CHAPTER 4

Geometrical considerations

4.1 INTRODUCTION

The ore deposits being mined by open pit techniques today vary considerably in size, shape, orientation and depth below the surface. The initial surface topographies can vary from mountain tops to valley floors. In spite of this, there are a number of geometry based design and planning considerations fundamental to them all. These are the focus of this chapter. By way of introduction consider Figure 4.1 which is a diagrammatic representation of a volume at the earth's surface prior to and after the development of an open pit mine.

The orebody is mined from the top down in a series of horizontal layers of uniform thickness called benches. Mining starts with the top bench and after a sufficient floor area has been exposed, mining of the next layer can begin. The process continues until the bottom bench elevation is reached and the final pit outline achieved. To access the different benches a road or ramp must be created. The width and steepness of this ramp depends upon the type of equipment to be accommodated. Stable slopes must be created and maintained during the creation and operation of the pit. Slope angle is an important geometric parameter which has significant economic impact. Open pit mining is very highly mechanized. Each piece of mining machinery has an associated geometry both related to its own physical size, but also with the space it requires to operate efficiently. There is a complementary set of drilling, loading and hauling equipment which requires a certain amount of working space. This space requirement is taken into account when dimensioning the so-called working benches. From both operating and economic viewpoints certain volumes must or should, at least, be removed before others. These volumes have a certain minimum size and an optimum size.

It is not possible in this short chapter to try and fully cover all of the different geometrical aspects involved in open pit mine planning and design. However, the general principles associated with the primary design components will be presented and whenever possible illustrated by examples.

4.2 BASIC BENCH GEOMETRY

The basic extraction component in an open pit mine is the bench. Bench nomenclature is shown in Figure 4.2.

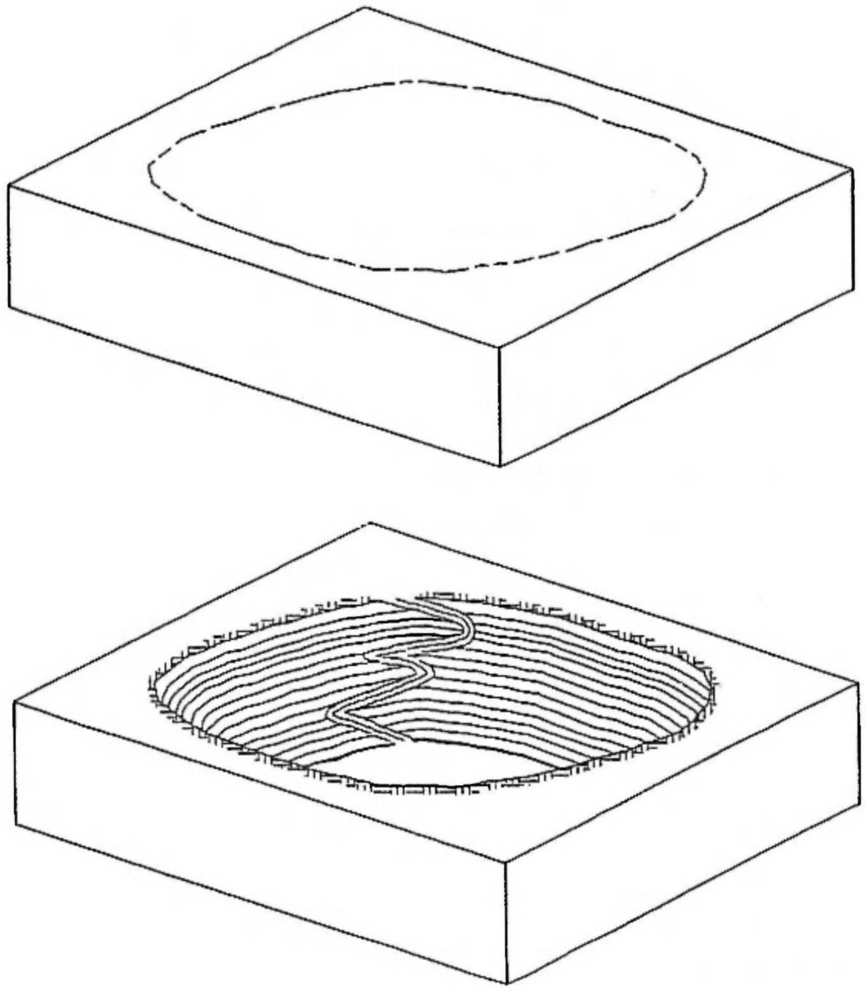


Figure 4.1. Geometry change in pit creation.

Each bench has an upper and lower surface separated by a distance H equal to the bench height. The exposed subvertical surfaces are called the bench faces. They are described by the toe, the crest and the face angle α (the average angle the face makes with the horizontal). The bench face angle can vary considerably with rock characteristics, face orientation and blasting practices. In most hard rock pits it varies from about 55° to 80°. A typical initial design value might be 65°. This should be used with care since the bench face angle can have a major effect on the overall slope angle.

Normally bench faces are mined as steeply as possible. However, due to a variety of causes there is a certain amount of back break. This is defined as the distance the actual bench crest is back of the designed crest. A cumulative frequency distribution plot of measured average bench face angles is shown in Figure 4.3.

The exposed bench lower surface is called the bench floor. The bench width is the distance between the crest and the toe measured along the upper surface. The bank width is the horizontal projection of the bench face.

There are several types of benches. A working bench is one that is in the process of being mined. The width being extracted from the working bench is called the cut. The width of the working bench W_B is defined as the distance from the crest of the bench floor to the new

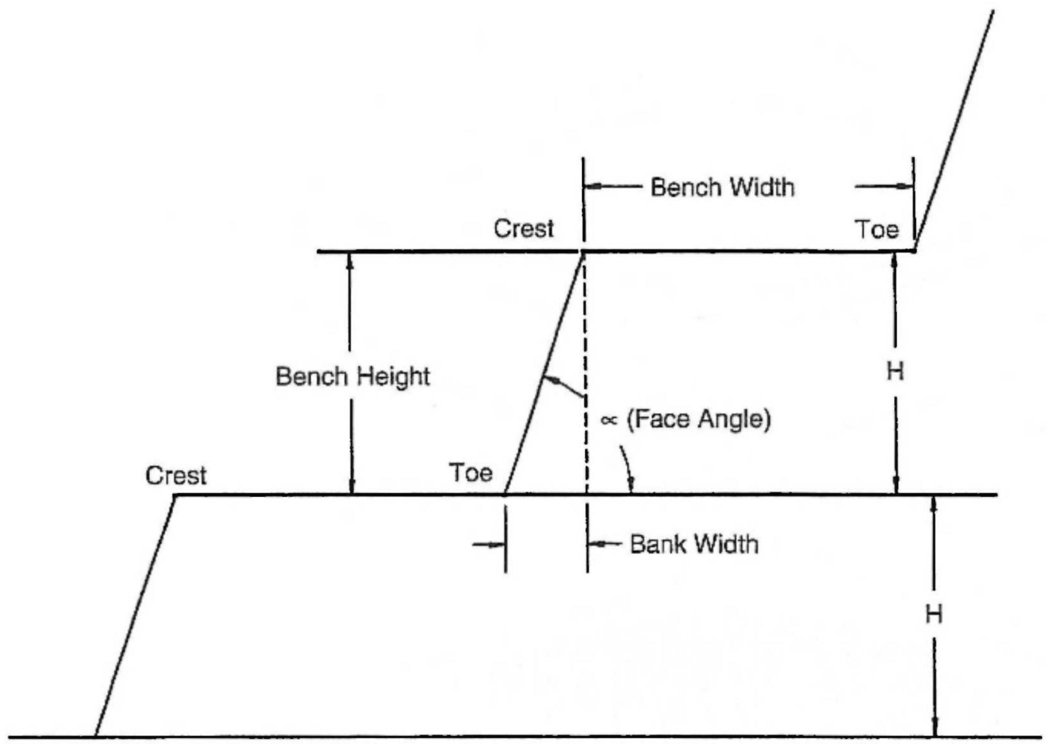


Figure 4.2. Parts of a bench.

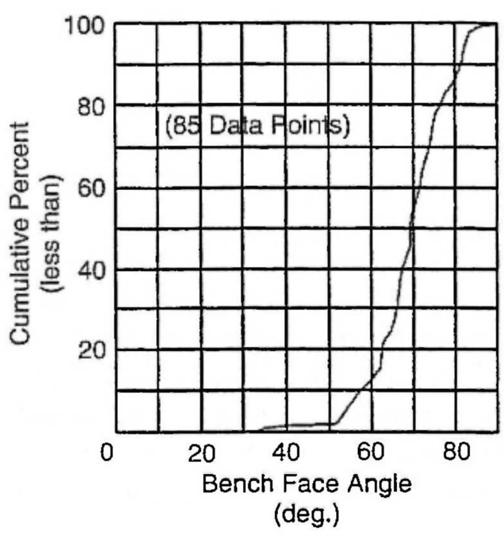


Figure 4.3. Cumulative frequency distribution of measured bench face angles (Call, 1986).

toe position after the cut has been extracted (see Fig. 4.4). A detailed calculation of cut and working bench dimensions is found in Subsection 4.4.5. After the cut has been removed, a safety bench or catch bench of width S_B remains.

The purpose of these benches is to:

(a) collect the material which slides down from benches above,

(b) stop the downward progress of boulders.

During primary extraction, a safety bench is generally left on every level. The width varies with the bench height. Generally the width of the safety bench is of the order of $2/3$ of the

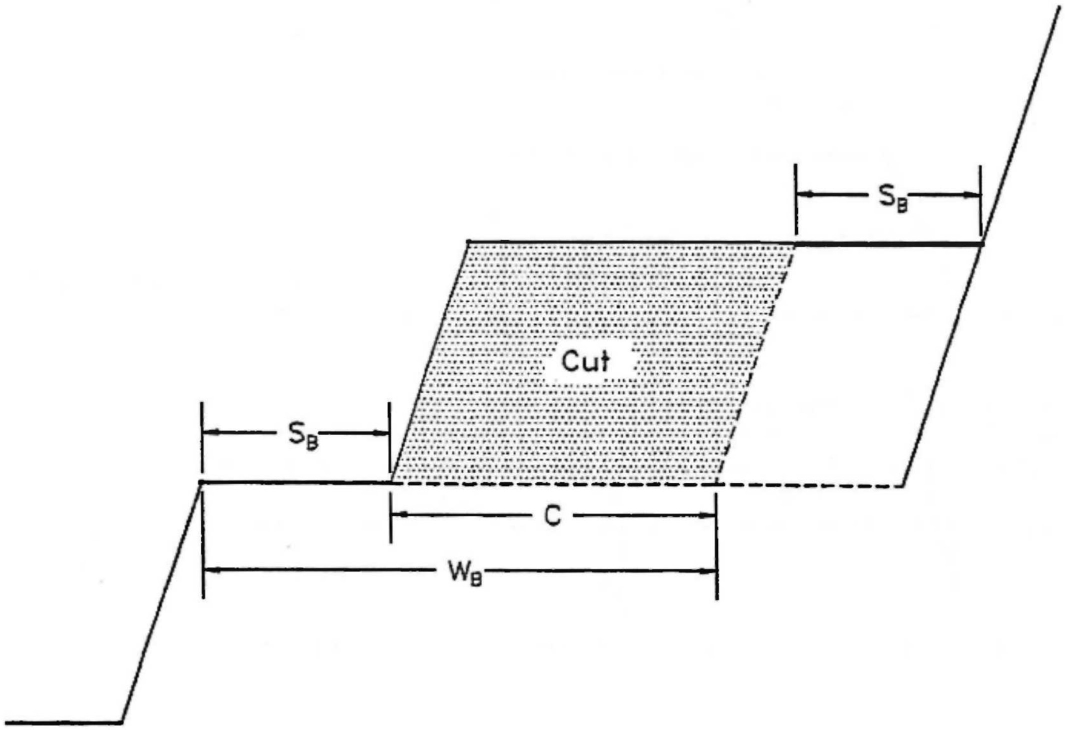


Figure 4.4. Section through a working bench.

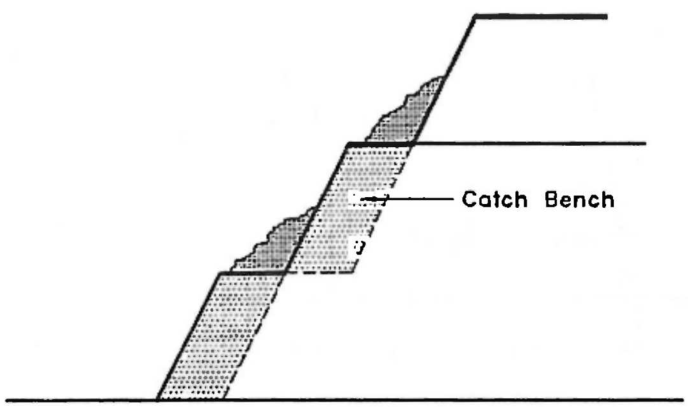


Figure 4.5. Functioning of catch benches.

bench height. At the end of mine life, the safety benches are sometimes reduced in width to about ⅓ of the bench height.

Sometimes double benches are left along the final pit wall (Fig. 4.6). These are benches of double height which consequently permit, at a given overall slope angle, a single catch bench of double width (and hence greater catching capability). Along the final pit contour careful blasting is done to maintain the rock mass strength characteristics.

In addition to leaving the safety benches, berms (piles) of broken materials are often constructed along the crest. These serve the function of forming a 'ditch' between the berm and the toe of the slope to catch falling rocks. Based upon studies of rock falls made by Ritchie (1963), Call (1986) has made the design catch bench geometry recommendations given in Table 4.1 and illustrated in Figure 4.7.

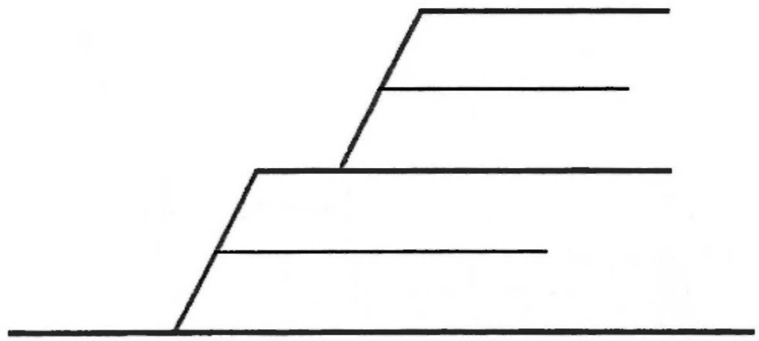


Figure 4.6. Double benches at final pit limits.

Table 4.1. Typical catch bench design dimensions (Call, 1986).

Bench height (m)	Impact zone (m)	Berm height (m)	Berm width (m)	Minimum bench width (m)
15	3.5	1.5	4	7.5
30	4.5	2	5.5	10
45	5	3	8	13

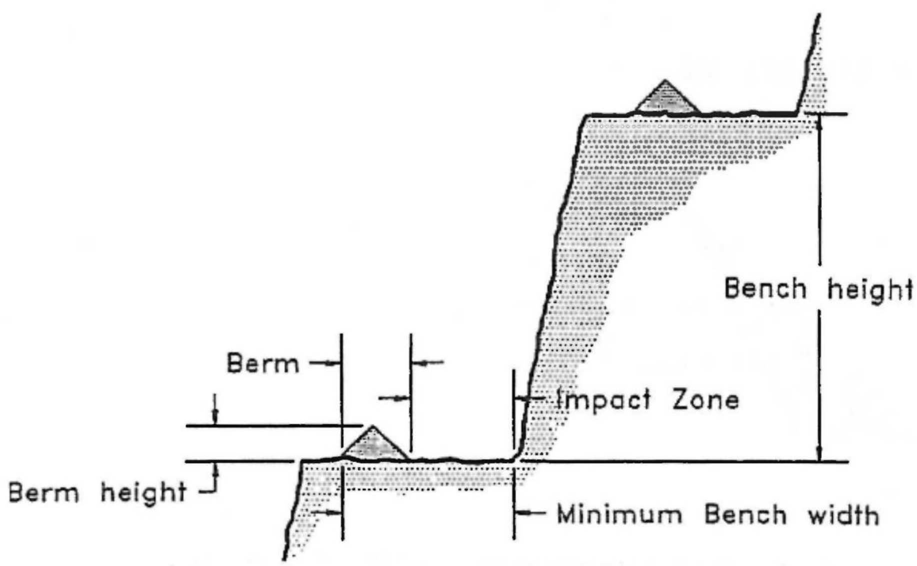


Figure 4.7. Catch bench geometry (Call, 1986).

A safety berm is also left (Fig. 4.8) along the outer edge of a bench to prevent trucks and other machines from backing over. It serves much the same function as a guard rail on bridges and elevated highways. Normally the pile has a height greater than or equal to the tire radius. The berm slope is taken to be about 35° (the angle of repose).

In some large open pits today median berms are also created in the center of haulage roads. In this book the word 'berm' is used to refer to the piles of rock materials used to improve mine safety. Others have used the word 'berm' as being synonymous with bench.

In the extraction of a cut, the drills operate on the upper bench surface. The loaders and trucks work off of the bench floor level.

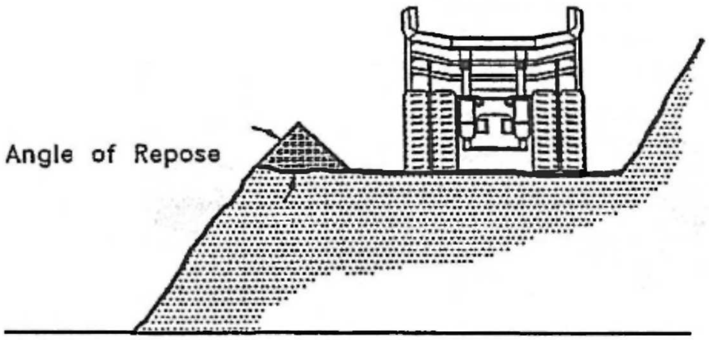


Figure 4.8. Safety berms at bench edge.

A number of different factors influence the selection of bench dimensions. Bench height becomes the basic decision since once this is fixed the rest of the dimensions follow directly. A common bench height in today's large open pits is 50 ft (15 m). For smaller pits the value might be 40 ft (12 m). For small gold deposits a typical value could be 25 ft (7.5 m). A general guideline is that the bench height should be matched to the loading equipment. When using shovels, the bench height should be well within the maximum digging height. For the 9 yd capacity shovel shown in Figure 4.9, it is seen that the maximum cutting height is 43′6″. Hence it could be used with 40 ft benches. A general rule of thumb is that the bench height should not be greater than that of the sheave wheel. Operating in benches with heights greater than this sometimes result in overhangs which endanger the loading and other operations.

Figure 4.10 shows typical reach heights for shovels and front end loaders as a function of bucket size.

At one time, bench heights were limited by drilling depth. Modern drills have largely removed such restrictions. However, in large open pit mines, at least, it is desirable to drill the holes in one pass. This means that the drill must have a mast height sufficient to accommodate the bench height plus the required subdrill.

A deposit of thickness T can be extracted in many ways. Two possibilities are shown in Figure 4.11:

(a) 3 benches of height 50 ft,

(b) 6 benches of height 25 ft.

Higher and wider benches yield:

– less selectivity (mixing of high and low grade and ores of different types);

– more dilution (mixing of waste and ore);

– fewer working places hence less flexibility;

– flatter working slopes; large machines require significant working space to operate efficiently.

On the other hand, such benches provide:

– fewer equipment setups, thus a lower proportion of fixed set up time;

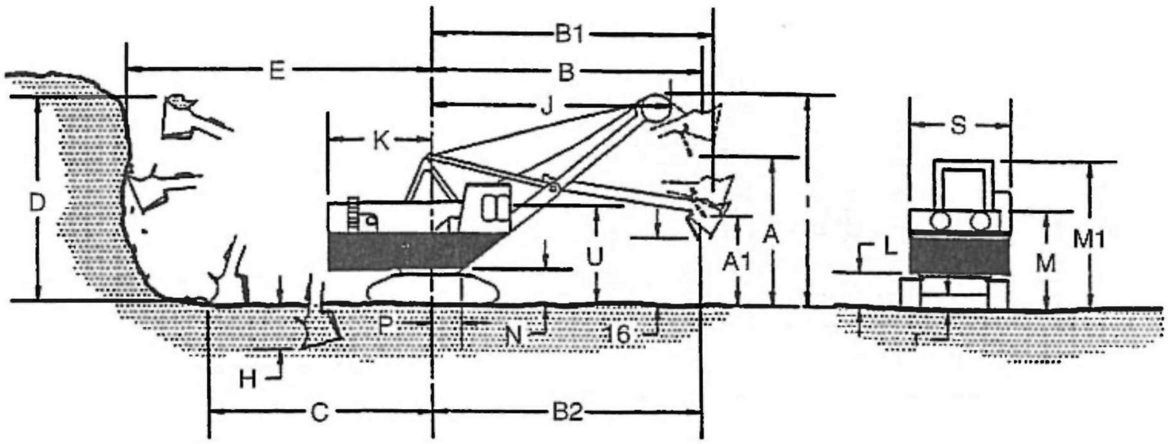
– improved supervision possibilities;

– higher mining momentum; larger blasts mean that more material can be handled at a given time;

– efficiencies and high productivities associated with larger machines.

The steps which are followed when considering bench geometry are:

(1) Deposit characteristics (total tonnage, grade distribution, value, etc.) dictate a certain geometrical approach and production strategy.



Shovel Working Range

Dipper Capacity (Nominal) cu.yds	9
Dipper Capacities (Range) cu.yds	6 ½–6
Length of Boom	41'–6"
Effective length of dipper handle	25'–6"
Overall length of dipper handle	30'–9"

These dimensions will vary slightly depending upon dipper selection.

	Angle of boom	45°	
A	Dumping height – maximum	28'–0"	A
A1	Dumping height at maximum radius – B1	20'–6"	A1
B	Dumping radius at maximum height – A	45'–6"	B
B1	Dumping radius – maximum	47'–6"	B1
B2	Dumping radius at 16'0" dumping height	47–0"	B2
D	Cutting height – maximum	43'–6"	D
E	Cutting radius – maximum	54'–6"	E
G	Radius of level floor	35'–3"	G
H	Digging depth below ground level – maximum	8'-6"	H
I	Clearance height – boom point sheaves	42–3"	I
J	Clearance radius – boom point sheaves	40'–0"	J
K	Clearance radius – revolving frame	19–9"	K
L	Clearance Under frame – to ground	6'-2"	L
M	Clearance height top of house	18'–10"	M
M1	Height of A-frame	31'–2"	M1
N	Height of boom foot above ground level	9–11"	N
P	Distance – boom foot to center of rotation	7–9"	P
S	Overall width of machinery house & operating cab	22'–6"	S
T	Clearance under lowest point in truck frame	14"	T
U	Operator's eye level	18'–0"	U

Figure 4.9. Diagrammatic representation of a 9 yd^3 shovel (Riese, 1993).

