that they handle each shift and the distance that the material must be moved. Specifically, each blade load carries with it a volume that will be moved over a distance at a speed typically specified in the manufacturer's documentation. With these three parameters (speed, distance, and capacity), an engineer can approximate productivity through the following general relationship:

$$\text{productivity, t/h} = (\text{volume, m}^3 \times \text{density, t/m}^3 \times \text{velocity, m/h}) \div \text{distance, m}$$

And with that, the engineer can estimate daily use as follows:

$$\text{daily use, h/d} = \text{production rate, t/d} \div \text{productivity, t/h}$$

When hourly costs (available from the sources mentioned earlier) are applied to the daily use (hours per day) values, the results report in terms of dollars per day. To arrive at a final value in terms of dollars per metric ton of ore, evaluators need only to divide the dollars per day value by the metric tons of ore mined each day.

Evaluators can use a similar process to arrive at daily use values for the graders and dust suppressant tankers. For instance, consider a road built from friable material in a wet environment that would need to be graded twice daily. If the length and width of the road are known, then an estimator can use the following general relationship to approximate the amount of time spent grading the road each day:

$$\text{productivity, m}^2\text{/h} = \text{velocity, m/h} \times \text{blade width, m}$$

And with this, the evaluator can estimate daily use (in terms of hours per day) as follows:

$$\text{daily use, h/d} = \text{grading requirements, m}^2\text{/d} \div \text{productivity, m}^2\text{/h}$$

As previously demonstrated, when hourly costs are applied to the daily use values, the results report in terms of dollars per day and subsequently in dollars per metric ton of ore.

CAPITAL AND OPERATING COSTS

For the purpose of project evaluation, costs are typically categorized as either *operating* or *capital* (as opposed to *fixed* or *variable*) so that they can be subjected to after-tax, discounted cash-flow analyses. In short, operating costs are those that can be fully expensed in the year incurred. The expenses of the consumables (including those associated with equipment operation), wages, and salaries are typically all considered operating costs and are most often estimated either in terms of dollars per metric ton of ore or dollars per year. Capital costs are those that cannot be fully expensed in the year incurred and include items such as the following:

- Mine and mill equipment purchase
- Development
- Engineering and construction management
- Infrastructure
- Working capital
- Postproduction reclamation
- Preproduction stripping
- Property acquisition
- Exploration
- Buildings
- Contingency fund

Although this chapter deals primarily with costs and cost estimation, it is worth mentioning that from an after-tax economic-viability standpoint, it is best to minimize the preproduction capital expenses and incur them as close to start-up as possible. Because of the time value of money, capital expenses accrued later in the operation have a lesser impact on the overall project net present value. So, during the preliminary mine-design process, evaluators find that it is worth the effort to structure the project in a way that expedites production.

COST COMPONENTS

While expenses at a mine can be categorized as either *capital costs* or *operating costs*, both are comprised almost entirely of labor, supply, and equipment components. Whether building a processing plant, constructing a tailings impoundment, or mining an ore deposit, evaluators find that most of the money spent on the project goes to either the workers (laborers, skilled tradesmen, equipment operators, supervisors, technicians, managers, etc.), the supply vendors (to purchase wood, drill bits, concrete, steel, explosives, tires, diesel, etc.), or to equipment manufacturers (to purchase machines or buy parts).

LABOR

Wages and salaries, and the "burdens" associated with each, very often represent the largest expenditures at any mineral-development project. In fact, wages, benefits, mandated employment taxes, and bonuses can sometimes account for more than half of the total operating costs. Depending on the size of the mine, labor costs can account for anywhere from 15% to 60% of the total operating costs. These costs can escalate if the mine is situated in a remote area without a local source of skilled labor.

Wages also tend to be one of the more variable components of an evaluation. Project location has a significant impact, and evaluators are urged to examine wages on a regional level to properly account for the associated expenses in their evaluation. Wages and salaries for miners in several countries are tracked and reported in publications such as *U.S. Metal and Industrial Mineral Mine Salaries, Wages and Benefits: 2009 Survey Results* (Salzer 2009).