(2) The production strategy yields daily ore-waste production rates, selective mining and blending requirements, numbers of working places.

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

(4) Each equipment set has a certain optimum associated geometry.

(5) Each piece of equipment in the set has an associated operating geometry.

(6) A range of suitable bench geometries results.

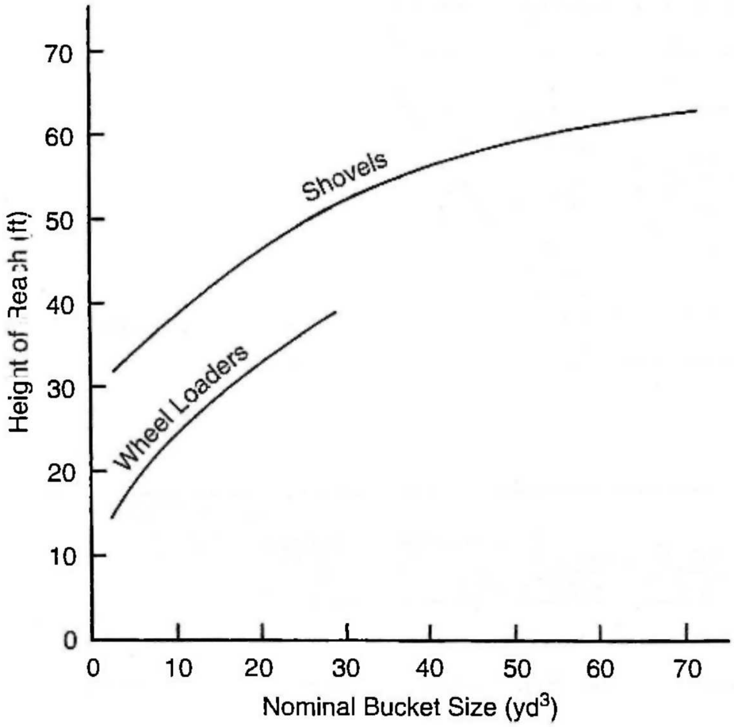


Figure 4.10. Height of reach as a function of bucket size.

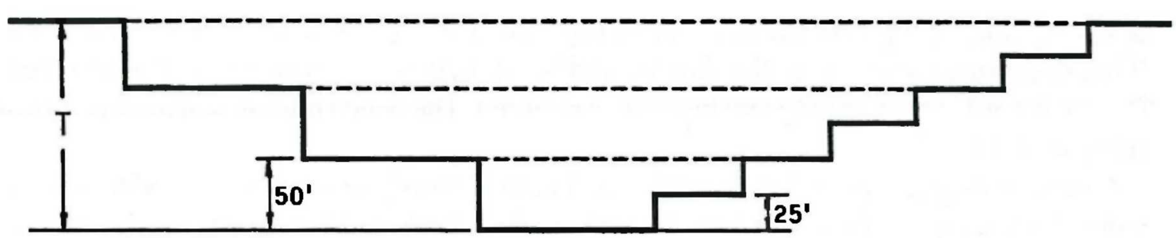


Figure 4.11. Two different bench height scenarios.

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

(8) The 'best' of the various alternatives is selected.

In the past when rail bound equipment was being extensively used, great attention was paid to bench geometry. Today highly mobile rubber tired/ crawler mounted equipment has reduced the detailed evaluation requirements somewhat.

4.3 ORE ACCESS

One of the topics which is little written about in the mining literature is gaining initial physical access to the orebody. How does one actually begin the process of mining? Obviously the approach depends on the topography of the surrounding ground. To introduce the topic it will be assumed that the ground surface is flat. The overlying vegetation has been removed

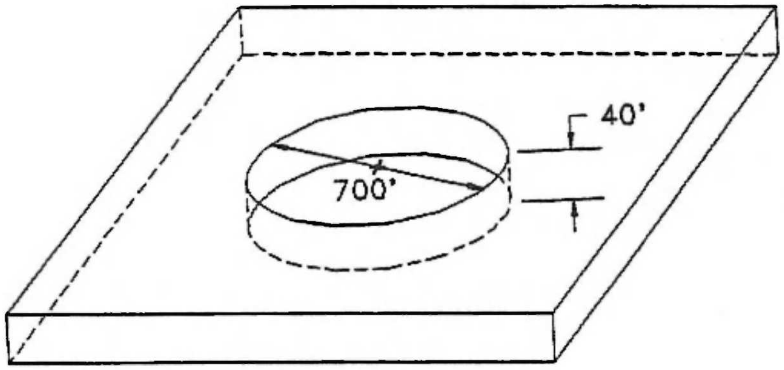


Figure 4.12. Example orebody geometry.

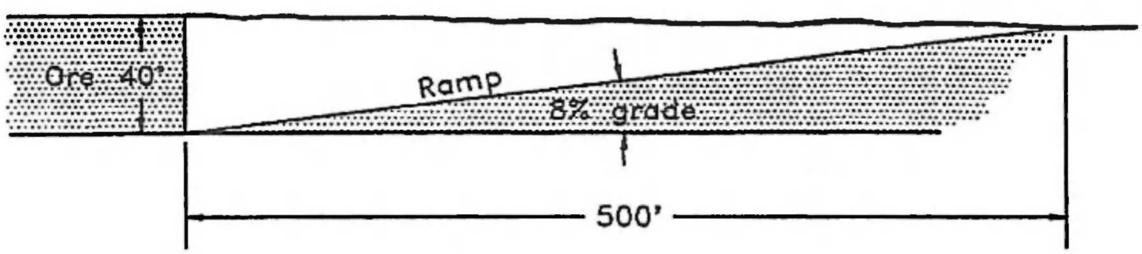


Figure 4.13. Ramp access for the example orebody.

as has the soil/sand/gravel overburden. In this case it will be assumed that the orebody is 700 ft in diameter, 40 ft thick, flat dipping and is exposed by removing the soil overburden. The ore is hard so that drilling and blasting is required. The bench mining situation is shown in Figure 4.12.

A vertical digging face must be established in the orebody before major production can begin. Furthermore a ramp must be created to allow truck and loader access. A drop cut is used to create the vertical breaking face and the ramp access at the same time. Because vertical blastholes are being fired without a vertical free face, the blast conditions are highly constrained. Rock movement is primarily vertically upwards with only very limited sideways motion. To create satisfactory digging conditions the blastholes are normally rather closely spaced. Here only the geometry aspects will be emphasized. To access the orebody, the ramp shown in Figure 4.13 will be driven. It has an 8% grade and a width of 65 ft. Although not generally the case, the walls will be assumed vertical. To reach the 40 ft desired depth the ramp in horizontal projection will be 500 ft in length. There is no general agreement on how the drop cut should be drilled and blasted. Some companies drill the entire cut with holes of the same length. The early part of the ramp then overlies blasted rock while the final portion is at grade. In the design shown in Figure 4.14 the drop cut has been split into three portions. Each is blasted and loaded out before the succeeding one is shot. Rotary drilled holes $9^7/_8''$ in diameter are used. The minimum hole depth is 15 ft. This is maintained over the first 90 ft of the ramp. The hole depth is then maintained at 7 ft below the desired final cut bottom. A staggered pattern of holes is used.

The minimum width of the notch is controlled largely by the dimensions of the loading machine being used. In this example, it will be assumed that the loading machine is the 9 yd^3 capacity shovel shown diagrammatically in Figure 4.9.

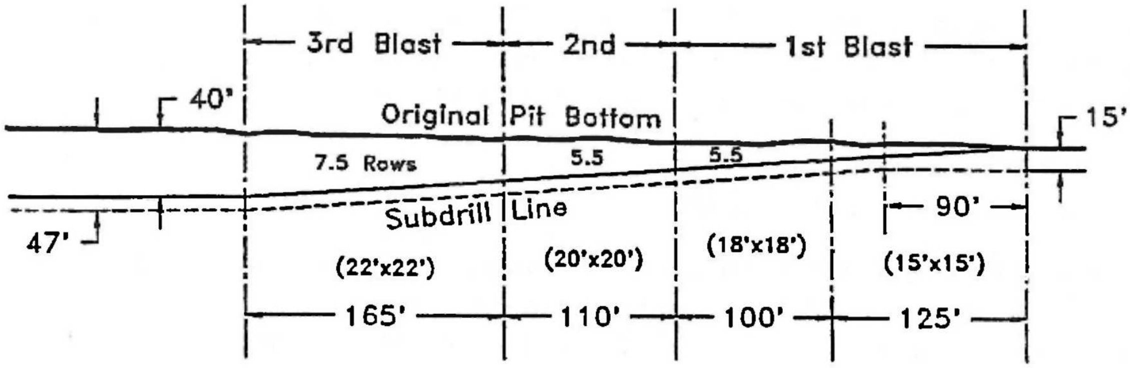


Figure 4.14. Blast design for the ramp excavation.

In the generally tight confines of the drop cut the following shovel dimensions:
K, the clearance radius of the revolving frame,
J, the clearance radius of the boom point sheaves,
G, the maximum digging radius of the level floor, and
E, the maximum cutting radius
are of importance. As can be seen from Figure 4.9, these are:

$$K = 19'9''$$

$$J = 40'0''$$

$$G = 35'3''$$

$$E = 54'6''$$

The minimum width of the drop cut is given by

$$\text{Minimum width} = K + J$$

In this case it is

$$\text{Minimum width} = 19'9'' + 40'0'' = 59'9''$$

This is such that both the front and rear portions of the machine can clear the banks on the two sides as it revolves in the digging and dumping modes.

The maximum digging radius of the level floor is used to indicate the maximum drop cut width for the shovel working along one cutting path. The maximum value is that which the shovel dipper (bucket) can be moved horizontally outward, thereby accomplishing floor cleanup.

The maximum width of the cut at floor level would be

$$\text{Maximum cut width (floor)} = 2 \times 35'3'' = 70'6''$$

The maximum width of the cut at crest level would be

$$\text{Maximum cut width (crest)} = 2 \times 54'6'' = 109'$$

In practice the cutting width for the shovel moving along one path is relatively tightly constrained by the shovel dimensions. In this case:

$$\text{Minimum cut width (crest)} \cong 60\,\text{ft}$$

$$\text{Maximum cut width (floor)} \cong 71\,\text{ft}$$

$$\text{Maximum cut width (crest)} = 109\,\text{ft}$$

For typical cut slope angles of 60 to 80°, the maximum cut width (floor) is the controlling dimension. When the cutting path is down the center of the cut and the shovel is digging to both sides the maximum floor and minimum crest radii would be

$$\text{Maximum floor radius} = 35'3''$$

$$\text{Minimum crest radius} = 40'0''$$

In any case, for laying out the blasting round and evaluating minimum pit bottom dimensions one wants to exceed the minimum working space requirements.

Figures 4.15A through 4.15D show the minimum floor bottom geometry when the shovel moves along the two cutting paths. The loading would first be from one bank. The shovel would then move over and load from the other. This would be considered very tight operating conditions and would be used to create a final cut at the pit bottom.

The usual drop cut is shown in Figures 4.16A through 4.16C where the shovel moves along the cut centerline and can dig to both sides. It will be noted that the shovel must swing through large angles in order to reach the truck.

In both cases the working bench geometry at this stage is characterized by cramped operating conditions.

Two locations for the drop cut/ramp will be considered. The first (case A) is entirely in the waste surrounding the pit. It is desired to have the floor of the ramp at the bottom of ore just as it reaches the ore-waste contact. This is shown diagrammatically in Figure 4.17. The volume of waste rock mined in excavating the ramp is

$$\text{Ramp volume} = \frac{1}{2}H\frac{100H}{g}R_W$$

where R_W is the average ramp width, H is the bench height, and g is the road grade (%). In this case it becomes

$$\text{Ramp volume} = \frac{1}{2}\frac{(40)^2 \times 100 \times 65}{8} = 650,000\,\text{ft}^3$$

This waste must be excavated and paid for before any ore can be removed. However in this arrangement all of the ore can be removed. If it is assumed that the orebody can be extracted with vertical walls, then the ore volume extracted is

$$\text{Ore volume} = \frac{\pi D^2 H}{4} = \frac{\pi}{4}(700)^2 \times 40 = 15,400,000\,\text{ft}^3$$

Upon entering the orebody mining proceeds on an ever expanding front (Fig. 4.18).

As the front expands the number of loading machines which can effectively operate at the same time increases. Hence the production capacity for the level varies with time.

In summary for this ramp placement (case A):

Waste removed (road) = 650,000 ft^3

Ore extracted = 15,400,000 ft^3

% ore extracted = 100%

Another possibility (case B) as is shown in Figure 4.19 is to place the ramp in ore rather than to place the ramp in waste rock. This would be driven as a drop cut in the same way as discussed earlier. The volume excavated is obviously the same as before but now it is ore. Since the material is ore it can be processed and thereby profits are realized earlier.

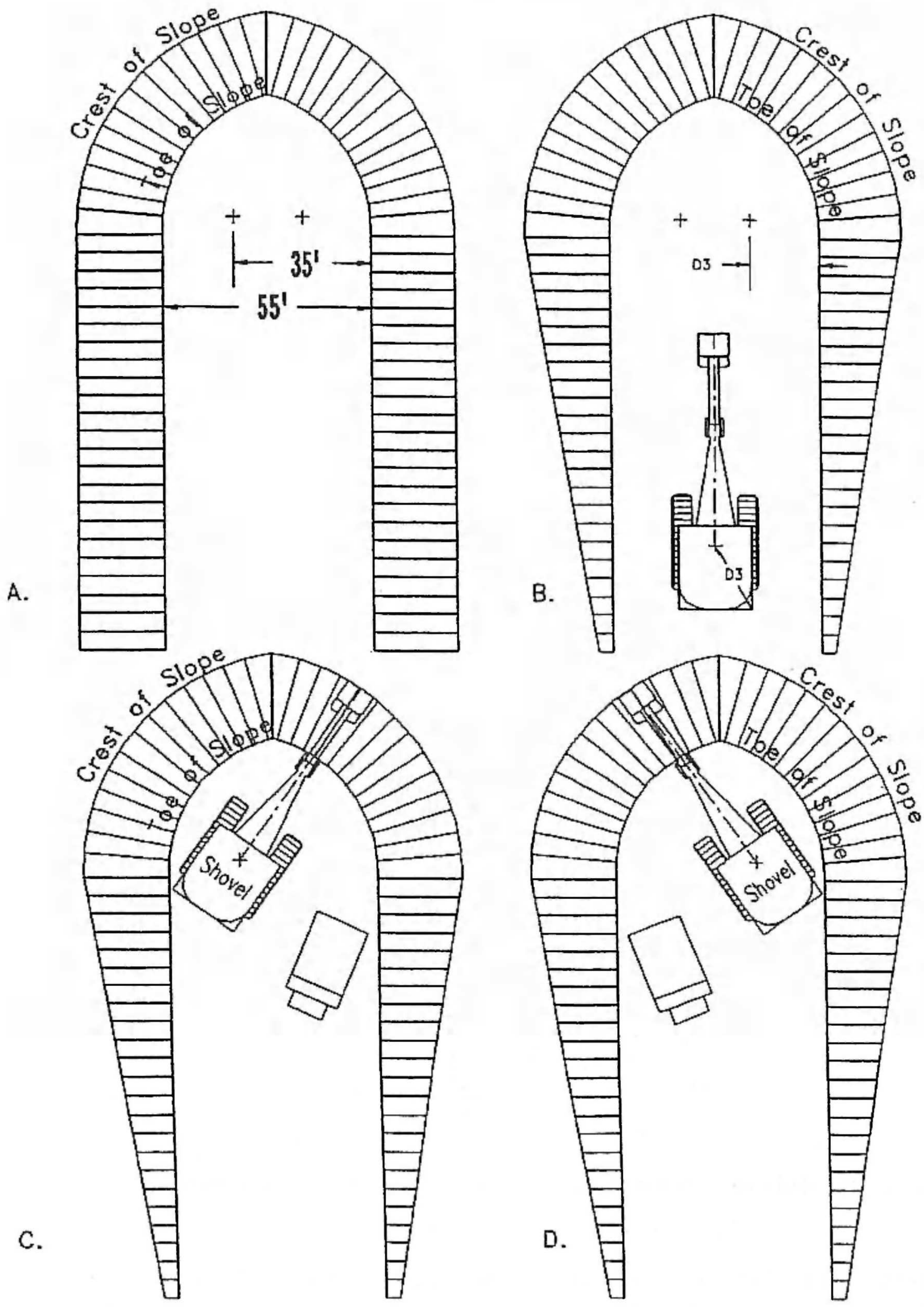


Figure 4.15. Minimum width drop cut geometry with shovel alternating from side to side.

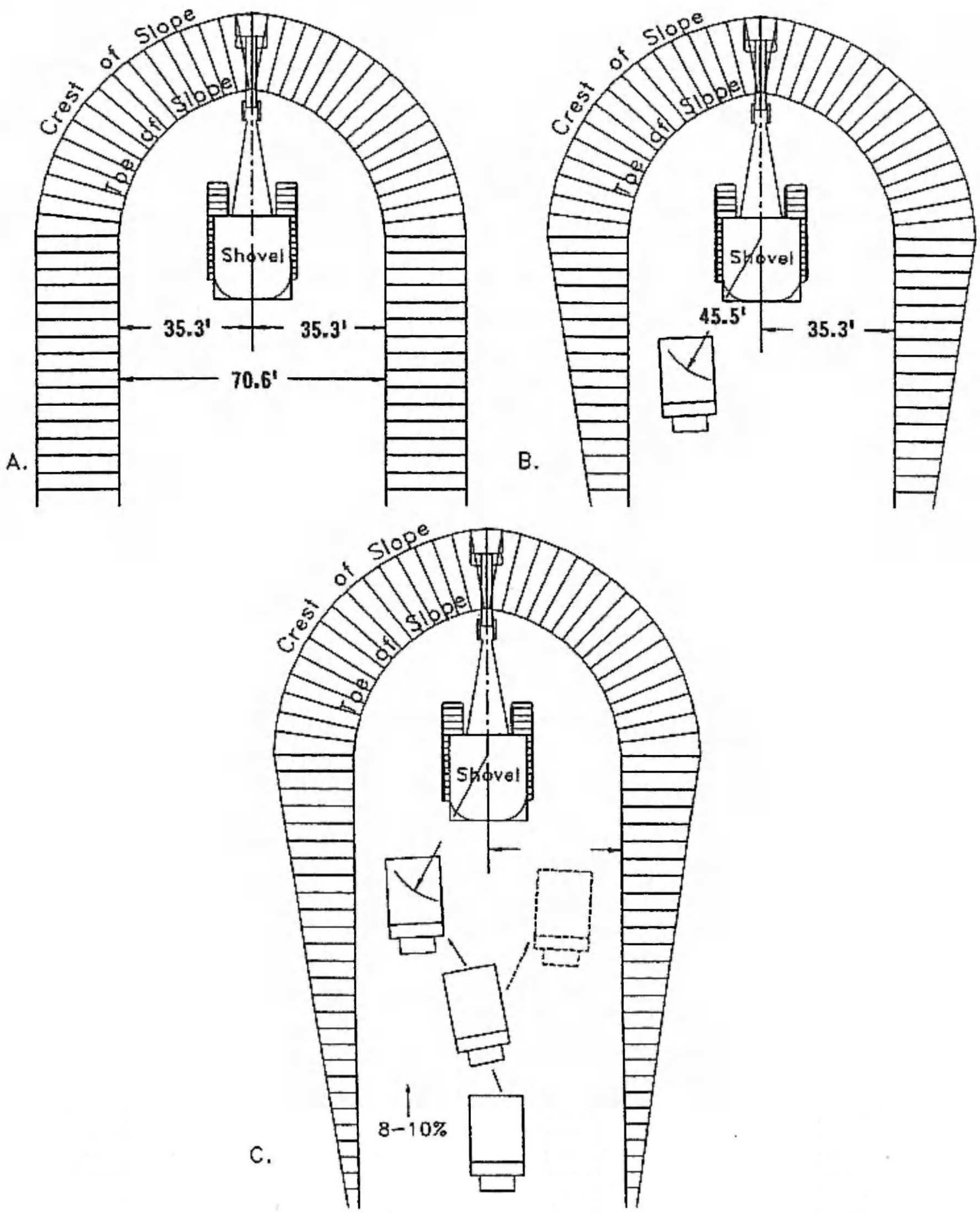


Figure 4.16. Minimum width drop cut geometry with shovel moving along centerline.

From the ramp bottom, the extraction front is gradually increased in length (Fig. 4.20). Obviously the disadvantage is that when mining is completed a quantity of ore remains locked up in the ramp. This quantity is equal to the amount of waste extracted in case A.

Thus the two important points to be made are:

– If the haul road is added external to the planned pit boundaries, then an additional quantity of material equal to the volume of the road must be extracted.

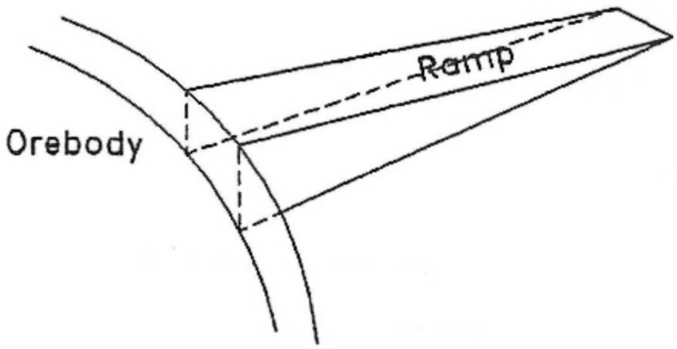


Figure 4.17. Isometric view of the ramp in waste approaching the orebody.