Base wages are "loaded" with mandated employment taxes, including Social Security, Medicare, unemployment taxes, and workers' compensation taxes. Other items add to the burden factor, such as shift differentials; overtime; medical, dental, and vision benefits; retirement plans; short- and long-term disability insurance; life insurance; accidental death and dismemberment insurance; sick leave; vacation and holiday pay; and other benefits. To retain employees, companies often use creative benefits such as paid tuition, transportation to remote mine sites, attendance bonuses, safety bonuses, family and individual assistance plans, and paid fitness-club memberships.

Many mines pay their production miners a bonus based on meeting development or production goals. These bonus systems are sometimes modified to include safety, ground conditions, and other factors. Safety violations can reduce or even eliminate a production bonus. Other criteria sometimes used to calculate bonuses are individual performance, safety performance, commodity price, profit, recovery, ore grade, production, and cost savings to calculate bonuses.

SUPPLIES

Supply prices are less volatile than wages and salaries, but they still vary from one region to the next and from one vendor to another. While it is always preferable to obtain local prices from established vendors, it is often impractical to do so during the early stages of project evaluation. *Mining Cost Service* (InfoMine 2009b) provides an extensive array of supply costs that are reliable for early-stage feasibility work.

In the evaluation process, some supply costs are commonly reported as equipment-operating costs because their consumption rates are directly tied to machine use. Diesel fuel, gasoline, electricity, tires, and lubricants all fall into this category. And, as demonstrated earlier, many of the project's labor requirements and subsequent costs are also directly dictated by daily machine use. In addition to individual equipment-operator requirements, mechanic, electrician, machinist, and equipment-maintenance worker requirements also vary in proportion to machine use.

EquipmentWatch's *Cost Reference Guide* (EquipmentWatch 2009), InfoMine USA's *Mine and Mill Equipment Costs: An Estimator's Guide* (InfoMine USA 2009a), and equipment manufacturer publications such as Caterpillar's *Caterpillar Performance Handbook* (Caterpillar 2009) provide invaluable equipment operating parameter and cost information. As such, they also serve as indirect guides to mechanic requirements and equipment-related supply consumption rates (fuel, electricity, tires, and lubricants). Some recent prices for supplies commonly consumed at surface mines are listed in Table 4.9-1.

Table 4.9-1 Supply prices (2009 dollars)

Item	Price per Unit, US$
Ammonium nitrate fuel oil (ANFO)	1.06/kg
Extra-gelatin dynamite	4.50/kg
Primers (0.45 kg)	4.58 each
Blasting caps (nonelectric, 6-m lead)	3.15 each
Detonation cord (25 grain)	0.063/m
Detonation cord (40 grain)	0.073/m
Rotary drill bits (17.15 cm)	2,746 each
Rotary drill pipe	81.58/m
Percussion drill bits (4.45 cm)	58.80 each
Percussion drill steel	78.63/m
Dust suppressant	0.82/L
Diesel fuel	0.720/L
Lubricants	2.171/L
Cement	112.36/t
Tailings pipe (20.3-cm abrasion-resistant steel)	56.98/m
Water return pipe (10.2-cm polyvinyl chloride)	15.88/m
Synthetic liner (36-mil Hypalon)	11.04/m^2
Geotextile	4.16/m^2
Soil stabilizer	3.20/L

EQUIPMENT

As with the supply prices, equipment purchase prices are typically obtained from vendors. However, in the early stages of an evaluation, it is even more difficult to obtain these values than it is the supply costs because the necessary machines can only be specified in the most general terms. InfoMine USA's *Mining Cost Service* (InfoMine 2009b) and *Mine and Mill Equipment Costs: An Estimator's Guide* (InfoMine USA 2009a) also contain extensive purchase price lists for machines commonly used at surface mines. For early-stage feasibility work, equipment prices are usually based on list prices as suggested by the manufacturers, with no discounts assumed and no options added.

Early in the evaluation process, most evaluators specify new machines for all production-related project requirements, and purchase prices reflect this. If anticipated use is minimal, estimators may specify previously owned (used) machines for some of the secondary support equipment, such as water tankers and road graders. If previously owned machines are relied on for production work, the equipment productivity and availability (and the associated operating costs) should be adjusted accordingly in anticipation of increased maintenance and repair requirements.