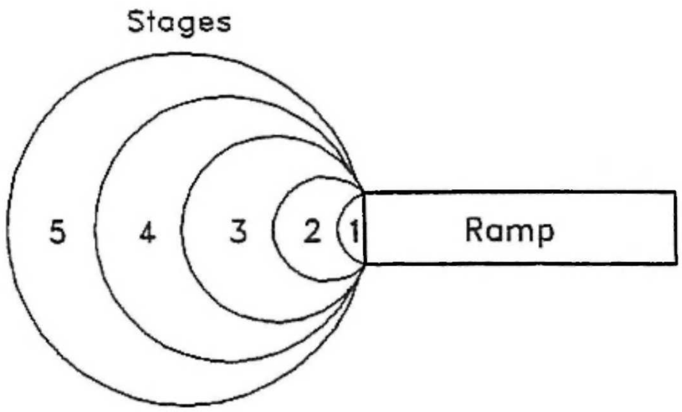


Figure 4.18. Diagrammatic representation of the expanding mining front.

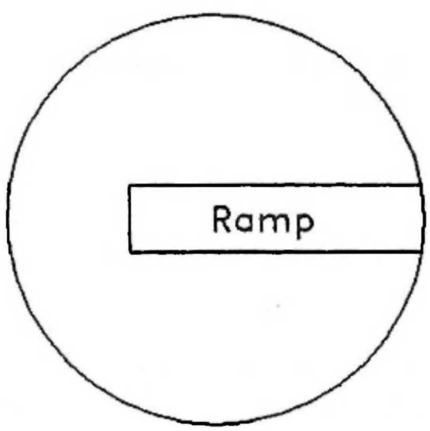


Figure 4.19. Dropcut/ramp placement in ore.

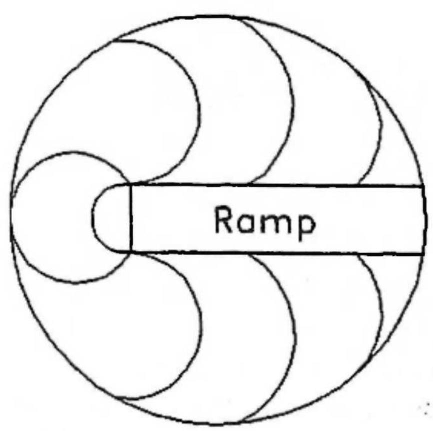


Figure 4.20. Expansion of the mining front.

– If the haul road is added internal to the original planned boundaries, then a quantity of material equal to the road volume must be left in place.

Rather than a straight road such as shown in case A, one might have considered a curved road such as shown in plan in Figure 4.21. With the exception of the final portion, the road is entirely driven in waste. The road could be placed so that the 'ore' left is in the poorest grade.

Assume that the pit is not 1 bench high but instead consists of 2 benches such as is shown in Figure 4.22. The idea is obviously to drive the ramp down to the ore level and establish

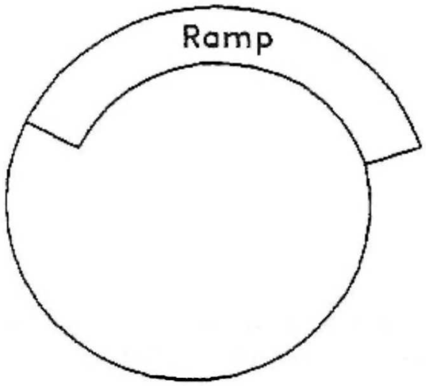


Figure 4.21. Ramp starting in waste and ending in ore.

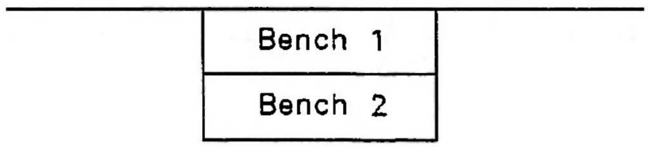


Figure 4.22. Section through a two bench mine.

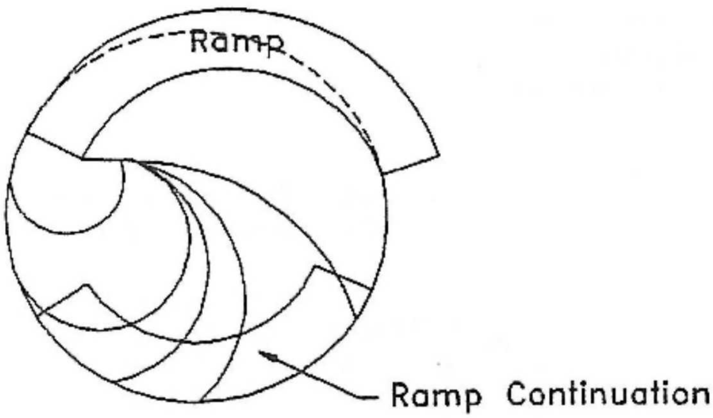


Figure 4.23. Two ramp sections with pit expansion.

the desired production rate. Then while mining is underway on level 1, the ramp would be extended in ore to the lower level as shown in Figure 4.23 through the use of a drop cut. All of the ore lying below the ramp is obviously sterilized. For a multi-bench operation, the procedure continues as shown in Figure 4.24. Note that a flat section having a length of 200 ft has been left in this example between the decline segments. The ramp has a corkscrew shape and the coils get tighter and tighter as the pit is deepened. Rather soon in this example, the pit would reach a final depth simply because the ramp absorbed all of the available working space.

A vertical section taken through the final pit with the orebody superimposed is shown in Figure 4.25. For this particular design where only the initial segment of the ramp is in waste, a large portion of the orebody is sterilized. The amount of waste removed is minimized, however.

An alternative design is one where the ramp is underlain by waste and all of the ore is removed. To make this construction one starts the road design at the lowest bench and works back out. This exercise is left to the reader.

The actual design will generally be somewhere in between these two alternatives with the upper part of the ramp underlain by waste and the lower part by 'ore'.

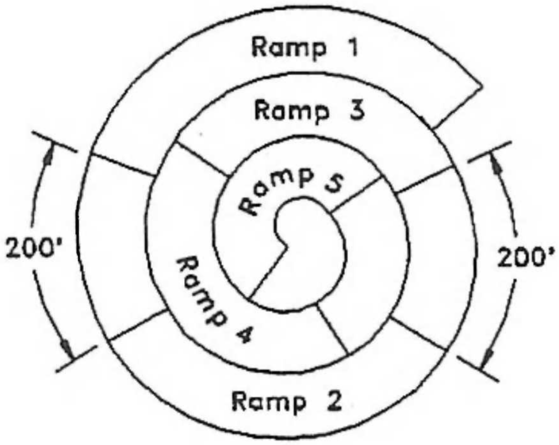


Figure 4.24. Plan view showing ramp locations for a five bench mine.

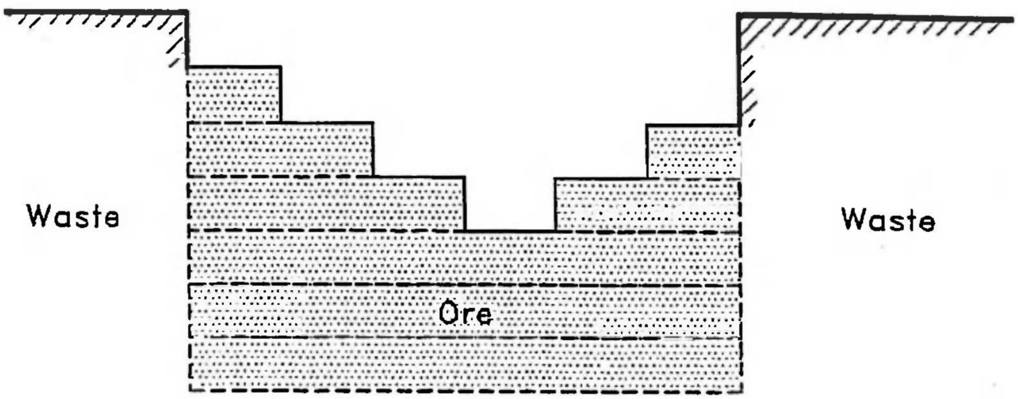


Figure 4.25. Section view showing the sterilization of reserves by ramp.

The excavation may start with attacking the ore first so that the cash flow is improved. Later during the mine life, the waste will be stripped as the main access is gradually moved outward.

In summary:

– there can be considerable volumes associated with the main ramp system;

– the location of the ramp changes with time;

– in the upper levels of the pit, the ramp is underlain by waste; in the lower ranges it is underlain by mineral;

– cash flow considerations are significantly affected by ramp timing;

– the stripping ratio, the percent extraction and the overall extraction are strongly affected simply by the haul road geometry (road width and road grade).

Drop cuts are used on every level to create a new bench. Figures 4.26A through 4.26D show the steps going from the current pit bottom through the mining out of the level. Often the ramp is extended directly off of the current ramp and close to the existing pit wall. This is shown in Figure 4.27. A two level loading operation is shown isometrically in Figure 4.28. The ramp access to both levels in this relatively simple example is easily seen.

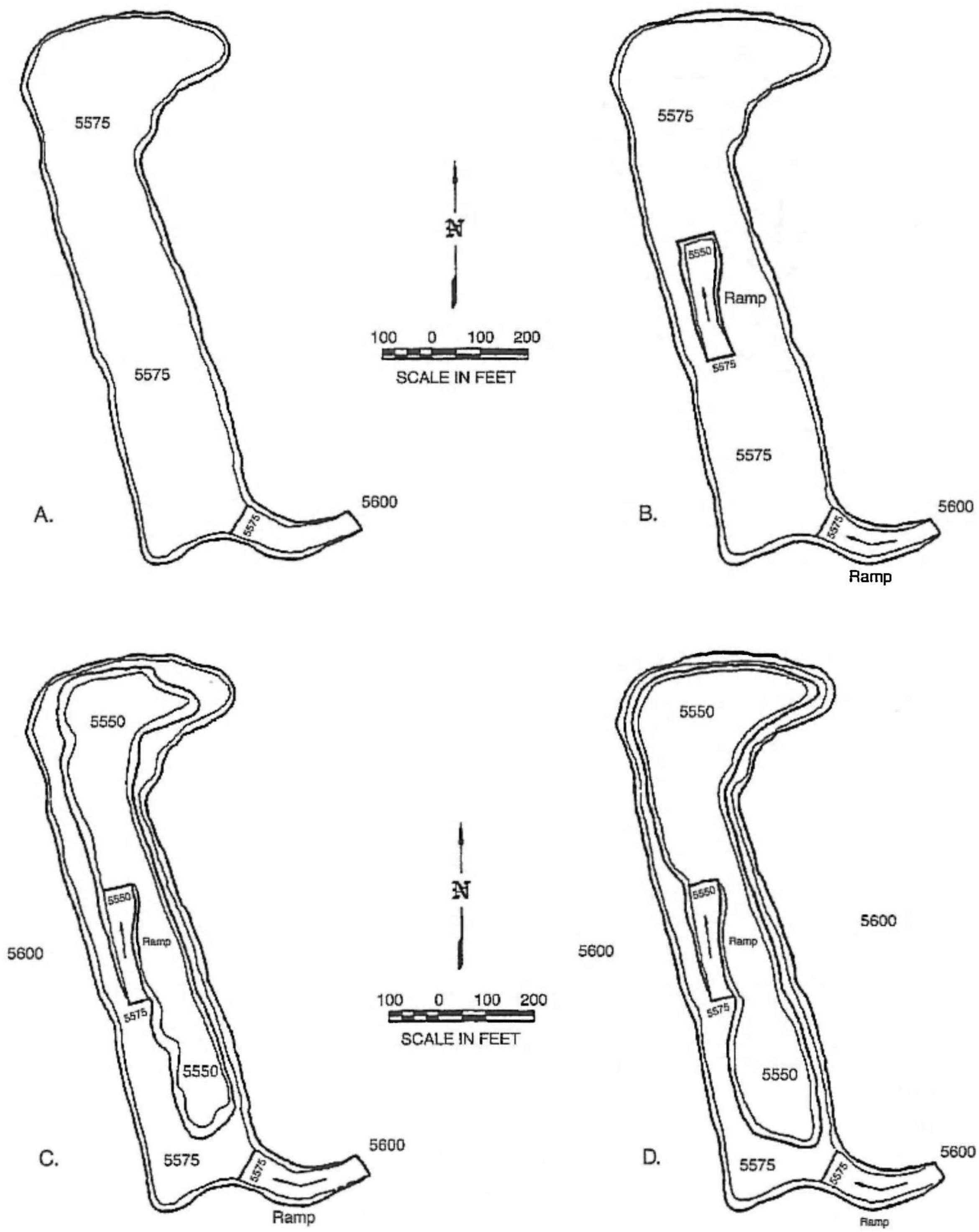


Figure 4.26. Plan view of an actual pit bottom showing drop cut and mining expansion (McWilliams, 1959).

There are many examples where the orebody lies in very rugged terrain. Figure 4.29 shows diagrammatically one possible case. Here the entry to the orebody is made by pushing back the hillside. Bench elevations are first established as shown on the figure. In this case the bench height is 50 ft. Initial benches are established by making pioneering cuts along the surface at convenient bench elevations.

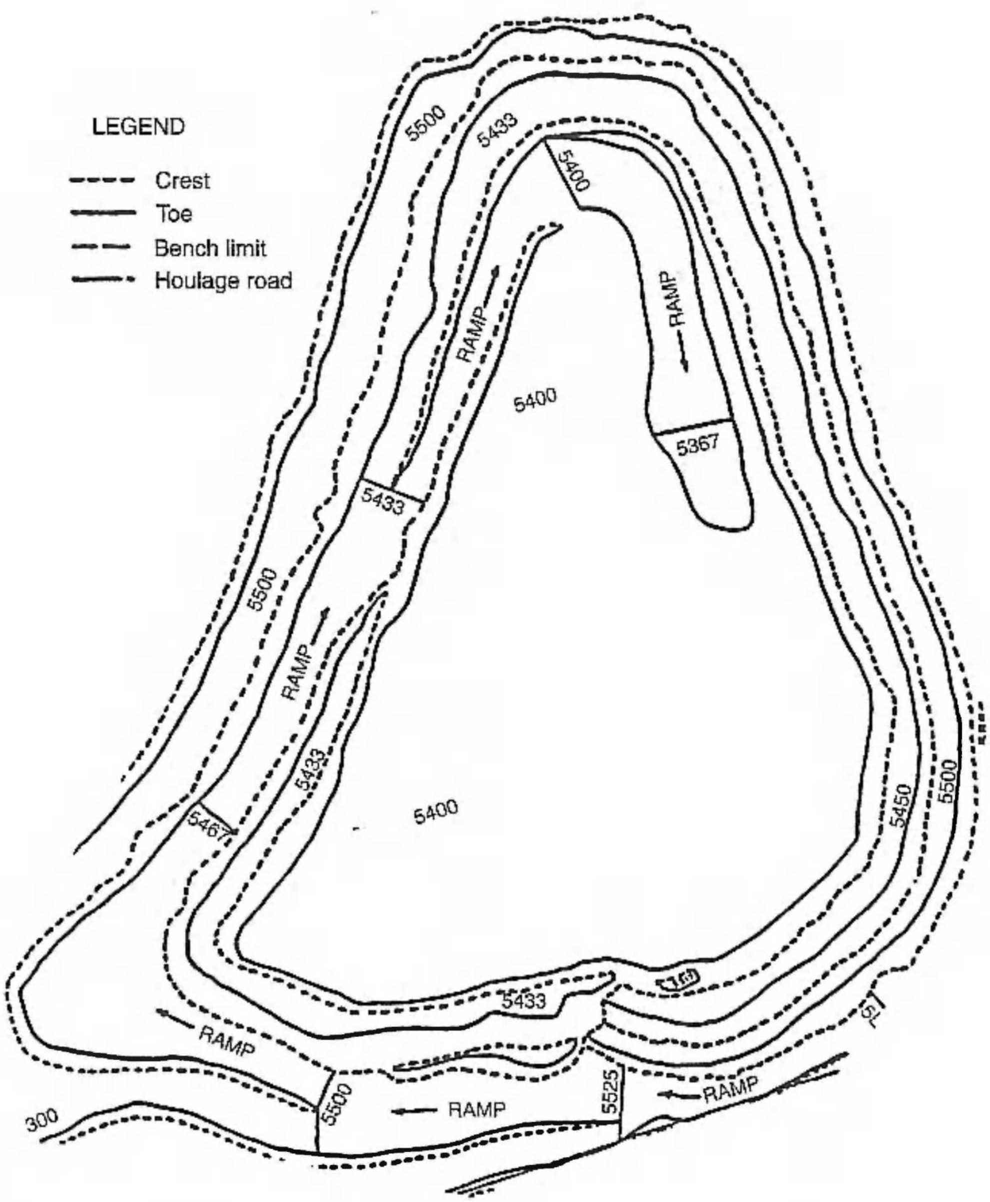


Figure 4.27. Extension of the current ramp close to the pit wall (McWilliams, 1959).

If the slope is composed of softer material, then a dozer can notch it without further assistance (Fig. 4.30). For harder rock types, ripping prior to dozing may be enough. However if the rock is hard or the slope is steep, drilling and blasting will probably be necessary for the pioneer cut. Generally air track types of drills are used. They can reach and drill in very difficult places and can tow their own air compressors/generators.

As shown in Figure 4.31 a shovel can be used instead of a dozer for notching a slope. The notch is enlarged by taking successive cuts until the full bench height is achieved.

Once these initial benches are established, mining of the full faces with vertical blast holes proceeds. Obviously the upper benches have to be advanced before the lower ones.

The final pit outline for this section is shown in Figure 4.33. The reader is encouraged to consider the pit development sequence and the point where drop cuts would be used.

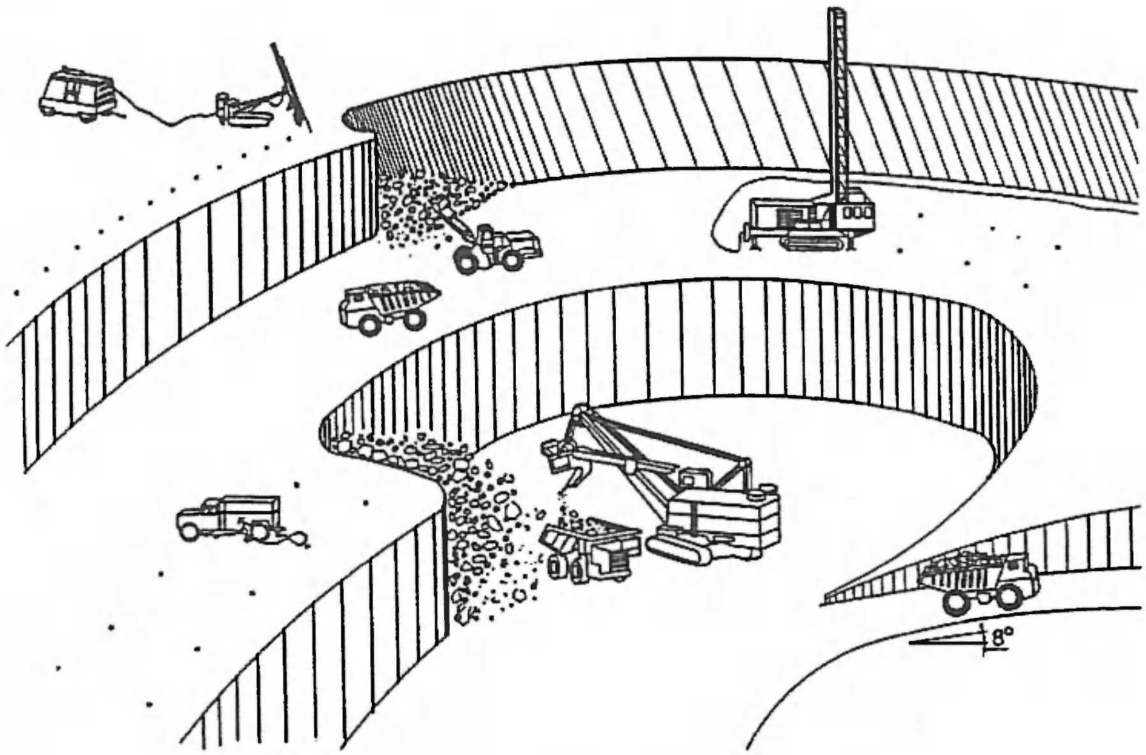


Figure 4.28. Isometric view of simultaneous mining on several levels (Tamrock, 1978).

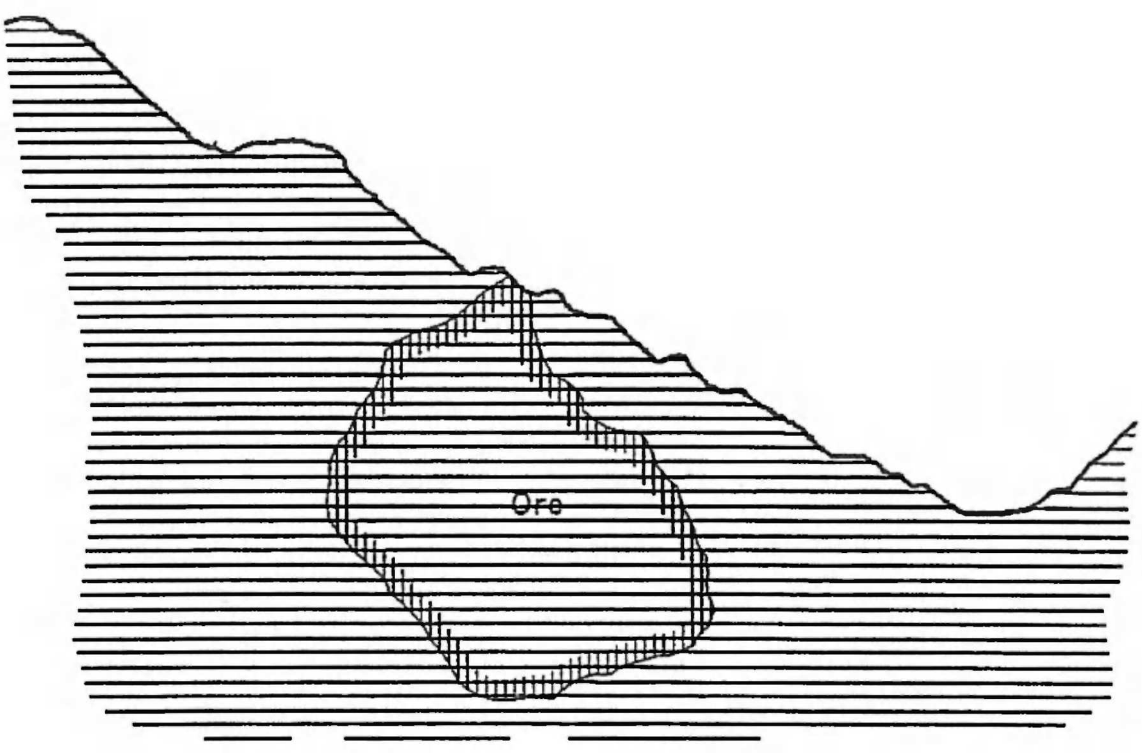


Figure 4.29. Deposit located in mountainous terrain.