COST MODELS

When evaluators have limited deposit information, they can use mine models for order-of-magnitude estimates. In addition, models can be used to provide insight into the nature of mining costs in general. The impact of changes in operating parameters can be easily understood when presented in a format that compares costs associated with one configuration directly to those associated with another.

Appendix 4.9A presents three cost models that evaluators can use to make preliminary estimates for projects. The cost models include surface mines of 1,000, 10,000, and 80,000 t/d.

Each cost model compares stripping ratios of 1:1, 2:1, 4:1, and 8:1 (waste to ore). In these cost models, note the

Table 4.9-2 Hourly wages for personnel (2009 dollars)

Worker	Hourly Wage, US$
Driller	22.20
Blaster	22.36
Excavator operator	22.80
Truck driver	19.97
Heavy equipment operator	21.98
Utility operator	18.17
Mechanic	22.53
Electrician	23.90
Maintenance worker	18.20
Laborer	17.30

Source: Salzer 2009.

Table 4.9-3 Annual salaries for professionals (2009 dollars)

Job Title	Annual Salary, US$
Mine manager	147,600
Superintendent	92,600
Foreman	71,700
Engineer	79,500
Geologist	73,000
Technician	48,700
Accountant	64,600
Purchasing agent	63,100
Personnel manager	89,800
Secretary	35,700
Clerk	39,700

Source: Salzer 2009.

unexpected changes dictated by increases in stripping ratios at the larger operations, and also note the ratio of labor costs to equipment operating costs as production rates increase.

These models are theoretical and are not representative of any existing mine. Note the pit and haul parameters, which provide the basis for each design. These should be one of the key points of comparison if the models are to be used to provide estimates for any proposed operation. Costs associated with each model account for all pertinent labor, material, supply, and equipment operating expenses accrued at the mine site. Costs for supervision, administration, and on-site project management are all included. Expenses associated with preproduction development, equipment purchase and installation, and building and facility construction are also included. In these models, costs for the following operations and facilities are considered:

- Ore and waste drilling, blasting, and excavation
- Ore haul from the active face to the mill site
- Overburden and waste haul from the active face to the dumpsite
- Constructing and operating the facilities required for equipment maintenance and repair, electricity and fuel distribution, drainage, explosives storage, and sanitation
- Constructing a mine office, a warehouse, and a worker changehouse plus all associated site work

The mines in the cost models are located in areas of moderate relief with warm summers and temperate winters. Wages and salaries used in the cost models are from *U.S. Metal and Industrial Mineral Mine Salaries, Wages and Benefits: 2009 Survey Results* (Salzer 2009).

Representative hourly wages used in the models are listed in Table 4.9-2. In the models, the above wages are adjusted upward to account for a 39.0% burden rate. Salaries for professionals that were used in the models are shown in Table 4.9-3, and in the models, the salaries are adjusted upward to account for a 39.0% burden rate. Models were constructed with Sherpa Cost Estimating Software for Surface Mines (Stebbins 2009).

REFERENCES

Caterpillar, Inc. 2009. *Caterpillar Performance Handbook.* Periodically updated. Peoria, IL: Caterpillar.

EquipmentWatch. 2009. *Cost Reference Guide.* Periodically updated. San Jose, CA: EquipmentWatch. Available from www.EquipmentWatch.com.

Hoskins, J.R. 1977. *Mineral Industry Costs.* Spokane, WA: Northwest Mining Association.

InfoMine USA. 2009a. *Mine and Mill Equipment Costs: An Estimator's Guide.* Periodically updated. Spokane, WA: InfoMine USA. Available from http://costs.infomine.com.

InfoMine USA. 2009b. *Mining Cost Service*. Periodically updated. Spokane, WA: InfoMine USA. Available from http://www.costs/infomine.com.

Mular, A.L. 1982. *Mining and Mineral Processing Equipment Costs and Preliminary Capital Cost Estimates.* Special Volume 25. Ottawa: Canadian Institute of Mining and Metallurgy.

O'Hara, T.A. 1980. Quick guides to the evaluation of ore bodies. *CIM Bulletin*.

Olofsson, S.O. 1997. *Applied Explosives Technology for Construction and Mining.* Arla, Sweden: Dyno Nobel Allied Explosives Technology.

Salzer, K.N. 2009. *U.S. Metal and Industrial Mineral Mine Salaries, Wages, and Benefits: 2009 Survey Results.* Periodically updated. Spokane, WA: InfoMine USA. Available from http://costs/infomine.com.

Scales, M., ed. 2009. *Mining Source Book*. Don Mills, ON: Canadian Mining Journal.

Schumacher, O., and Stebbins, S. 1995. *Other Cost Estimating Methods*. Produced for the Bureau of Land Management National Training Center.

Stebbins, S.A. 2009. Sherpa Cost Estimating Software for Surface Mines. Spokane, WA: Aventurine Mine Cost Engineering.

USBM (U.S. Bureau of Mines). 1987. *Bureau of Mines Cost Estimating System Handbook.* Information Circulars 9142 and 9143. Washington, DC: U.S. Bureau of Mines.

Vilbrandt, F.C., and Dryden, C.E. 1959. Factored capital cost estimate guide. In *Chemical Engineering Plant Design*. New York: McGraw-Hill.

APPENDIX 4.9A

The following tables present cost models for a surface mine producing ore at waste-to-ore strip ratios of 1:1, 2:1, 4:1, and 8:1.

Table 4.9A-1 Surface mines: 1,000 metric tons of ore per day

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Ore production, t/d	1,000	1,000	1,000	1,000
Waste production, t/d	1,000	2,000	4,000	8,000
Total resource, million t	3.12	3.12	3.12	3.12
Final pit dimension				
Pit depth, m	84	97	115	141
Pit floor length, m	154	177	210	256
Pit floor width, m	77	88	105	128
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	87	91	98	110
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	414	564	770	1,031
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	342	394	468	571
Gradient, %	2.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	117	130	150	180
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	112	173	251	343
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	171	197	234	286
Gradient, %	2.0	2.0	2.0	2.0
Stockpile base to surface				
Distance, m	213	269	339	427
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	109	137	172	217
Gradient, %	0.0	0.0	0.0	0.0
Hours per shift	8	8	8	8
Shifts per day	2	2	2	2
Days per year	312	312	312	312
Bench height—ore, m	3.66	3.66	3.66	3.66
Bench height—waste, m	4.88	4.88	4.88	4.88
Powder factor—ore, kg/t	0.35	0.35	0.35	0.35
Powder factor—waste, kg/t	0.31	0.31	0.31	0.31
Development				
Preproduction stripping, t	30,000	60,000	120,000	240,000
Haul road construction, m	1,565	1,955	2,483	3,165
Equipment, number and size				
Hydraulic shovels, m^3	1 each 2.3	1 each 2.3	1 each 2.3	1 each 2.3
Front-end loaders, m^3	1 each 2.3	1 each 2.3	1 each 3.8	1 each 12.2
Rear-dump trucks, t	4 each 32.0	5 each 32.0	4 each 41.0	6 each 54.0

(continues)

Table 4.9A-1 Surface mines: 1,000 metric tons of ore per day (continued)