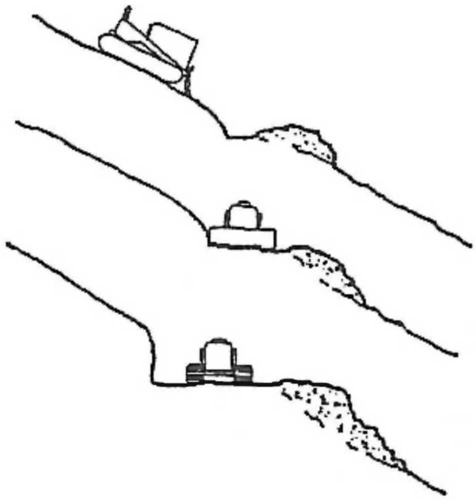


Figure 4.30. Creating initial access/benches (Nichols, 1956).

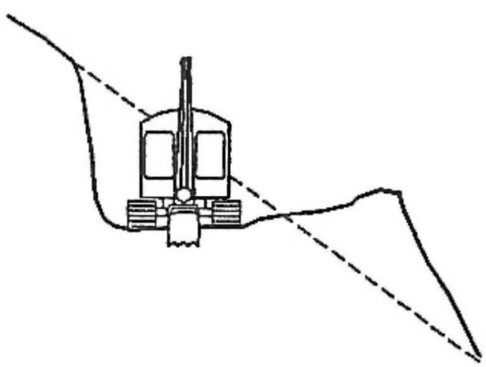


Figure 4.31. Sidehill cut with a shovel (Nichols, 1956).

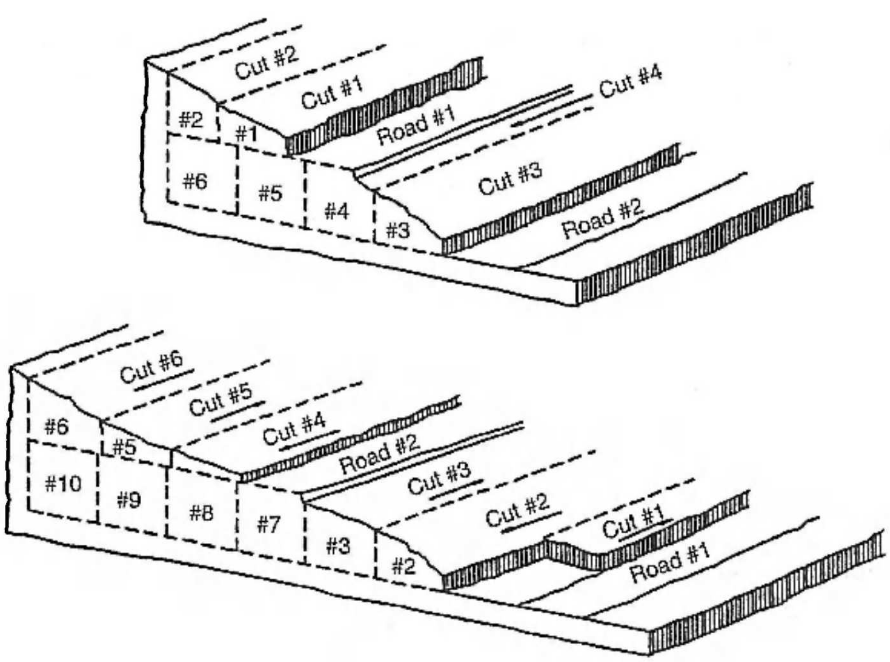


Figure 4.32. Shovel cut sequence when initiating benching in a hilly terrain (Nichols, 1956).

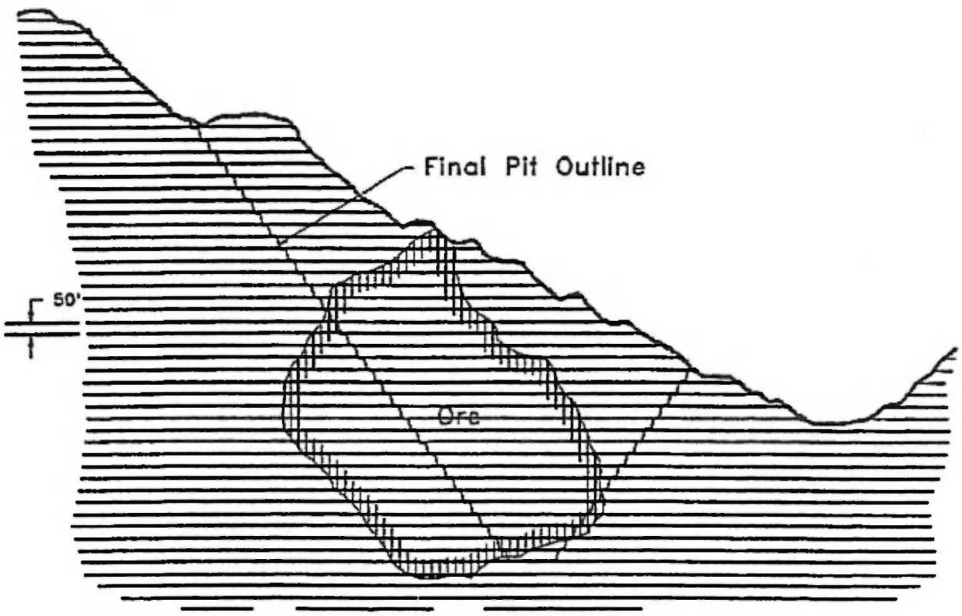


Figure 4.33. Final pit outline superimposed on a section.

4.4 THE PIT EXPANSION PROCESS

4.4.1 *Introduction*

When the drop cut has reached the desired grade, the cut is expanded laterally. Figure 4.34 shows the steps. Initially (Fig. 4.34A) the operating space is very limited. The trucks must turn and stop at the top of the ramp and then back down the ramp towards the loader. When the pit bottom has been expanded sufficiently (Fig. 4.34B), the truck can turn around on the pit bottom. Later as the working area becomes quite large (Fig. 4.34C) several loaders can be used at the same time. The optimum face length assigned to a machine varies with the size and type. It is of the range 200 to 500 ft.

Once access has been established the cut is widened until the entire bench/level has been extended to the bench limits. There are three approaches which will be discussed here:

1. Frontal cuts.
2. Parallel cuts – drive by.
3. Parallel cuts – turn and back.

The first two apply when there is a great deal of working area available, for example at the pit bottom. The mining of more narrow benches on the sides of the pit is covered under number three.

4.4.2 *Frontal cuts*

The frontal cut is shown diagrammatically in Figure 4.35.

The shovel faces the bench face and begins digging forward (straight ahead) and to the side. A niche is cut in the bank wall. For the case shown, double spotting of the trucks is used. The shovel first loads to the left and when the truck is full, he proceeds with the truck on the right. The swing angle varies from a maximum of about 110° to a minimum of 10°. The average swing angle is about 60° hence the loading operation is quite efficient. There

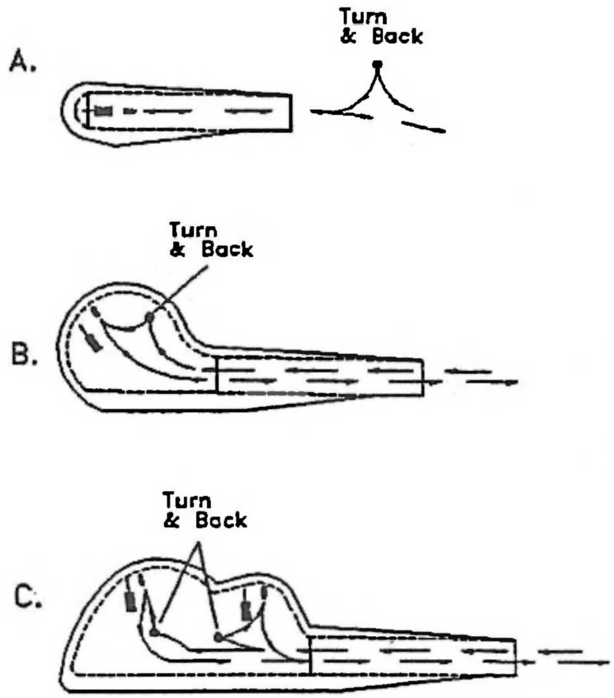


Figure 4.34. Detailed steps in the development of a new production level (Carson, 1961).

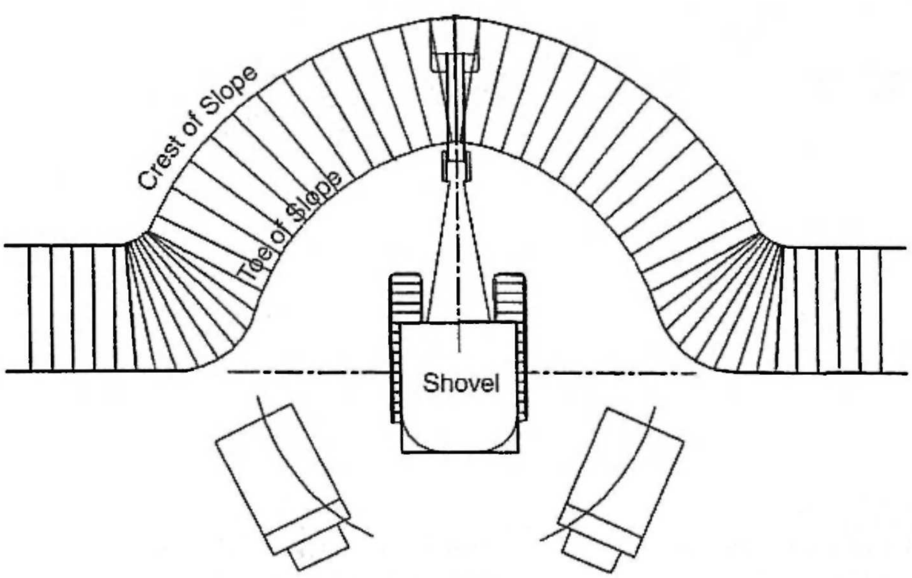


Figure 4.35. Diagrammatic representation of a frontal cutting operation.

must be room for the trucks to position themselves around the shovel. The shovel penetrates to the point that the center of swing is in line with the face. It then moves parallel to itself and takes another frontal cut (Fig. 4.36).

With a long face and sufficient bench width, more than one shovel can work the same face (Fig. 4.37). A fill-in cut is taken between the individual face positions (Fig. 4.38). From the shovels view point this is a highly efficient loading operation. The trucks must however stop and back into position.

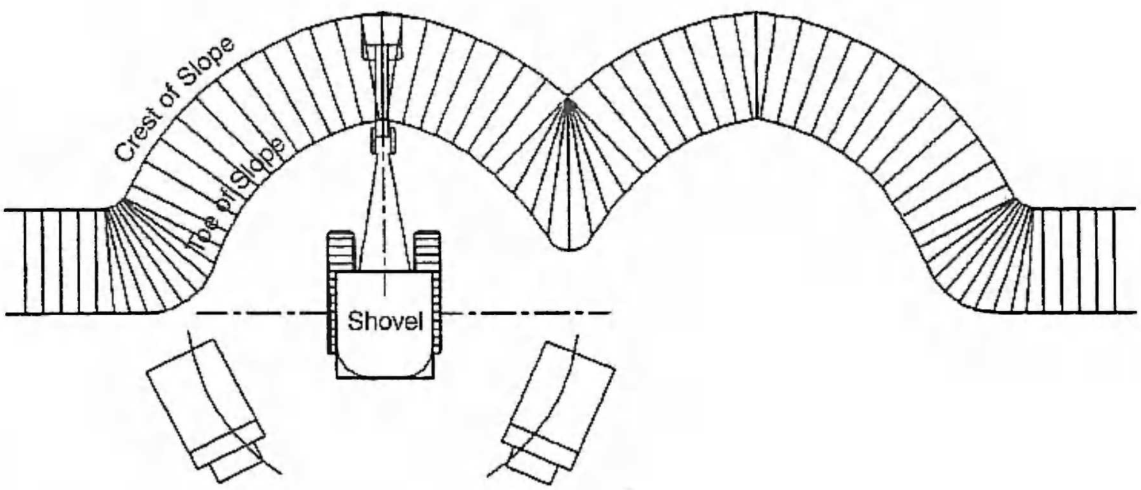


Figure 4.36. Shovel move to adjacent cutting position.

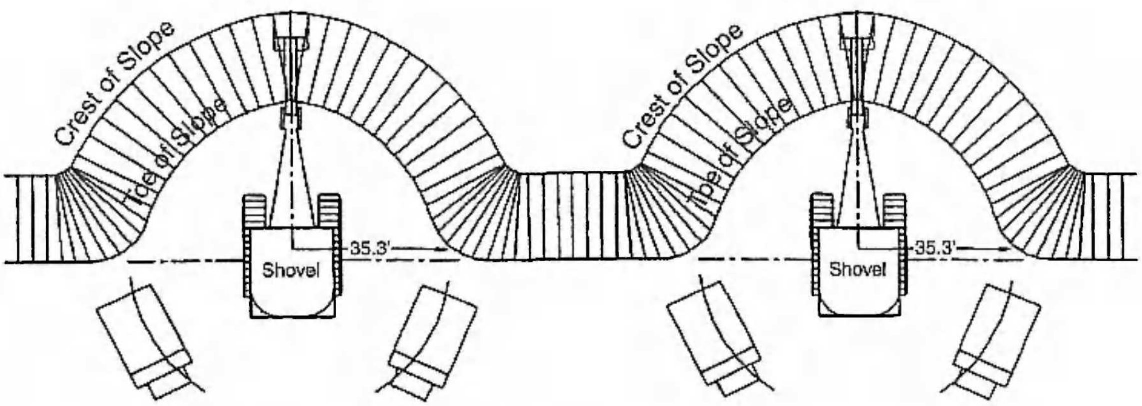


Figure 4.37. Two shovels working the same face.

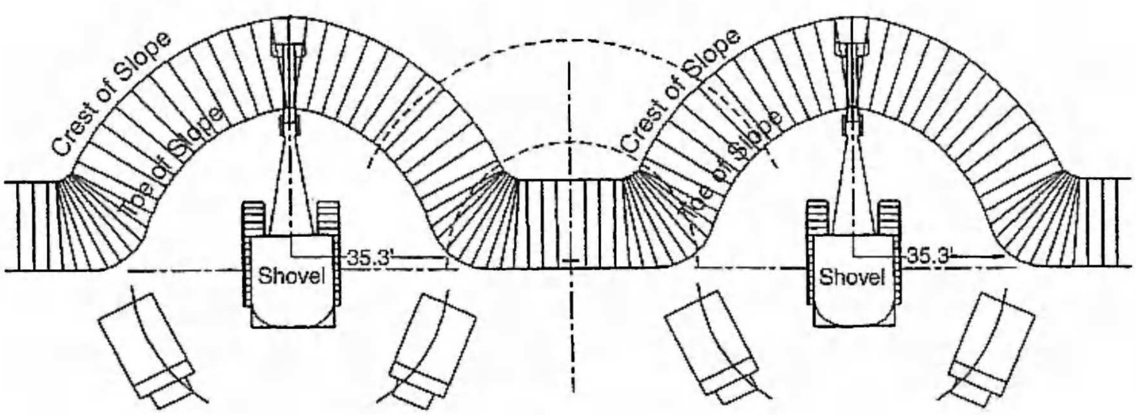


Figure 4.38. Fill-in cutting to complete the face.

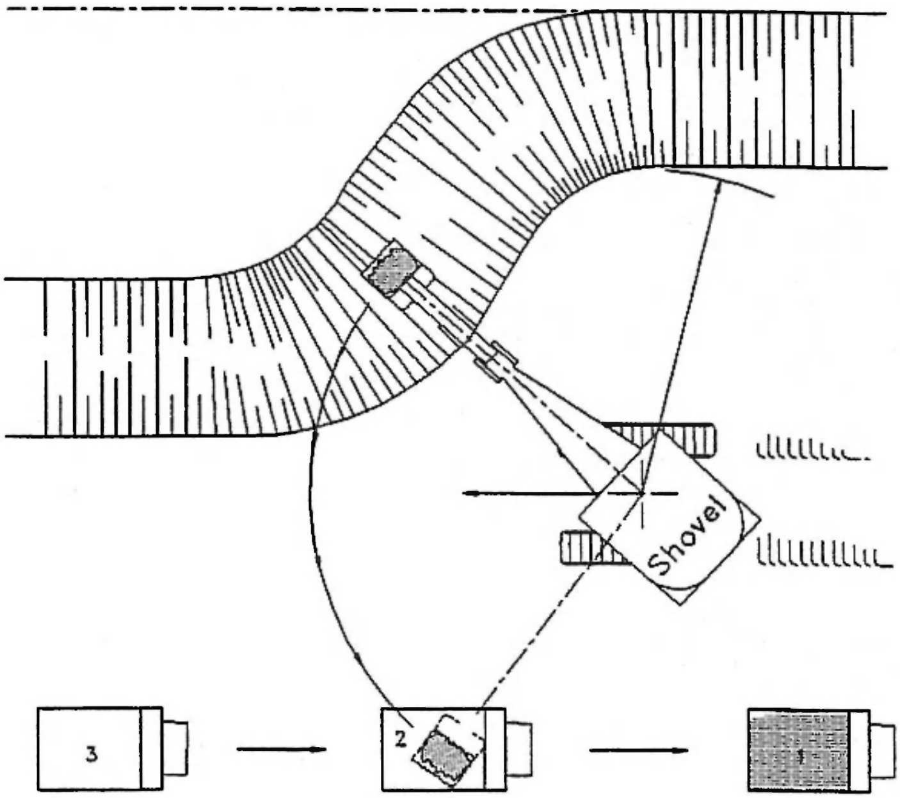


Figure 4.39. Parallel cut with drive-by.

4.4.3 *Drive-by cuts*

Another possibility when the mine geometry allows is the parallel cut with drive-by. This is shown diagrammatically in Figure 4.39. The shovel moves across and parallel to the digging face. For this case, bench access for the haul units must be available from both directions. It is highly efficient for both the trucks and the loader. Although the average swing angle is greater than for the frontal cut, the trucks do not have to back up to the shovel and spotting is simplified.

4.4.4 *Parallel cuts*

The expansion of the pit at the upper levels is generally accomplished using parallel cuts. Due to space limitations there is only access to the ramp from one side of the shovel. This means that the trucks approach the shovel from the rear. They then stop, turn and back into load position. Sometimes there is room for the double spotting of trucks (Fig. 4.40) and sometimes for only single spotting (Fig. 4.41).

Pit geometry is made up of a series of trade-offs. Steeper slopes result in a savings of stripping costs. On the other hand they can, by reducing operating space, produce an increase in operating costs.

Figure 4.42 shows the single spotting sequence. Truck 2 (Fig. 4.42B) waits while the shovel completes the loading of truck 1. After truck 1 has departed (Fig. 4.42C), truck 2 turns and stops (Fig. 4.42D) and backs into position (Fig. 4.42E). While truck 2 is being

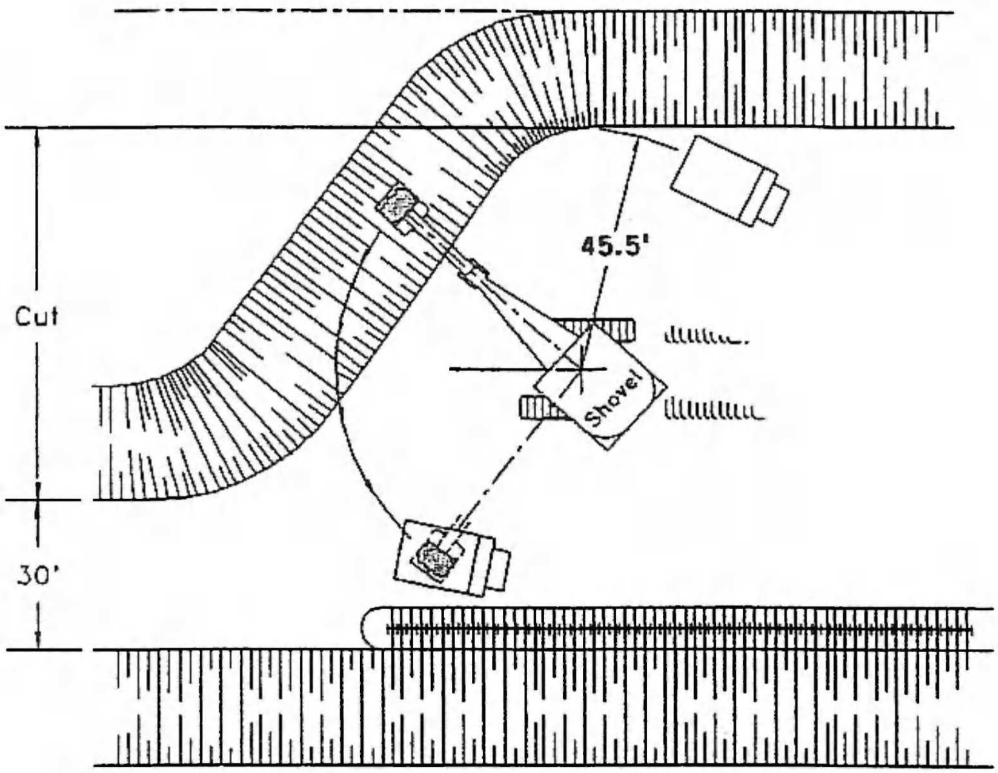


Figure 4.40. Parallel cut with the double spotting of trucks.