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Percussion drills, cm	2 each 6.35	3 each 6.35	4 each 6.35	6 each 6.35
Bulldozers, kW	2 each 60	3 each 60	4 each 60	3 each 110
Graders, kW	1 each 105	1 each 105	1 each 115	1 each 115
Water tankers, L	—	—	1 each 9,500	1 each 9,500
Service/tire trucks, kg GVW*	2 each 1,800	3 each 1,800	2 each 6,800	3 each 6,800
Bulk trucks, kg GVW	1 each 2,000	1 each 2,000	1 each 2,000	1 each 2,000
Light plants, kW	4 each 7.8	4 each 7.8	4 each 7.8	4 each 7.8
Pumps, kW	2 each 3.7	2 each 7.5	2 each 11.2	2 each 14.9
Pickup trucks	3	3	3	5
Buildings				
Shop, m^2	266	332	337	583
Dry, m^2	157	209	232	313
Office, m^2	204	230	256	383
Warehouse, m^2	167	167	174	224
Hourly personnel requirements				
Drillers	3	4	5	9
Blasters	2	2	2	2
Excavator operators	3	3	3	3
Truck drivers	5	8	7	11
Equipment operators	5	7	7	7
Utility operators	1	1	3	3
Mechanics/electricians	3	4	5	7
Laborers/maintenance	5	7	8	12
Total hourly personnel	**27**	**36**	**40**	**54**
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	0	0	0	0
Foreman	2	2	2	2
Engineer	1	1	1	1
Geologist	0	0	0	1
Supervisor	1	1	1	2
Technician	2	2	3	4
Accountant	0	0	0	0
Clerk	0	1	1	1
Personnel manager	0	0	0	0
Secretary	1	1	1	2
Warehouse	0	0	0	1
Total salaried personnel	**8**	**9**	**10**	**15**
Primary supply requirements				
Diesel fuel, L/d	1,502	2,195	3,047	5,225
Powder, kg/d	660	970	1,590	2,830
Caps, units/d	50	69	106	180
Drill bits, units/d	0.101	0.149	0.244	0.434
Detonation cord, m/d	445	653	1,071	1,906

(continues)

Table 4.9A-1 Surface mines: 1,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	4.21	6.17	10.09	17.92
Labor	4.84	6.23	7.61	10.01
Equipment operation	1.67	2.31	4.05	7.56
Administration	2.20	2.46	2.67	3.75
Sundry items	1.29	1.72	2.44	3.92
Total operating costs	**14.21**	**18.89**	**26.86**	**43.16**
Capital costs, $				
Equipment	2,725,700	3,493,500	4,575,100	8,027,800
Haul roads/site work	534,600	642,900	1,241,100	1,600,600
Preproduction stripping	164,100	281,100	502,300	880,800
Buildings	691,200	780,700	1,349,100	1,985,200
Electrical system	68,700	95,000	101,900	111,500
Working capital	417,200	565,200	793,800	1,324,600
Engineering and management	342,300	433,900	636,000	1,036,600
Contingency	452,700	572,700	840,600	1,364,200
Total capital costs	**5,396,500**	**6,865,000**	**10,039,900**	**16,331,300**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Ore production, t/d	10,000	10,000	10,000	10,000
Waste production, t/d	10,000	20,000	40,000	80,000
Total resource, million t	37.44	37.44	37.44	37.44
Final pit dimension				
Pit depth, m	193	222	264	322
Pit floor length, m	352	404	481	587
Pit floor width, m	176	202	241	293
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	198	208	225	253
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	947	1,290	1,760	2,360
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	783	901	1,072	1,308
Gradient, %	2.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	268	298	345	412
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	254	394	572	782
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	392	451	536	654
Gradient, %	2.0	2.0	2.0	2.0

(continues)

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Stockpile base to surface				
Distance, m	489	616	776	977
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	249	313	395	497
Gradient, %	0.0	0.0	0.0	2.0
Hours per shift	10	10	10	10
Shifts per day	2	2	2	2
Days per year	312	312	312	312
Bench height—ore, m	4.60	4.60	4.60	4.60
Bench height—waste, m	6.72	6.72	6.72	6.72
Powder factor—ore, kg/t	0.33	0.33	0.33	0.33
Powder factor—waste, kg/t	0.29	0.29	0.29	0.29
Development				
Preproduction stripping, t	300,000	600,000	1,200,000	2,400,000
Haul road construction, m	3,580	4,470	5,681	7,244
Equipment, number and size				
Hydraulic shovels, m^3	1 each 8.4	1 each 8.4	1 each 8.4	1 each 8.4
Front-end loaders, m^3	1 each 12.2	2 each 16.1	2 each 19.9	4 each 19.9
Rear-dump trucks, t	11 each 54.0	11 each 77.0	20 each 91.0	34 each 100.0
Rotary drills, cm	3 each 20.00	2 each 25.08	2 each 27.94	4 each 31.12
Bulldozers, kW	4 each 110	5 each 140	6 each 180	9 each 180
Graders, kW	1 each 115	1 each 140	2 each 140	2 each 140
Water tankers, L	1 each 19,000	1 each 19,000	1 each 26,500	1 each 30,000
Service/tire trucks, kg GVW*	5 each 6,800	5 each 11,000	9 each 11,000	15 each 11,000
Bulk trucks, kg/min	1 each 450	1 each 450	1 each 450	2 each 450
Light plants, kW	4 each 8.9	4 each 10.1	5 each 10.1	7 each 10.1
Pumps, kW	3 each 37.3	3 each 74.6	4 each 74.6	5 each 93.2
Pickup trucks	7	8	12	17
Buildings				
Shop, m^2	908	1,168	2,404	5,030
Dry, m^2	394	441	738	1,168
Office, m^2	587	715	1,047	1,533
Warehouse, m^2	363	643	696	1,159
ANFO storage bin, m^2	64	80	130	234
Hourly personnel requirements				
Drillers	4	3	4	5
Blasters	2	2	2	4
Excavator operators	4	6	6	10
Truck drivers	19	19	36	61
Equipment operators	9	11	13	19
Utility operators	3	3	4	5