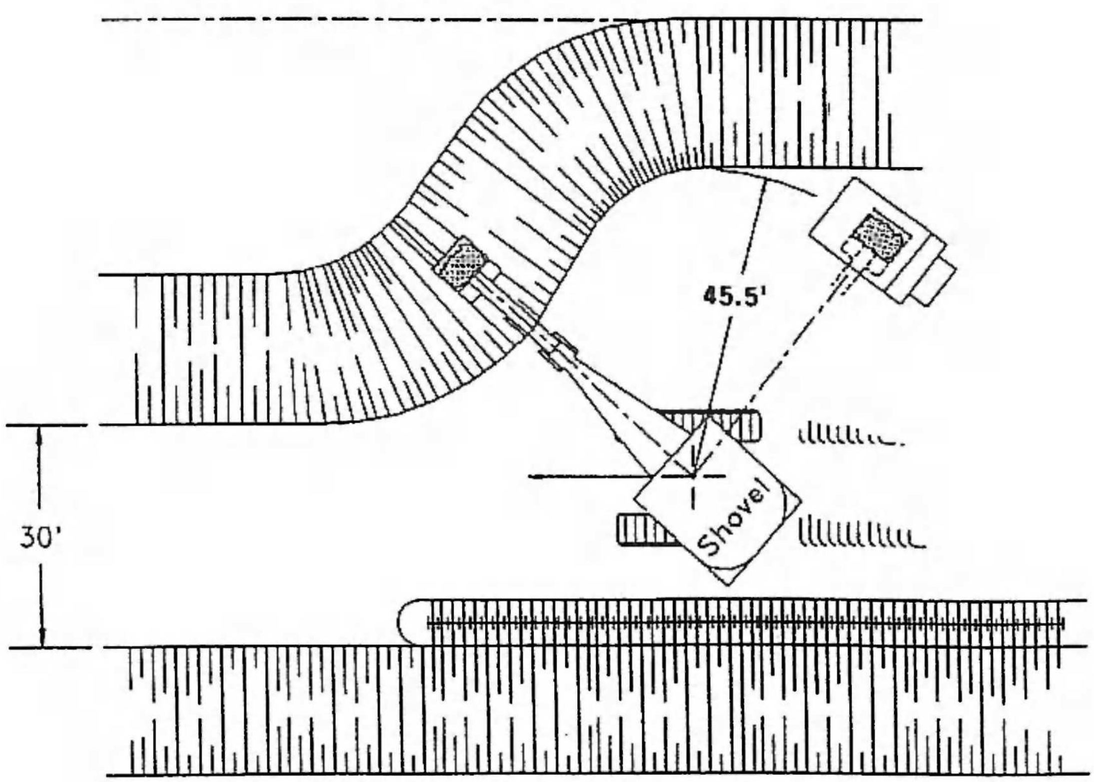


Figure 4.41. Parallel cut with the single spotting of trucks.

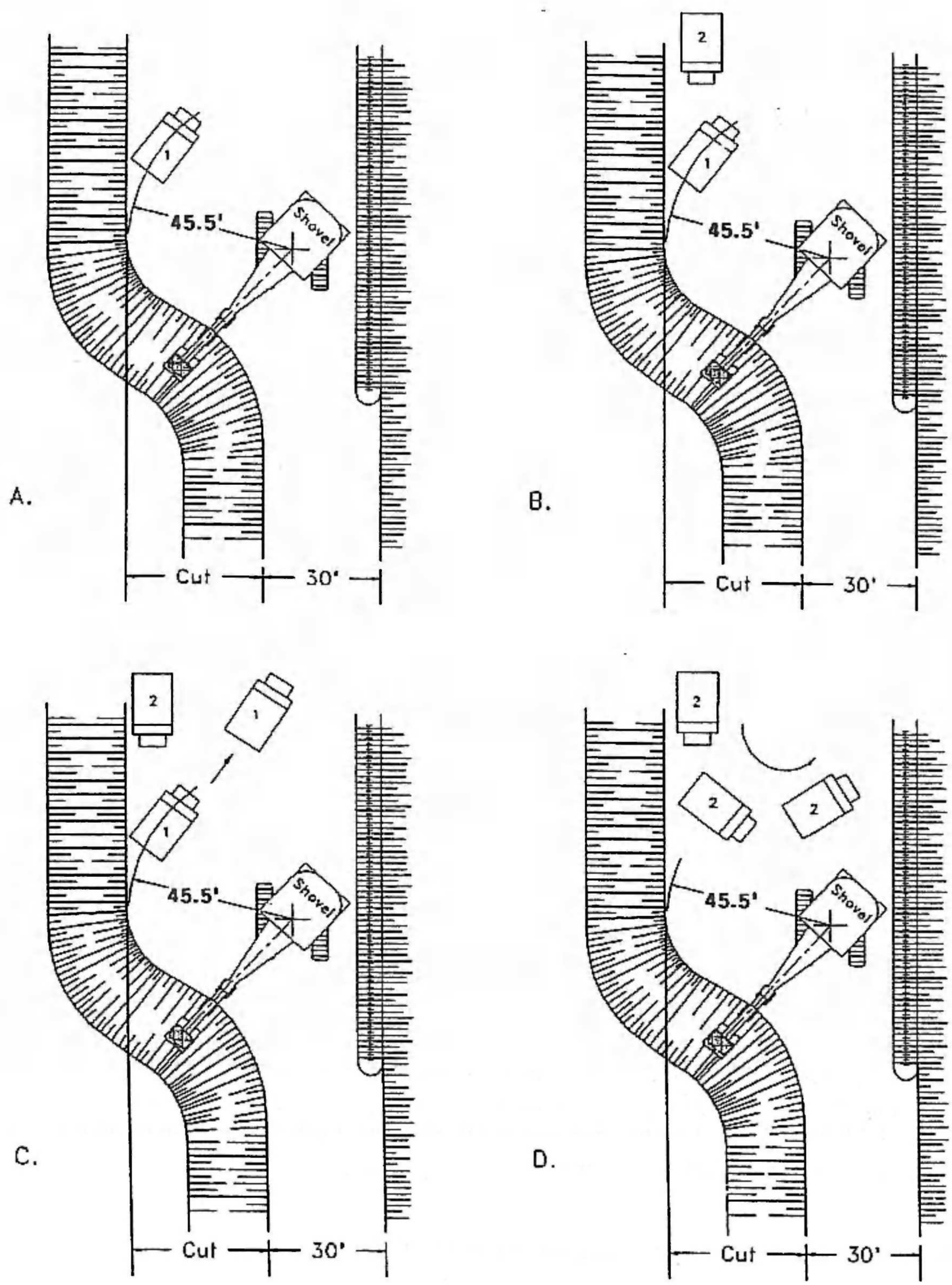
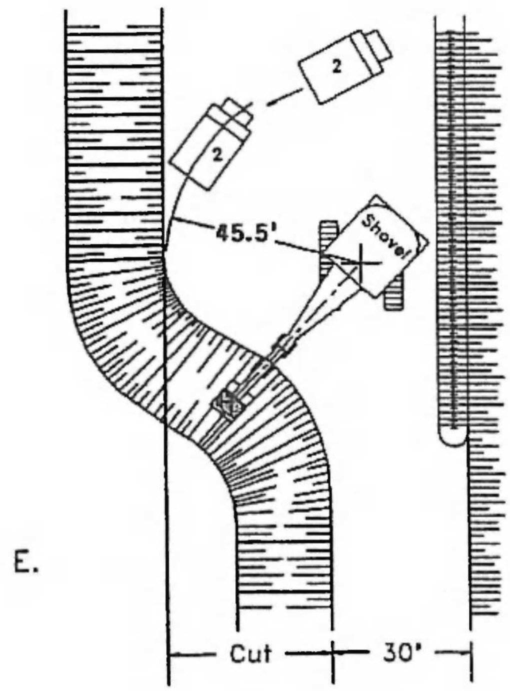


Figure 4.42. Time sequence showing shovel loading with single spotting.



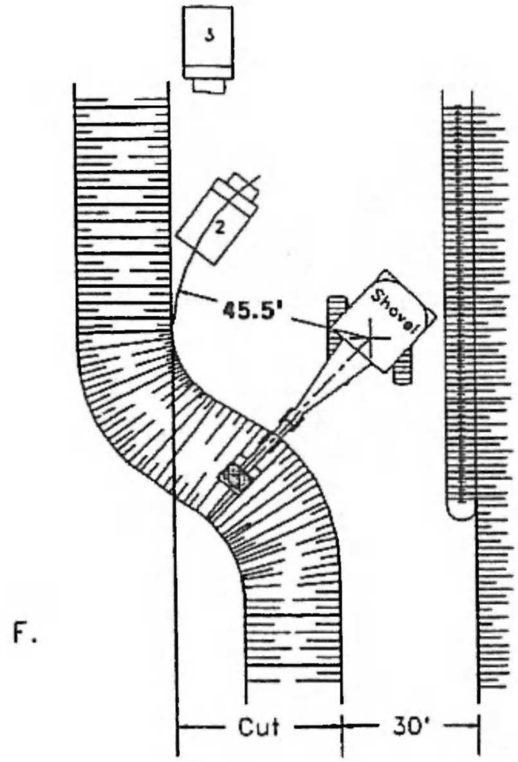


Figure 4.42. (Continued).

loaded truck 3 arrives (Fig. 4.42F). The process then repeats. In this situation both the trucks and the shovel must wait causing a reduction in the overall productivity.

The double spotting situation is shown in Figure 4.43. Truck 1 is first to be loaded (Fig. 4.43A).

Truck 2 arrives (Fig. 4.43B) and backs into position (Fig. 4.43C). When it is just in position the shovel has completed the loading of truck 1. As truck 1 departs (Fig. 4.43D) the shovel begins the loading of truck 2. As truck 2 is being loaded truck 3 arrives. It turns (Fig. 4.43E) and backs into position (Fig. 4.43F). As truck 2 leaves the shovel begins loading truck 3 (Fig. 4.43G). With this type of arrangement there is no waiting by the shovel and less waiting by the trucks. Thus the overall productivity of this system is higher than that for single spotting. The sequencing is unfortunately quite often not as the theory would suggest. Figures 4.43H and 4.43I show two rather typical situations. Both of these can be minimized through the use of an effective communications/dispatching system.

4.4.5 *Minimum required operating room for parallel cuts*

In the previous section the physical process by which a pit is expanded using parallel cuts was described. In this section, the focus will be on determining the amount of operating room required to accommodate the large trucks and shovels involved in the loading operation.

The dimension being sought is the width of the working bench. The working bench is that bench in the process of being mined. This width (which is synonymous with the term 'operating room') is defined as the distance from the crest of the bench providing the floor

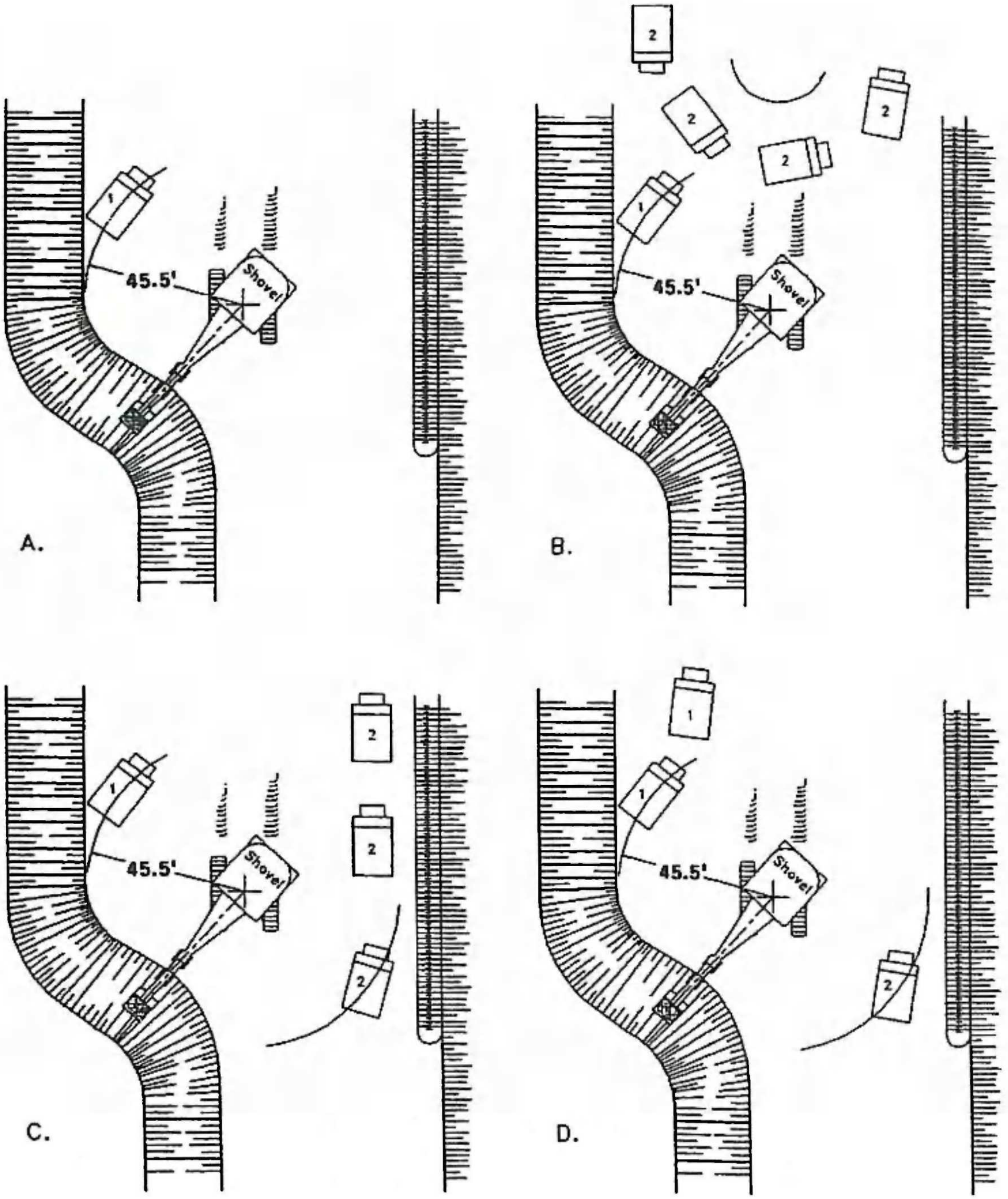
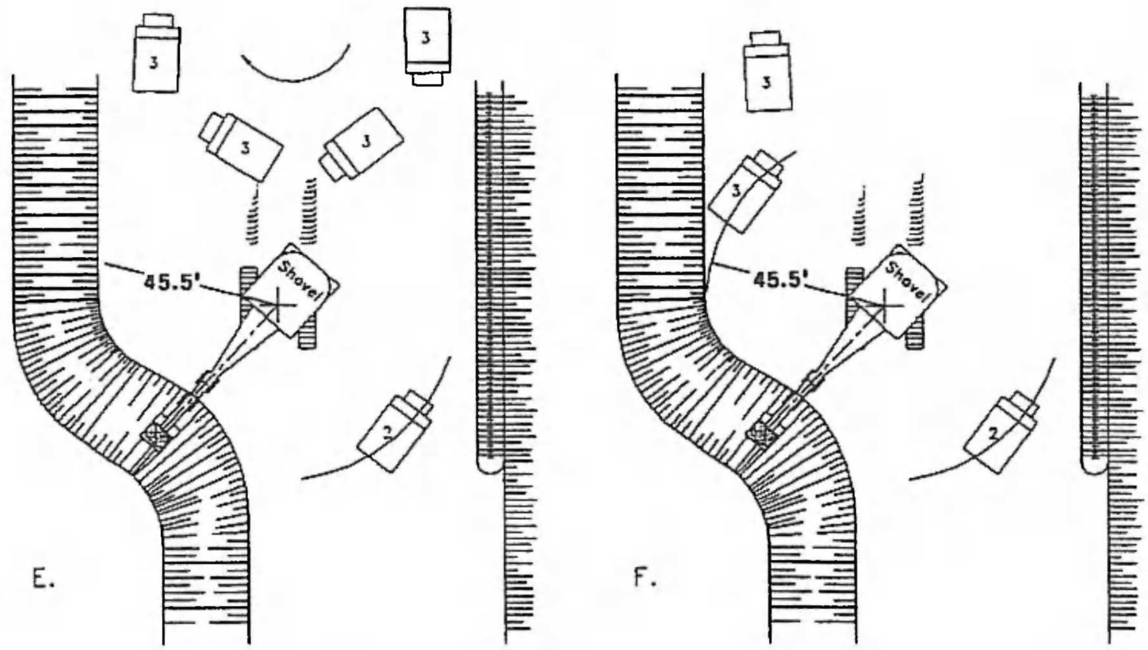


Figure 4.43. Time sequence showing shovel loading with double spotting.

for the loading operations to the bench toe being created as the parallel cut is being advanced. The minimum amount of operating room varies depending upon whether single or double spotting of trucks is used, with the latter obviously requiring somewhat more. The minimum width (W_B) is equal to the width of the minimum required safety bench (S_B) plus the width of the cut (W_C) being taken. This is expressed as

$$W_B = S_B + W_C$$



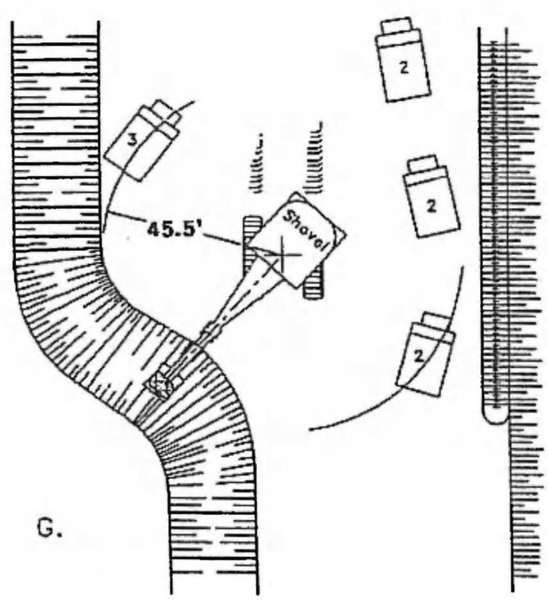


Figure 4.43. (Continued).

The easiest way of demonstrating the principles involved is by way of example. For this, the following assumptions will be made:

- Bench height = 40 ft.
- A safety berm is required.
- The minimum clearance between the outer truck tire and the safety berm = 5 ft.
- Single spotting is used.
- Bench face angle = 70°.
- Loading is done with a 9 yd^3 BE 155 shovel (specifications given in Fig. 4.9).
- Haulage is by 85 ton capacity trucks.

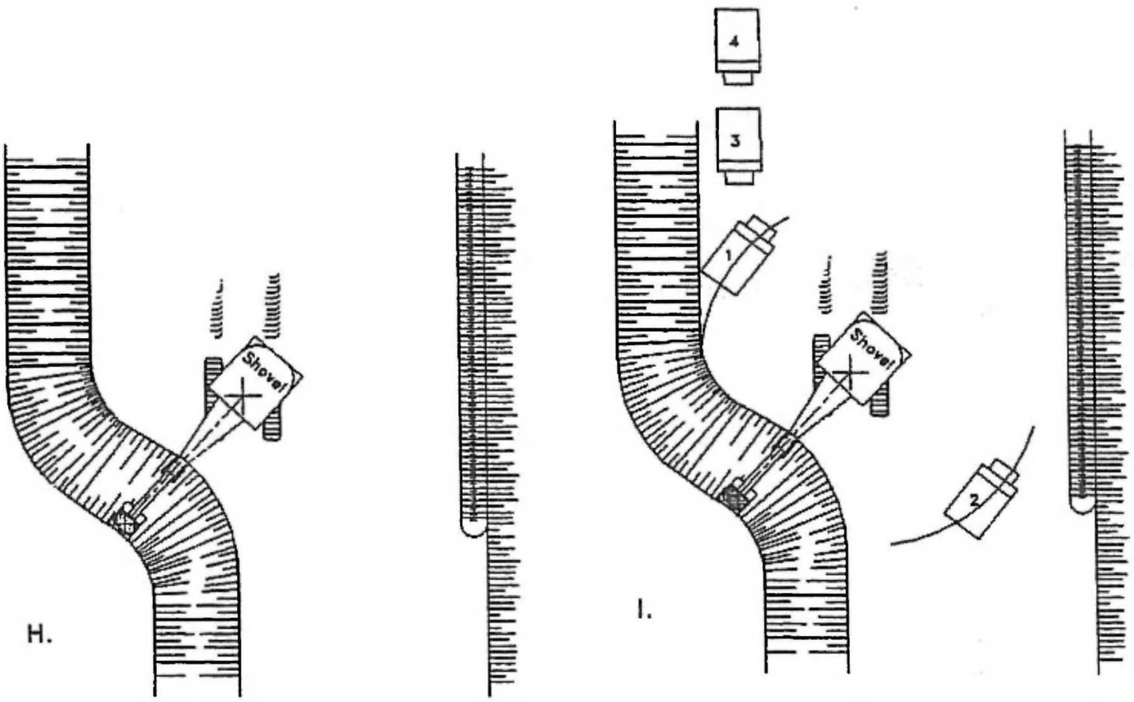


Figure 4.43. (Continued).

- Truck width = 16 ft.
- Tire rolling radius = 4 ft.

The general arrangement in plan and section is shown in Figure 4.44. The design shows that:

$$\text{Working bench width} = 102\,\text{ft}$$
$$\text{Cut width} = 60\,\text{ft}$$
$$\text{Safety bench width} = 42\,\text{ft}$$

The basic calculations (justification) behind these numbers will now be presented.

Step 1. A safety berm is required along the edge of this bench. As will be discussed in Subsection 4.9.5, the height of the berm should be of the order of the tire rolling radius. For this truck, the berm height would be about 4 ft. Assuming that the material has an angle of repose of 45°, the width of the safety berm is 8 ft (see Fig. 4.45). It is assumed that this berm is located with the outer edge at the crest.

Step 2. The distance from the crest to the truck centerline is determined assuming parallel alignment. A 5′ clearance distance between the safety berm and the wheels has been used. Since the truck is 16 ft wide, the centerline to crest distance (T_C) is 21 ft.

Step 3. The appropriate shovel dimensions are read from the specification sheet (Fig. 4.9):

(a) Shovel centerline to truck centerline. This is assumed to be the dumping radius (B) at maximum height,

$$B = 45'6'' = 45.5\,\text{ft}$$

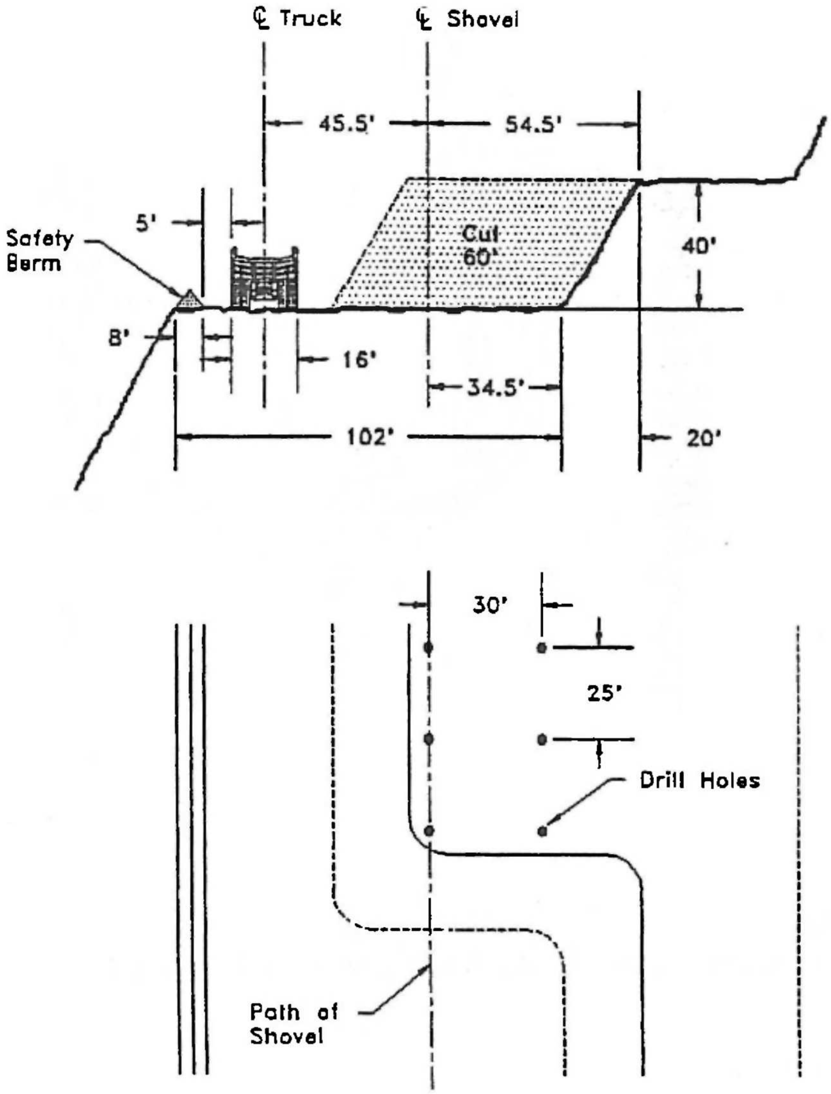


Figure 4.44. Section and plan views through a working bench.

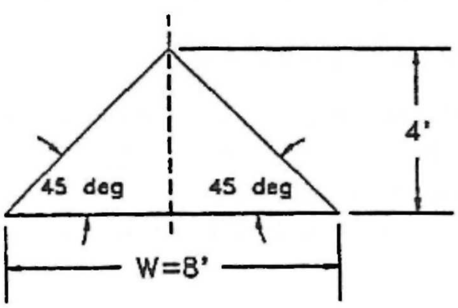


Figure 4.45. Simplified representation of a safety berm.

(b) The maximum dumping height (A) is more than sufficient to clear the truck,

$$A = 28\,\text{ft}$$

(c) The level floor radius dimension (G) is the maximum distance from the shovel centerline which the floor can be cleaned. In this case

$$G = 35'3'' = 35.25\,\text{ft}$$

This will be used as the maximum shovel centerline to toe distance.

Step 4. The desired working bench dimension becomes

$$W_B = T_C + B + G = 21 + 45.5 + 35.25 \doteqdot 102\,\text{ft}$$

Step 5. The corresponding width of cut is now calculated. In this case it has been assumed that the shovel moves along a single path parallel to the crest. Information from the shovel manufacturer suggests that the maximum cutting width (W_C) may be estimated by

$$W_C = 0.90 \times 2 \times G = 0.90 \times 2 \times 35.25 \doteqdot 63.5\,\text{ft}$$

This applies to the width of the pile of broken material. Therefore, to allow for swell and throw of the material during blasting, the design cut width should be less than this value. Here a value of 60 ft has been assumed.

Step 6. Knowing the width of the working bench and the cut width, the resulting safety bench has a width

$$S_B = 102 - 60 = 42\,\text{ft}$$

This is of the order of the bench height (40 ft) which is a rule of thumb sometimes employed.

Step 7. Some check calculations are made with regard to other dimensions.

a) The maximum cutting height of the shovel

$$D = 43'6'' = 43.5\,\text{ft}$$

is greater than the 40′ bench height. Thus the shovel can reach to the top of the bench face for scaling.

b) The maximum shovel cutting radius (E) is

$$E = 54'6'' = 54.5\,\text{ft}$$

Since the maximum radius of the level floor (G) is

$$G = 35'3'' = 35.25\,\text{ft}$$

the flattest bench face angle which could be scaled (Fig. 4.46) is

$$\text{Slope} = \tan^{-1}\frac{40}{54.50 - 35.25} = 64.3°$$

Thus the shovel can easily scale the 70° bench face.

Step 8. The cut dimension should be compared to the drilling and blasting pattern being used. In this particular case the holes are 12¼ ins. in diameter (D_e) and ANFO is the explosive. Using a common rule of thumb, the burden (B) is given by

$$B = 25\frac{D_e}{12} \cong 25\,\text{ft}$$

The hole spacing (S) is equal to the burden

$$S = 25\,\text{ft}$$

Thus two rows of holes are appropriate for this cut width.

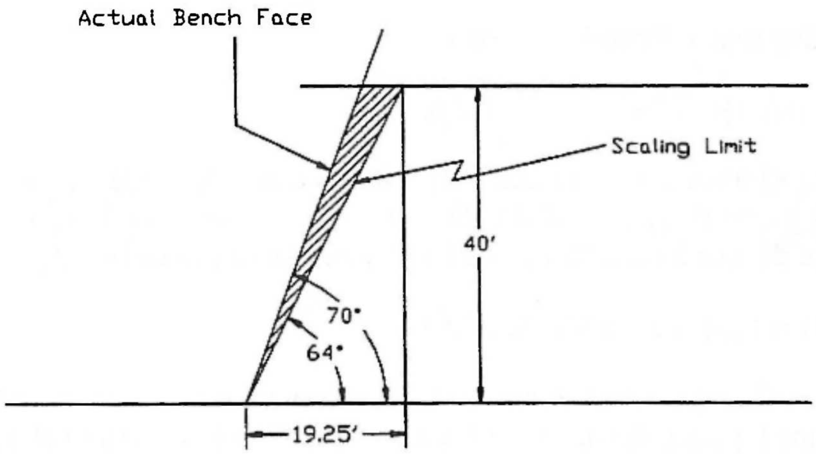


Figure 4.46. The bench/bench face geometry for the example.

A somewhat simplified approach has been applied to the matter of determining working bench width. The complications arise when one examines the best width from an overall economic viewpoint.

As will be discussed in Section 4.5, the working bench is generally one of a set of 3 to 5 benches being mined as a group. The others in the set each have a width equal to that of a safety bench. As the cut is extracted, the remaining portion of the working bench is reduced to a safety bench width. Since the width of the working bench is approximately equal to the combined widths of the others in the set, it has a major impact on the overall slope angle. A wider working bench means that the slope angle is flatter with the extra costs related to earlier/more stripping, but the equipment operating efficiency is higher (with lower related costs). On the other hand a more narrow working bench would provide a steeper overall slope at the cost of operating efficiency. Thus, there are other factors, beside those related to equipment geometries, which must be considered.

4.4.6 *Cut sequencing*

In the previous section the terms 'working bench,' 'cut' and 'safety bench' were introduced. These will now be applied to a simple example in which a 90 ft wide cut 1000 ft long will be taken from the right hand wall of the pit shown in Figure 4.47. As can be seen the wall consists of 4 benches. The entire bench 1 (B1) is exposed at the surface. Benches B2, B3 and B4 are safety benches, 35 ft wide. The process begins with the drill working off the upper surface of B1. The holes forming the cut to be taken from B1 are drilled and blasted (Fig. 4.48). The shovel then moves along the floor of bench B1 (upper surface of B2) and loads the trucks which also travel on this surface. The working bench has a width of 125 ft. When the cut is completed the geometry is as shown in Figure 4.49. The cut to be taken from bench 2 is now drilled and blasted. The shovel moves along the top of bench 3 taking a cut width (W_C). A portion of bench 2 remains as a safety bench. The process is repeated until the bottom of the pit is reached. The shovel then moves back up to bench 1 and the process is repeated. If it is assumed that the shovel can produce 10,000 tons/day, then the overall production from these 4 benches is 10,000 tons/day. The four benches associated with this shovel are referred to as a mine production unit.

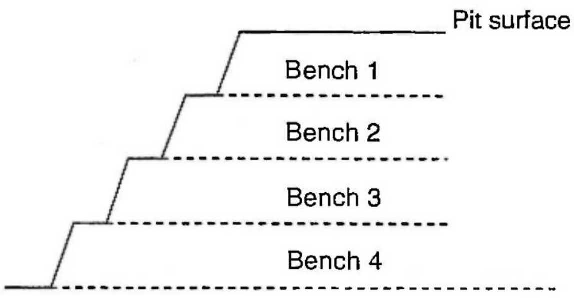


Figure 4.47. Initial geometry for the push back example.

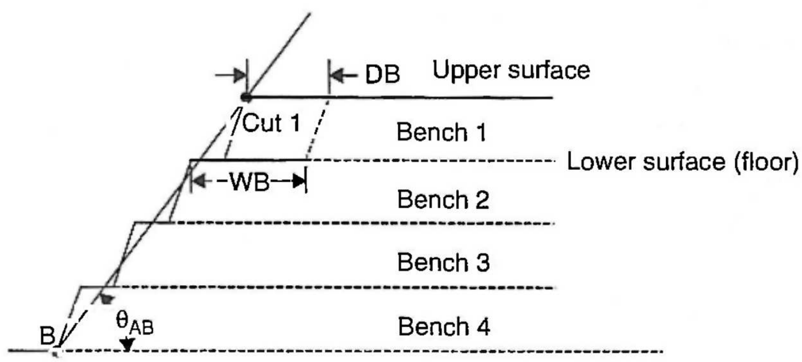


Figure 4.48. Cut mining from bench 1.

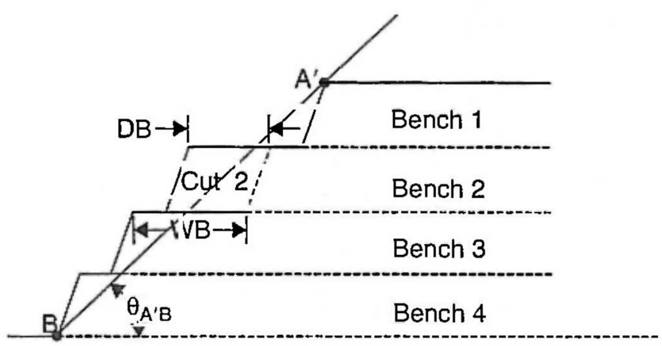


Figure 4.49. Cut mining from bench 2.

4.5 PIT SLOPE GEOMETRY

There are a number of 'slopes' which enter into pit design. Care is needed so that there is no confusion as to how they are calculated and what they mean. One slope has already been introduced. That is the bench face angle (Fig. 4.50). It is defined as the angle made with the horizontal of the line connecting the toe to the crest. This definition of the slope going from the toe to the crest will be maintained throughout this book.

Now consider the slope consisting of 5 such benches (Fig. 4.51). The angle made with the horizontal of the line connecting the lowest most toe to the upper most crest is defined as the overall pit slope,

$$\Theta(\text{overall}) = \tan^{-1}\frac{5\times 50}{4\times 35+\frac{5\times 50}{\tan 75^\circ}} = 50.4^\circ$$

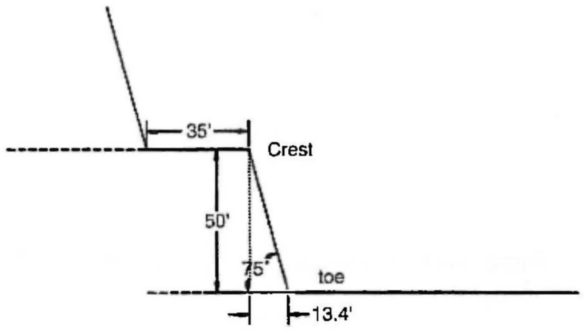


Figure 4.50. Safety bench geometry showing bench face angle.

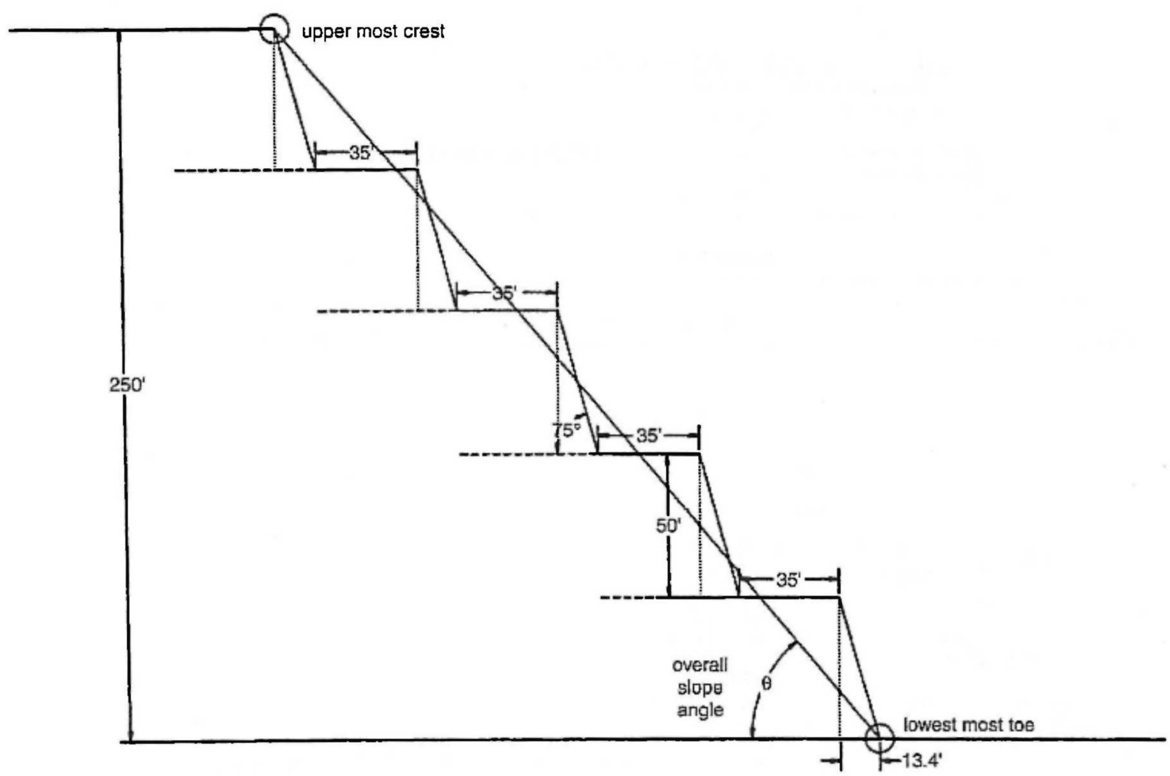


Figure 4.51. Overall slope angle.

If as is shown in Figure 4.52 an access ramp with a width of 100 ft is located half way up bench 3, the overall pit slope becomes

$$\Theta(\text{overall}) = \tan^{-1}\frac{5 \times 50}{4 \times 35 + \frac{5\times 50}{\tan 75^\circ} + 100} = 39.2^\circ$$

As can be seen, the presence of the ramp on a given section has an enormous impact on the overall slope angle.

The ramp breaks the overall slope into two portions (Fig. 4.53) which can each be described by slope angles. These angles are called interramp angles (between-the-ramp angles). In this case

$$\mathrm{IR}_1 = \mathrm{IR}_2 = \tan^{-1}\frac{125}{2 \times 35 + \frac{2\times 50}{\tan 75^\circ} + \frac{25}{\tan 75^\circ}} = 50.4^\circ$$

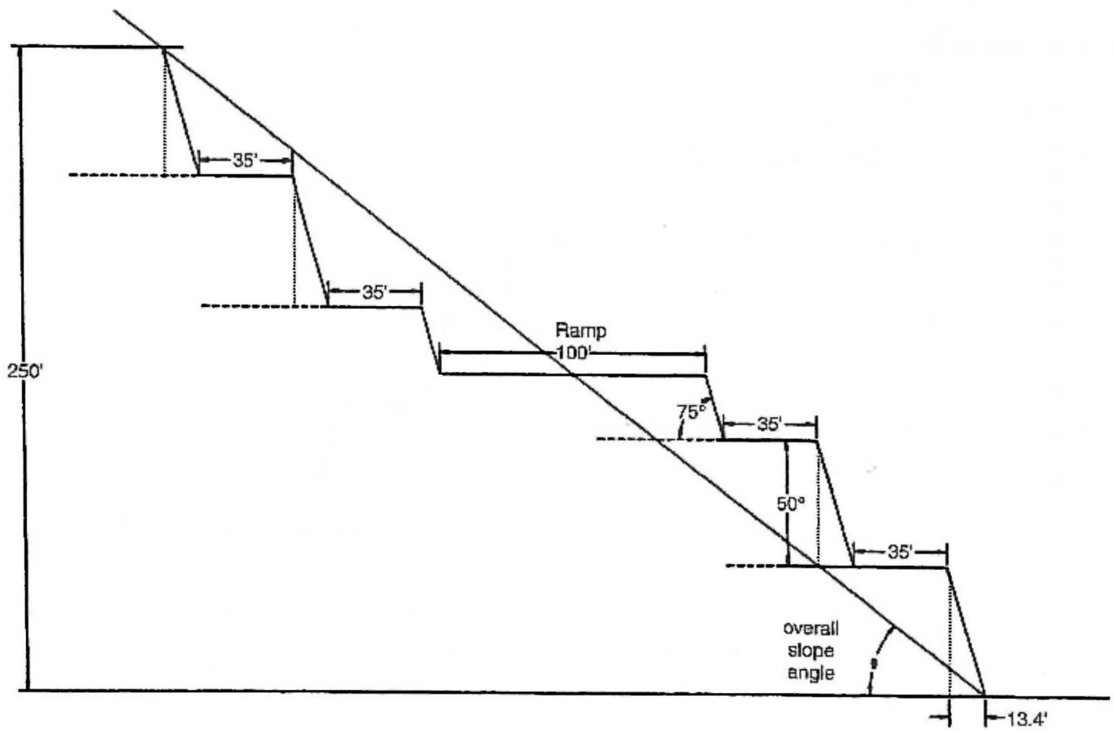


Figure 4.52. Overall slope angle with ramp included.

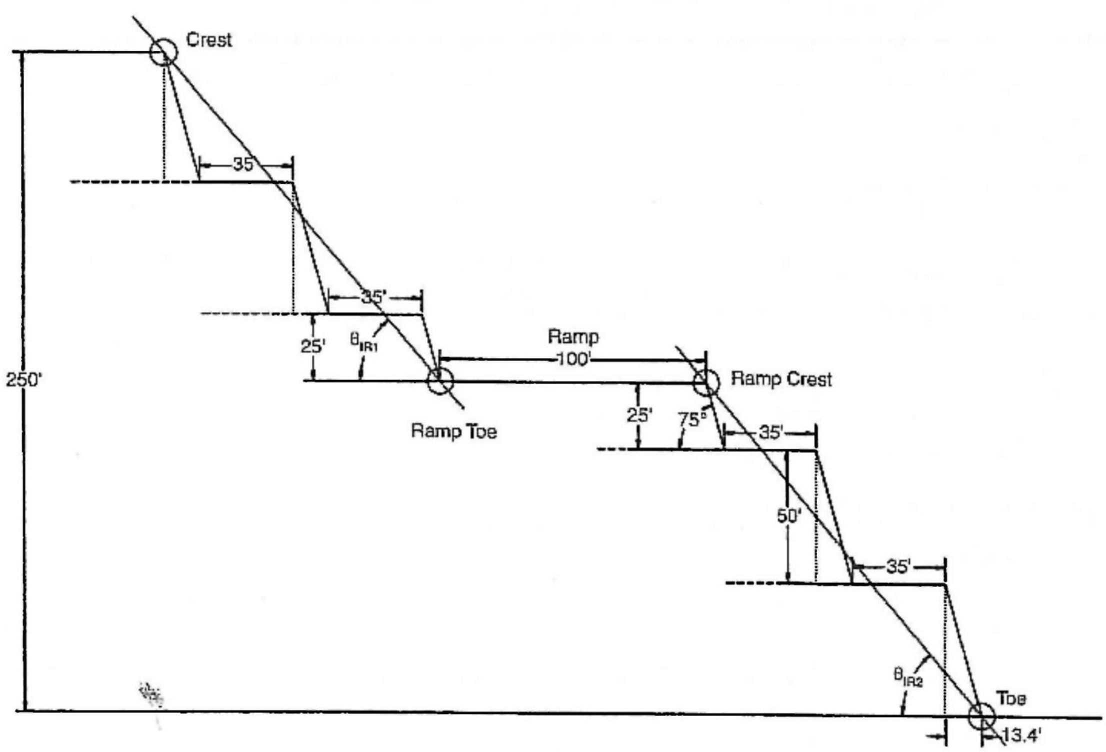


Figure 4.53. Interramp slope angles for Figure 4.52.