(continues)

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Mechanics/electricians	10	12	24	37
Laborers/maintenance	17	20	39	60
Total hourly personnel	**68**	**76**	**128**	**201**
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	1	1	1	1
Foreman	2	2	4	4
Engineer	2	2	3	5
Geologist	1	2	3	4
Supervisor	3	3	6	9
Technician	5	6	8	11
Accountant	1	1	2	3
Clerk	2	3	4	7
Personnel manager	1	2	2	4
Secretary	3	4	5	8
Security	1	1	2	3
Total salaried personnel	**23**	**28**	**41**	**60**
Primary supply requirements				
Diesel fuel, L/d	11,307	18,396	32,654	62,939
Powder, kg/d	7,250	9,102	14,903	26,505
Caps, units/d	72	55	67	92
Primers, units/d	68	51	63	88
Drill bits, units/d	1.570	1.253	1.654	2.372
Detonation cord, m/d	861	688	907	1,301
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	1.69	2.07	3.33	5.74
Labor	1.91	2.42	3.68	6.33
Equipment operation	1.44	2.49	4.79	9.44
Administration	0.69	0.84	1.20	1.74
Sundry items	0.57	0.78	1.30	2.33
Total operating costs	**6.30**	**8.60**	**14.30**	**25.58**
Capital costs, $				
Equipment	13,956,400	22,375,800	45,083,900	88,465,500
Haul roads/site work	2,183,300	3,050,600	6,007,700	7,942,200
Preproduction stripping	824,200	1,438,700	2,731,400	5,609,500
Buildings	3,217,500	3,803,900	6,191,900	10,826,200
Electrical system	179,200	190,200	406,100	428,000
Working capital	1,631,900	2,035,800	3,282,600	5,567,800
Engineering and management	2,105,500	3,252,400	6,310,300	11,877,200
Contingency	2,246,600	3,411,200	6,673,100	12,514,900
Total capital costs	**26,344,600**	**39,558,600**	**76,687,000**	**143,231,300**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Ore production, t/d	80,000	80,000	80,000	80,000
Waste production, t/d	80,000	160,000	320,000	640,000
Total resource, million t	350.4	350.4	350.4	350.4
Final pit dimension				
Pit depth, m	407	468	557	679
Pit floor length, m	740	852	1,014	1,237
Pit floor width, m	370	426	507	618
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	418	438	474	533
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	2,000	2,720	3,708	4,973
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	1,650	1,899	2,260	2,756
Gradient, %	0.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	565	628	726	868
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	533	831	1,204	1,648
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	825	949	1,130	1,378
Gradient, %	2.0	2.0	2.0	2.0
Stockpile base to surface				
Distance, m	1,030	1,298	1,635	1,960
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	524	660	831	1,247
Gradient, %	0.0	0.0	0.0	0.0
Hours per shift	8	8	8	8
Shifts per day	3	3	3	3
Days per year	365	365	365	365
Bench height—ore, m	5.49	5.49	5.49	5.49
Bench height—waste, m	8.53	8.53	8.53	8.53
Powder factor—ore, kg/t	0.28	0.28	0.28	0.28
Powder factor—waste, kg/t	0.23	0.23	0.23	0.23
Development				
Preproduction stripping, t	2,400,000	4,800,000	9,600,000	19,200,000
Haul road construction, m	7,544	9,423	11,969	15,364