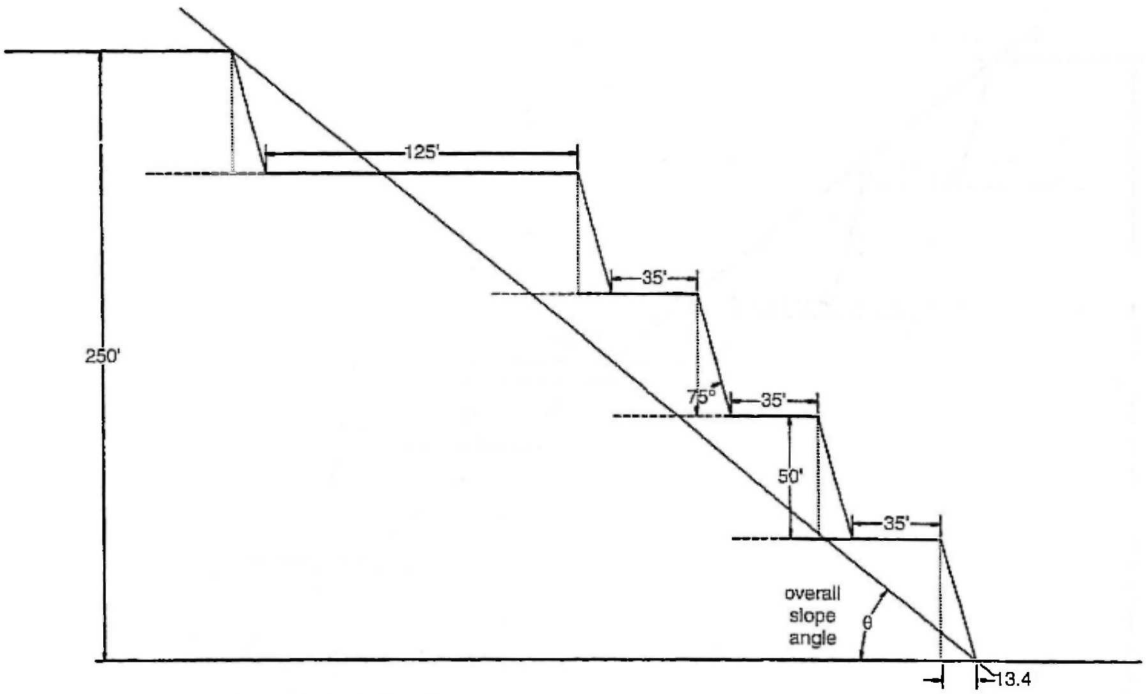


Figure 4.54. Overall slope angle with working bench included.

The interramp wall height is 125 ft for each segment. Generally the interramp wall heights and angles for the different slope segments would not be the same. From a slope stability viewpoint each interramp segment would be examined separately.

While active mining is underway, some working benches would be included in the overall slope. Figure 4.54 shows a working bench 125 ft in width included as bench 2. The overall slope angle is now

$$\Theta = \tan^{-1} \frac{5 \times 50}{125 + 4 \times 35 + \frac{5\times 50}{\tan 75°}} = 37.0°$$

The working bench is treated in the same way as a ramp in terms of interrupting the slope. The two interramp angles are shown in Figure 4.55. In this case

$$\Theta_{IR_1} = 75°$$

$$\Theta_{IR_2} = \tan^{-1} \frac{200}{3 \times 35 + \frac{4\times 50}{\tan 75°}} = 51.6°$$

The interramp wall heights are

$$H_1 = 50'$$

$$H_2 = 200'$$

For this section, it is possible that the ramp cuts bench 3 as before. This situation is shown in Figure 4.56.

The overall slope angle has now decreased to

$$\Theta = \tan^{-1} \frac{250}{125 + 3 \times 35 + 100 + \frac{5\times 50}{\tan 75°}} = 32.2°$$

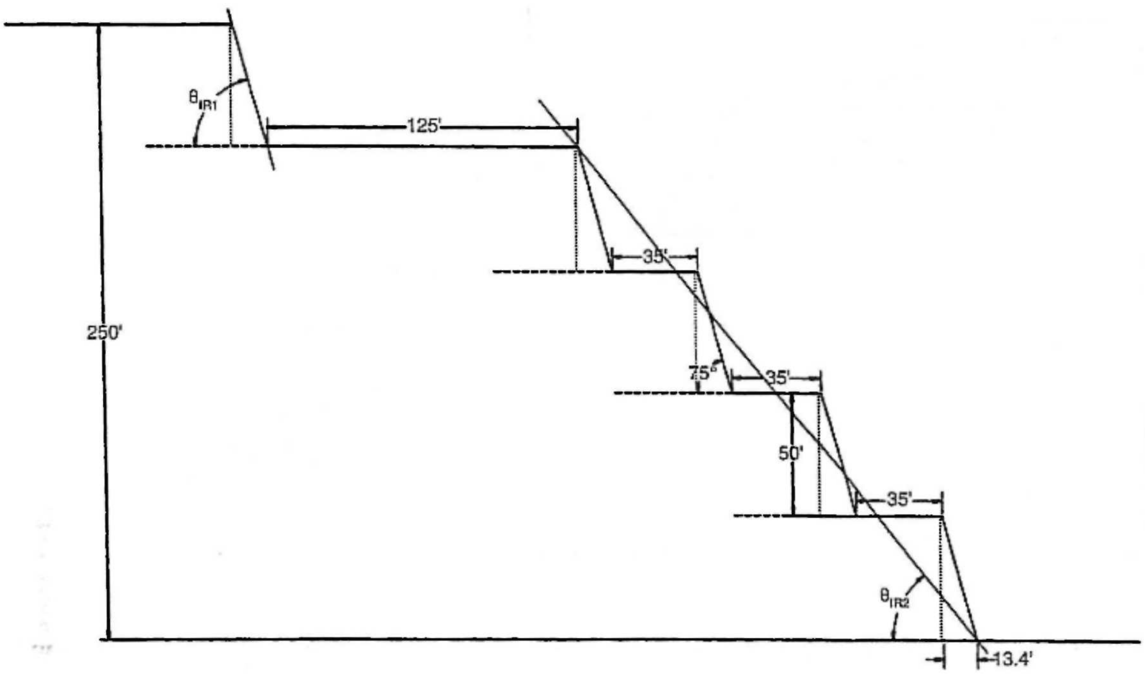


Figure 4.55. Interramp angles associated with the working bench.

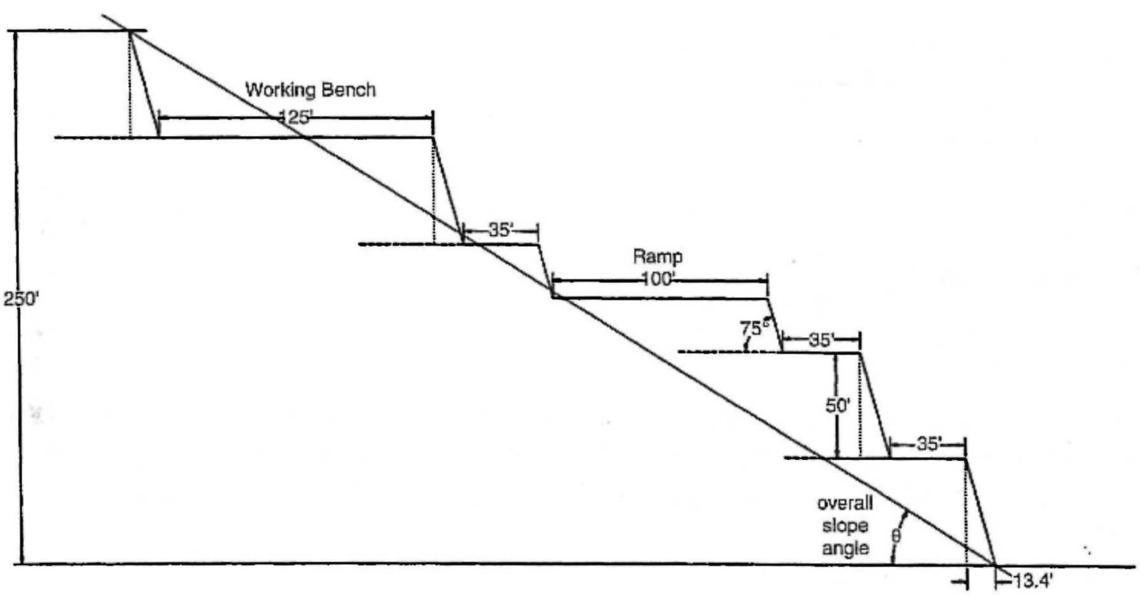


Figure 4.56. Overall slope angle with one working bench and a ramp section.

As shown in Figure 4.57, there are now three interramp portions of the slope. The interramp wall heights and angles are:

Segment 1:

$$\Theta_{IR_1} = 75^\circ$$
$$H = 50'$$

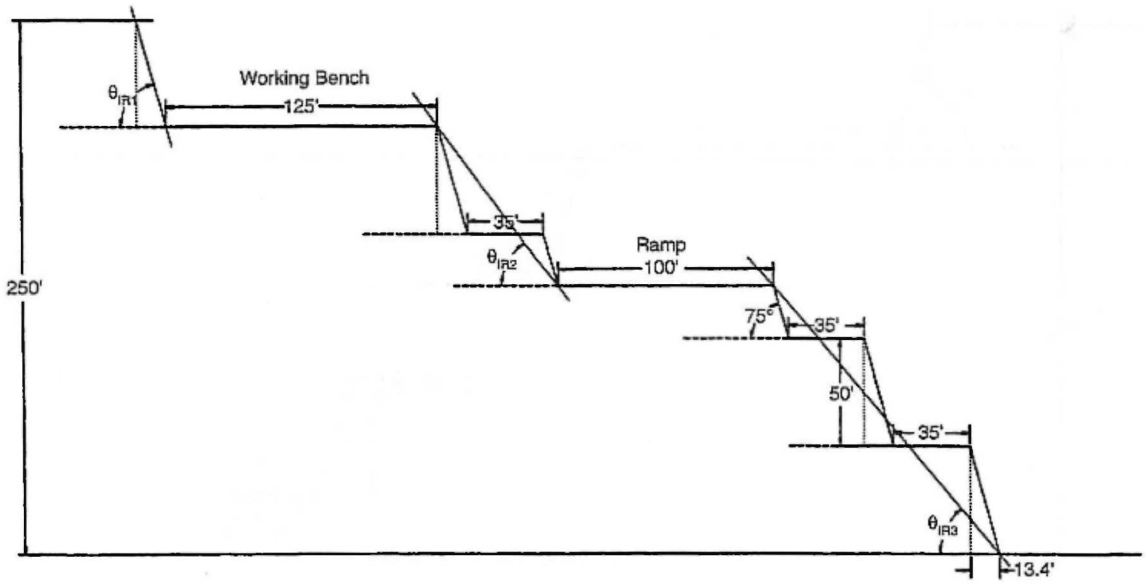


Figure 4.57. Interramp slope angles for a slope containing a working bench and a ramp.

Segment 2:

$$\Theta_{IR_2} = \tan^{-1}\frac{75}{35 + \frac{75}{\tan 75^\circ}} = 35.7^\circ$$

$$H = 75'$$

Segment 3:

$$\Theta_{IR_3} = \tan^{-1}\frac{125}{2 \times 35 + \frac{125}{\tan 75^\circ}} = 50.4^\circ$$

$$H = 125'$$

In Figure 4.57, the overall slope is shown to contain one working bench. Under some circumstances there may be several working benches involved in the mining of the slope. Figure 4.58 shows the case of a slope with 6 benches of which two are working benches 125 ft in width.

The overall (working slope) is given by

$$\Theta = \tan^{-1}\frac{300}{3 \times 35 + 2 \times 125 + \frac{300}{\tan 75^\circ}} = 34.6^\circ$$

The slope associated with each shovel working group is shown in Figure 4.59. In this case it is

$$\Theta = \tan^{-1}\frac{150}{125 + 35 + \frac{150}{\tan 75^\circ}} = 36.8^\circ$$

If the number of working benches is increased to 3 for the slope containing 6 benches, the overall slope would be further reduced. Thus to maintain reasonable slope angles, most mines have one working bench for a group of 4 to 5 benches.

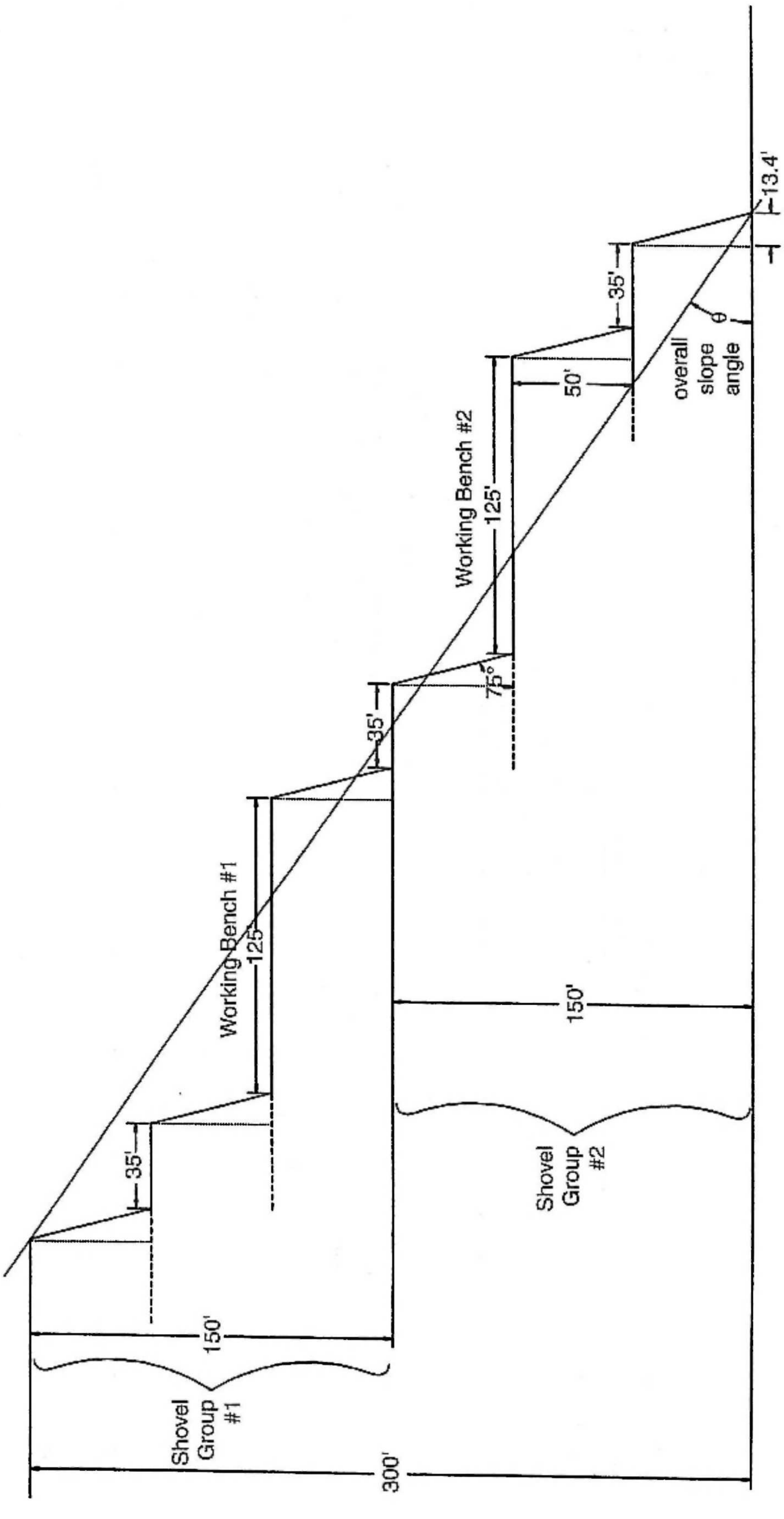


Figure 4.58. Overall slope angle for a slope containing two working benches.

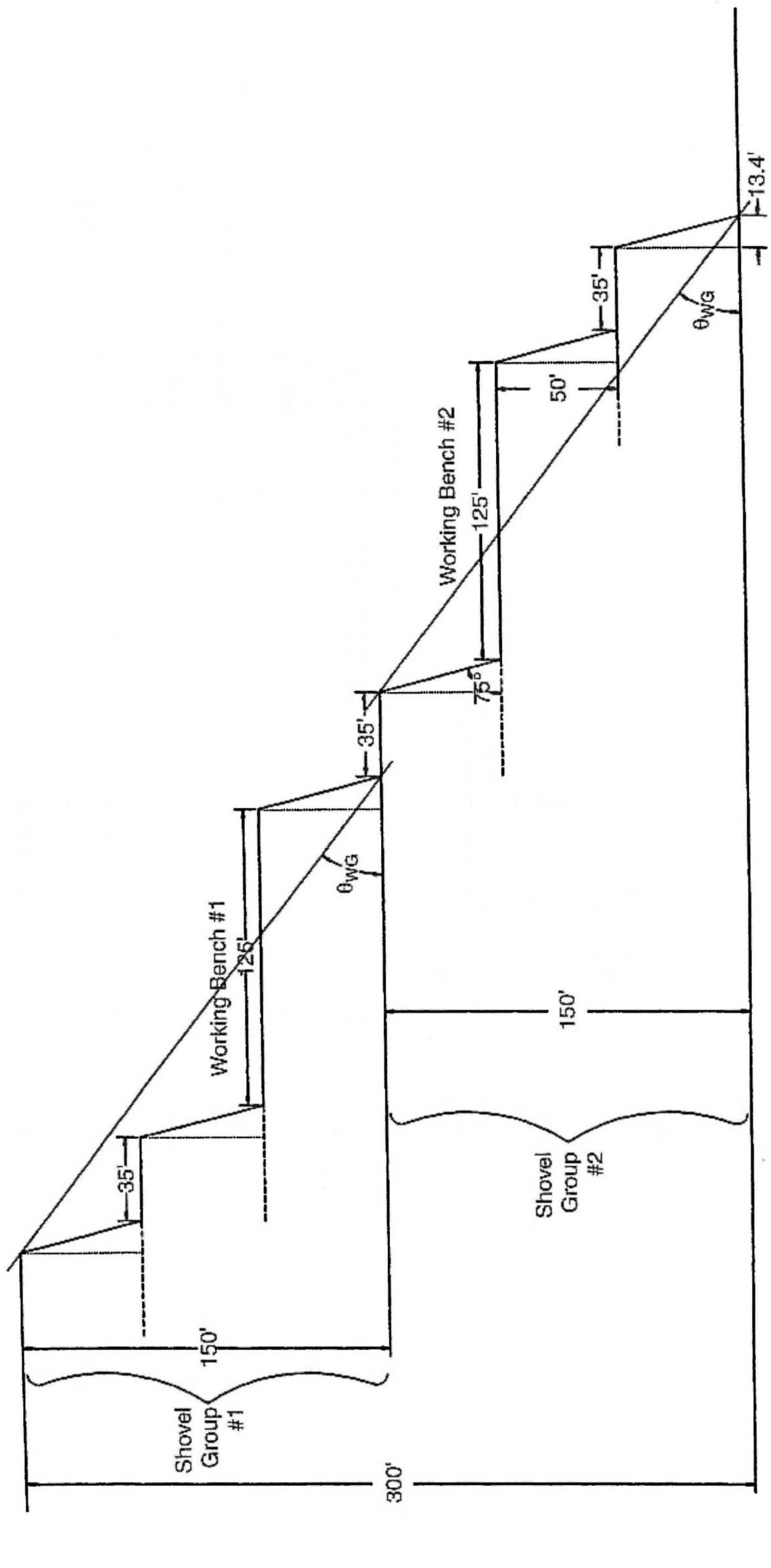


Figure 4.59. Slopes for each working group.

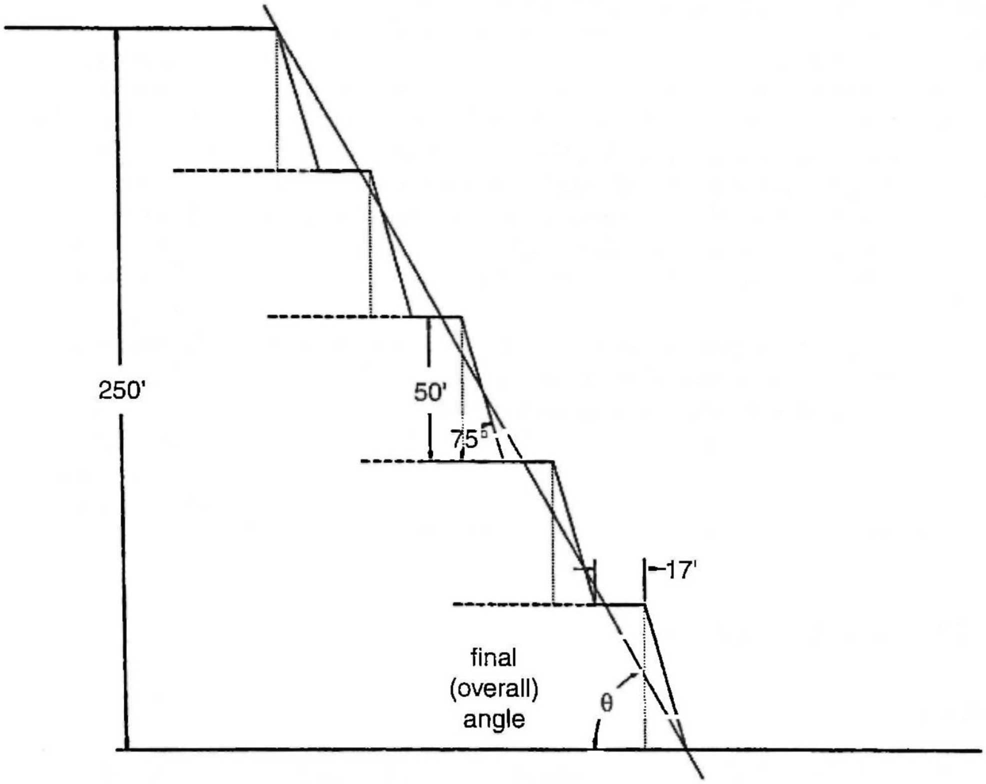


Figure 4.60. Final overall pit slope.

At the end of mining it is desired to leave the final slope as steep as possible. Some of the safety benches will be reduced in width while others may be eliminated entirely. For final walls, a bench width of approximately 1/3 of the bench height is commonly used. For this example with a bench height of 50 ft, the bench width becomes 17 ft. The final pit slope angle, assuming no ramp is needed on this wall (Fig. 4.60), becomes

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4 \times 17 + \frac{250}{\tan 75°}} = 61.6°$$

If the final bench faces could have been cut at 90° instead of 75°, then the final overall pit slope angle would be

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4 \times 17} = 74.8°$$

It is much more likely that the final face angles are 60° and the safety benches 20 ft wide. This gives

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4 \times 20 + \frac{250}{\tan 60°}} = 48°$$

Although much regarding final slope angles has to do with rock structure, care in blasting can make a major impact.