(continues)

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day (continued)

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Equipment, number and size				
Cable shovels—ore, m^3	1 each 26.0	1 each 35.2	1 each 61.2	1 each 61.2
Cable shovels—waste, m^3	1 each 26.0	2 each 35.2	2 each 61.2	4 each 61.2
Rear-dump trucks, t	32 each 181	47 each 218	61 each 327	134 each 327
Rotary drills, cm	3 each 38.10	5 each 38.10	6 each 38.10	10 each 38.10
Bulldozers, kW	4 each 305	6 each 305	6 each 305	9 each 305
Graders, kW	2 each 160	3 each 160	5 each 160	6 each 160
Water tankers, L	1 each 30,000	1 each 53,000	2 each 53,000	3 each 53,000
Service/tire trucks, kg GVW*	13 each 20,500	19 each 20,500	28 each 20,500	45 each 20,500
Bulk trucks, kg/min	2 each 450	3 each 450	4 each 450	6 each 450
Light plants, kW	4 each 10.1	5 each 10.1	5 each 10.1	7 each 10.1
Pumps, kW	6 each 93.2	7 each 93.2	8 each 93.2	9 each 186.4
Pickup trucks	26	33	47	75
Buildings				
Shop, m^2	5,621	8,892	15,172	33,071
Dry, m^2	1,122	1,803	3,096	6,184
Office, m^2	2,325	2,964	4,318	6,820
Warehouse, m^2	1,488	3,533	5,519	9,199
ANFO storage bin, m^3	358	682	844	1,492
Hourly personnel requirements				
Drillers	7	12	15	26
Blasters	8	14	16	28
Excavator operators	6	9	9	15
Truck drivers	89	129	165	365
Equipment operators	13	19	19	28
Utility operators	9	10	16	20
Mechanics/electricians	65	112	225	449
Laborers/maintenance	93	161	334	661
Total hourly personnel	**290**	**466**	**800**	**1,592**

(continues)

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day (continued)

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	1	1	1	1
Foreman	6	6	6	6
Engineer	8	11	17	27
Geologist	7	10	17	30
Supervisor	12	12	12	12
Technician	14	17	22	29
Accountant	6	8	14	24
Clerk	12	17	28	50
Personnel manager	6	9	15	27
Secretary	12	16	23	36
Security	6	8	13	24
Total salaried personnel	**91**	**116**	**169**	**267**
Primary supply requirements				
Diesel fuel, L/d	79,815	129,236	310,370	670,455
Electricity, kW·h/d	57,262	87,550	164,508	280,565
Powder, kg/d	40,808	77,605	96,021	169,638
Caps, units/d	92	154	184	307
Primers, units/d	88	150	180	303
Drill bits, units/d	2.436	4.632	5.731	10.124
Detonation cord, m/d	1,366	2,541	3,144	5,554
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	1.11	2.10	2.60	4.59
Labor	0.88	1.40	2.39	4.55
Equipment operation	1.70	3.32	6.22	12.14
Administration	0.28	0.36	0.52	0.83
Sundry items	0.40	0.72	1.17	2.21
Total operating costs	**4.37**	**7.90**	**12.90**	**24.32**
Capital costs, $				
Equipment	93,345,400	235,141,700	441,435,900	844,119,800
Haul roads/site work	10,946,400	27,513,100	35,694,800	43,404,000
Preproduction stripping	4,553,100	23,065,300	21,687,700	45,707,400
Buildings	12,079,400	26,577,200	33,517,000	65,059,900
Electrical system	1,485,900	1,870,200	2,698,700	3,526,400
Working capital	8,818,800	16,957,500	22,500,200	40,011,200
Engineering and management	15,375,000	39,536,200	67,666,100	126,902,900
Contingency	13,778,500	35,370,400	60,270,000	112,872,000
Total capital costs	**160,382,500**	**406,031,600**	**685,470,400**	**1,281,603,600**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.