Table 4.2. Classification of open pit slope problems (Hoek, 1970b).

	Category	Conditions	Method of solution
A.	Unimportant slopes	Mining a shallow high grade orebody in favorable geological and climatic conditions. Slope angles unimportant economically and flat slopes can be used.	No consideration of slope stability required.
B.	Average slopes	Mining a variable grade orebody in reasonable geological and climatic conditions. Slope angles important but not critical in determining economics of mining.	Approximate analysis of slope stability normally adequate.
C.	Critical slopes	Mining a low grade orebody in unfavorable geological and climatic conditions. Slope angles critical in terms of both economics of mining and safety of operation.	Detailed geological and groundwater studies followed by comprehensive stability analysis usually required.

4.6 FINAL PIT SLOPE ANGLES

4.6.1 *Introduction*

During the early feasibility studies for a proposed open pit mine, an estimate of the safe slope angles is required for the calculation of ore to waste ratios and for the preliminary pit layout. At this stage generally the only structural information available upon which to base such an estimate is that obtained from diamond drill cores collected for mineral evaluation purposes. Sometimes data from surface outcrops are also available. How well these final slope angles must be known and the techniques used to estimate them depends upon the conditions (Table 4.2) applicable. During the evaluation stage for categories B and C, the best engineering estimate of the steepest safe slope at the pit limits in each pit segment is used. Since the information is so limited, they are hedged with a contingency factor. If the property is large and has a reasonably long lifetime, initially the exact slope angles are of relatively minor importance. The effect of steeper slopes at the pit limits is to increase the amount of ore that can be mined and therefore increase the life of the mine. The effect of profits far in the future has practically no impact on the net present value of the property.

During the pre-production period and the first few years of production, the operating slopes should however be as steep as possible while still providing ample bench room for optimum operating efficiency. The minimization of stripping at this stage has a significant effect on the overall economics of the operation. The working slopes can then be flattened until they reach the outer surface intercepts. Steepening operations then commence to achieve the final pit slopes (Halls, 1970). Cases do occur where the viability of an orebody is highly dependent on the safe slope angle that can be maintained. Special measures, including the collection of drillhole data simply for making slope determinations are then taken.

There are a number of excellent references which deal in great detail with the design of pit slopes. In particular *Rock Slope Engineering* by Hoek & Bray (1977), and the series of publications developed within the *Pit Slope Manual* series produced by CANMET should be mentioned. This brief section focusses on a few of the underlying concepts, and presents some curves extracted largely from the work of Hoek (1970a, 1970b) which may be used for making very preliminary estimates.

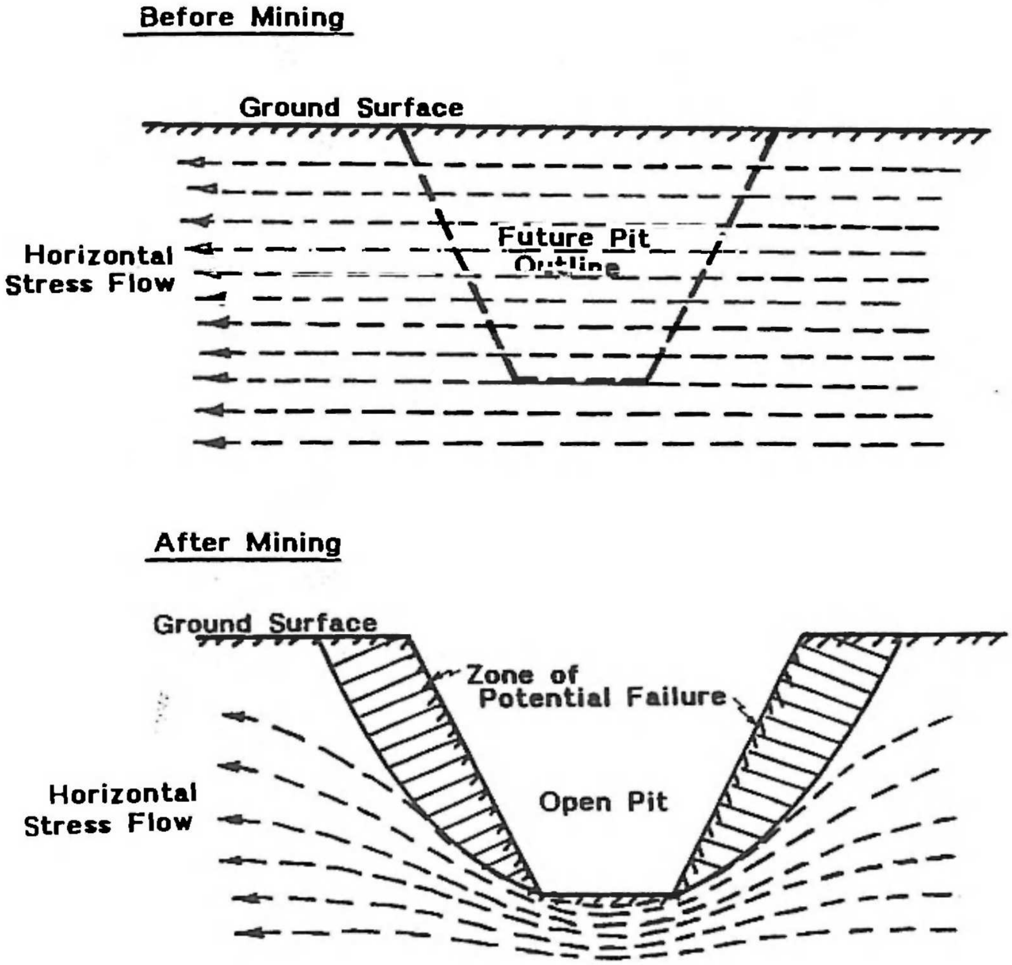
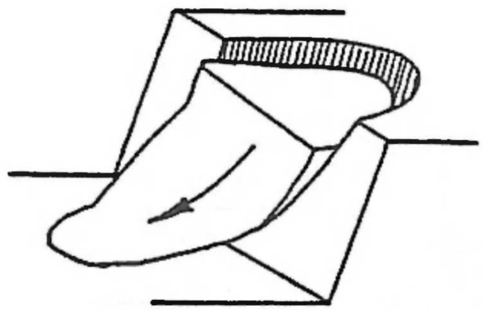


Figure 4.61. Horizontal stress redistribution due to the creation of a pit.

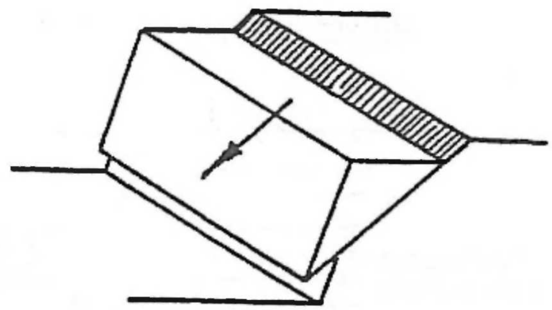
4.6.2 *Geomechanical background*

Figure 4.61 shows diagrammatically the horizontal flow of stress through a particular vertical section both with and without the presence of the final pit. With the excavation of the pit, the pre-existing horizontal stresses are forced to flow beneath the pit bottom (and around the pit ends).

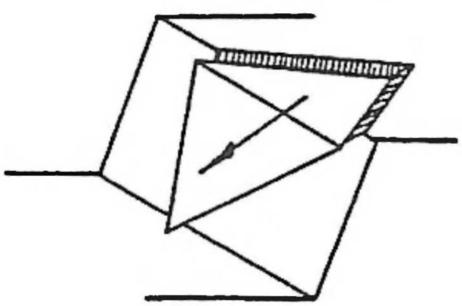
The vertical stresses are also reduced through the removal of the rock overlying the final slopes. This means that the rock lying between the pit outline and these flow lines is largely distressed. As a result of stress removal, cracks/joints can open with a subsequent reduction in the cohesive and friction forces restraining the rock in place. Furthermore, ground water can more easily flow through these zones, reducing the effective normal force on potential failure planes. As the pit is deepened, the extent of this destressed zone increases, and the consequences of a failure becomes more severe. The chances of encountering adverse structures (faults, dykes, weakness zones, etc.) within these zones increase as well. Finally, with increasing pit depth, the relative sizes of the individual structural blocks making up the slopes become small compared to the overall volume involved. Thus the failure mechanism may change from one of structural control to one controlled by the characteristics of a granular mass. Figure 4.62 shows the four major types of failure which occur in an open



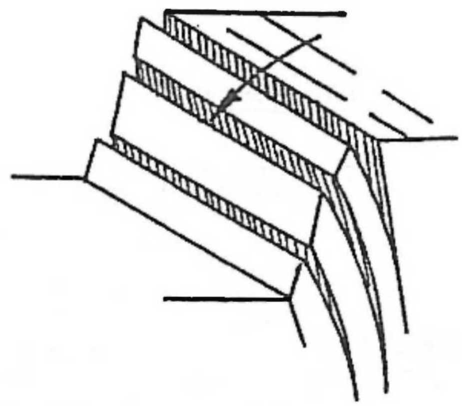
a. Circular failure in overburden soil, waste rock or highly fractured rock with no identifiable structural pattern.



b. Plane failure in rock with highly ordered structure such as slate.



C. Wedge failure on two intersecting discontinuities.



d. Toppling failure in hard rock which can form columnar structure separated by steeply dipping discontinuities.

Figure 4.62. The most common slope failure types (Hoek & Bray, 1977).

pit. In this section the discussion will concentrate on planar failure along major structures and circular failure.

4.6.3 *Planar failure*

Planar failure along various types of discontinuities can occur on the bench scale, interramp scale and pit wall scale (major fault, for example). Bench face instabilities due to the daylighting of major joint planes means that the overall slope must be flattened to provide the space required for adequate safety berms. The final slope is made up of flattened bench faces, coupled with the safety berm steps. The design slope angle may be calculated once an average stable bench face angle is determined. Since one is concerned with final pit wall stability, the analysis in this section applies to a major structure occurring in the pit wall, although the same type of analysis applies on the smaller scale as well. Figure 4.63 shows the dimensions and forces in a rock slope with a potential failure plane. The Mohr-Coulomb failure criterion has been used.

The following definitions apply:

i is the average slope angle from horizontal (degrees),

β is the angle of the discontinuity from the horizontal (degrees),

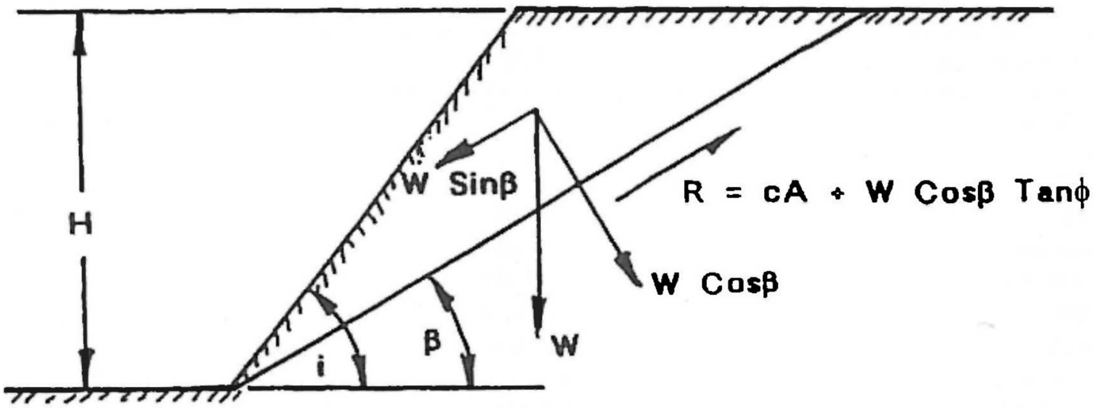


Figure 4.63. Dimensions and forces in a rock slope with a potential failure plane (Hoek, 1970a).

W is the block weight,
R is the resisting force,
c is the cohesion,
ϕ is the friction angle,
$W \cos\beta$ is the normal force,
$W \sin\beta$ is the driving force,
A is the area of the failure plane.

The factor of safety (F) is defined by

$$F = \frac{\text{Total force available to resist sliding}}{\text{Force tending to induce sliding}} \tag{4.1}$$

For the case shown in Figure 4.63 (drained slope) Equation (4.1) becomes

$$F = \frac{cA + W\cos\beta\tan\phi}{W\sin\beta} \tag{4.2}$$

If there is water present, then the factor of safety is expressed as

$$F = \frac{cA + (W\cos\beta - U)\tan\phi_a}{W\sin\beta + V} \tag{4.3}$$

where U is the uplift force along the base of the block due to water pressure, and V is the horizontal force along the face of the block due to water in the tension crack, ϕ_a is the friction angle (as affected by the water). Typical values for the cohesive strength and friction angles of soils and rock are given in Tables 4.3 and 4.4. As the height H of the slope increases the relative contribution of the cohesion to the total resistance decreases. For very high slopes, the stable slope angle approaches the friction angle ϕ. Hoek (1970a) has presented the relationship between slope height and slope angle functions for plane failure in a drained slope given in Figure 4.64.

Assume for example that the average planned slope angle i is 70°, the orientation of the potential failure plane β is 50° and the friction angle ϕ is 30°. Thus

$$X = 2\sqrt{(i-\beta)(\beta-\phi)} = 2\sqrt{20 \times 20} = 40°$$

From Figure 4.64 the slope height function Y is read as

$$Y = 14$$

Table 4.3. Cohesive strengths for 'intact' soil and rock (Robertson, 1971).

Material description	c (lb/ft^2)	c (kg/m^2)
Very soft soil	35	170
Soft soil	70	340
Firm soil	180	880
Stiff soil	450	2200
Very stiff soil	1600	7800
Very soft rock	3500	17,000
Soft rock	11,500	56,000
Hard rock	35,000	170,000
Very hard rock	115,000	560,000
Very very hard rock	230,000	1,000,000

Table 4.4. Friction angles (degrees) for typical rock materials (Hoek, 1970a).

Rock	Intact rock ϕ	Joint ϕ	Residual ϕ
Andesite	45	31–35	28–30
Basalt	48–50	47	
Chalk	35–41		
Diorite	53–55		
Granite	50–64		31–33
Graywacke	45–50		
Limestone	30–60		33–37
Monzonite	48–65		28–32
Porphyry		40	30–34
Quartzite	64	44	26–34
Sandstone	45–50	27–38	25–34
Schist	26–70		
Shale	45–64	37	27–32
Siltstone	50	43	
Slate	45–60		24–34

Other materials	Approximate ϕ
Clay gouge (remoulded)	10–20
Calcitic shear zone material	20–27
Shale fault material	14–22
Hard rock breccia	22–30
Compacted hard rock aggregate	40
Hard rock fill	38

Knowing that

$$c = 1600\,\text{lb/ft}^2$$

$$\gamma = 160\,\text{lb/ft}^3$$

the limiting ($F = 1$) slope height H with such a structure passing through the toe is found using

$$Y = 14 = \frac{\gamma H}{c} = \frac{160}{1600}H$$

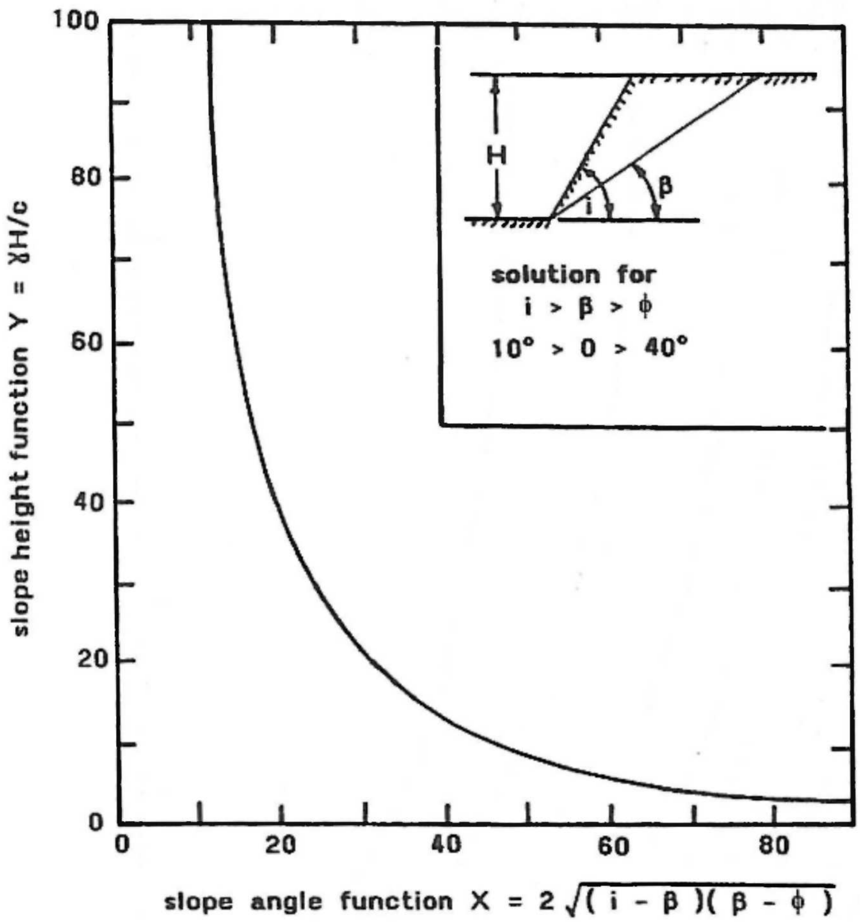


Figure 4.64. Relationship between slope height and slope angle functions for plane failure in a drained slope (Hoek, 1970a).

Thus

$$H = 140\,\text{ft}$$

If the planned pit depth is 500 ft, one could determine the limiting ($F = 1$) pit slope angle. The slope height function is

$$Y = \frac{\gamma H}{c} = \frac{160 \times 500}{1600} = 50$$

From Figure 4.64 one finds that

$$X = 17.5$$

Solving for i yields

$$i = 57.7°$$

The general family of curves corresponding to various safety factors is given in Figure 4.65.

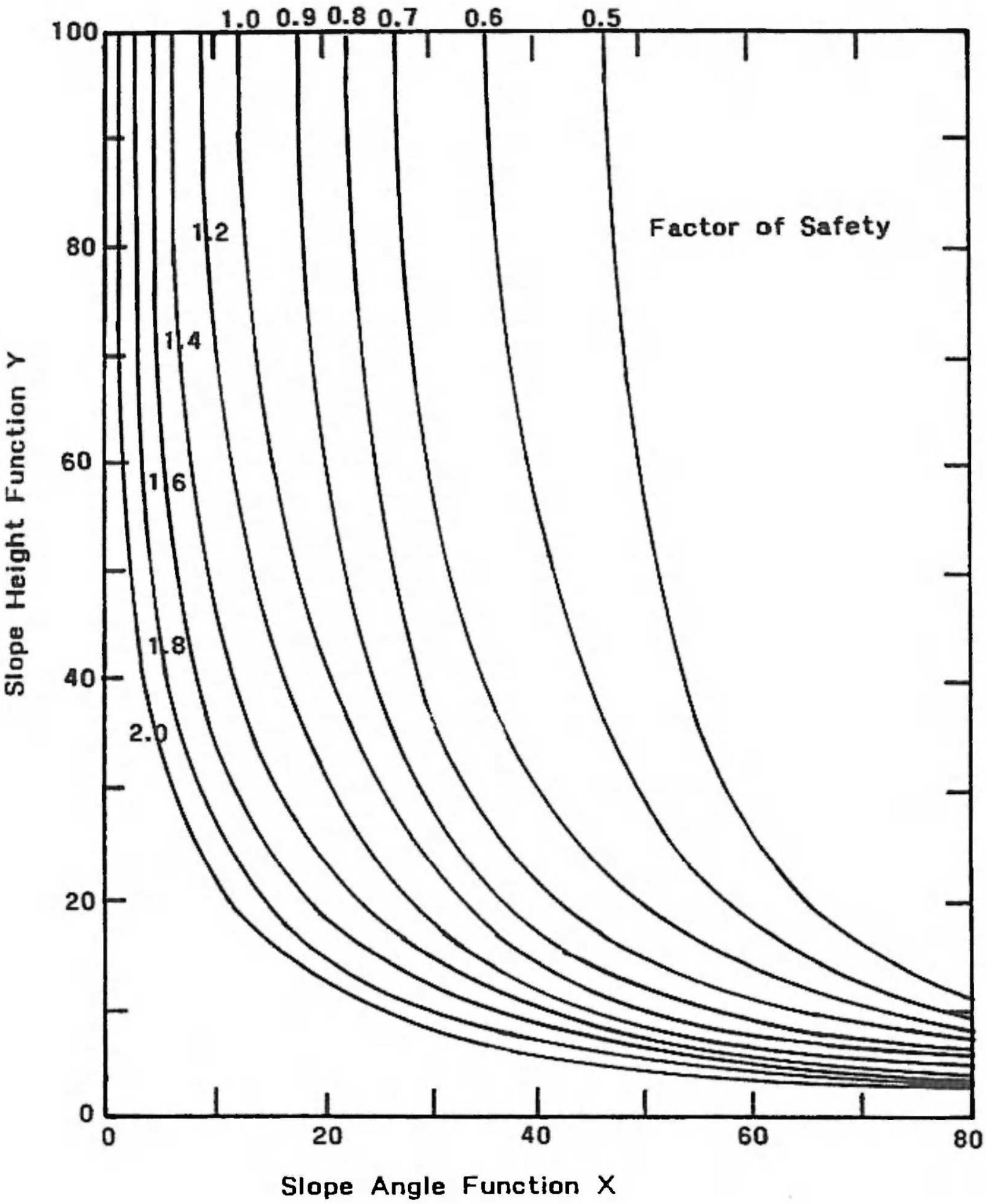


Figure 4.65. Slope design chart for plane failure including various safety factors (Hoek, 1970a).

The question naturally arises as to what an appropriate safety factor might be? This depends on the confidence one has in the 'goodness' of the input data and also on the function of the structure. Jennings & Black (1963) have provided the following advice:

For permanent structures, such as earth dams, F should not be less than 1.5 for the most critical potential failure surface, but for temporary constructions, where engineers are in continual attendance, a lower factor may be accepted. In civil engineering work, construction factors of safety are seldom allowed to be less than 1.30. An open pit is a 'construction' of a very particular type and it is possible that a factor of 1.20–1.30 may be acceptable in this case.

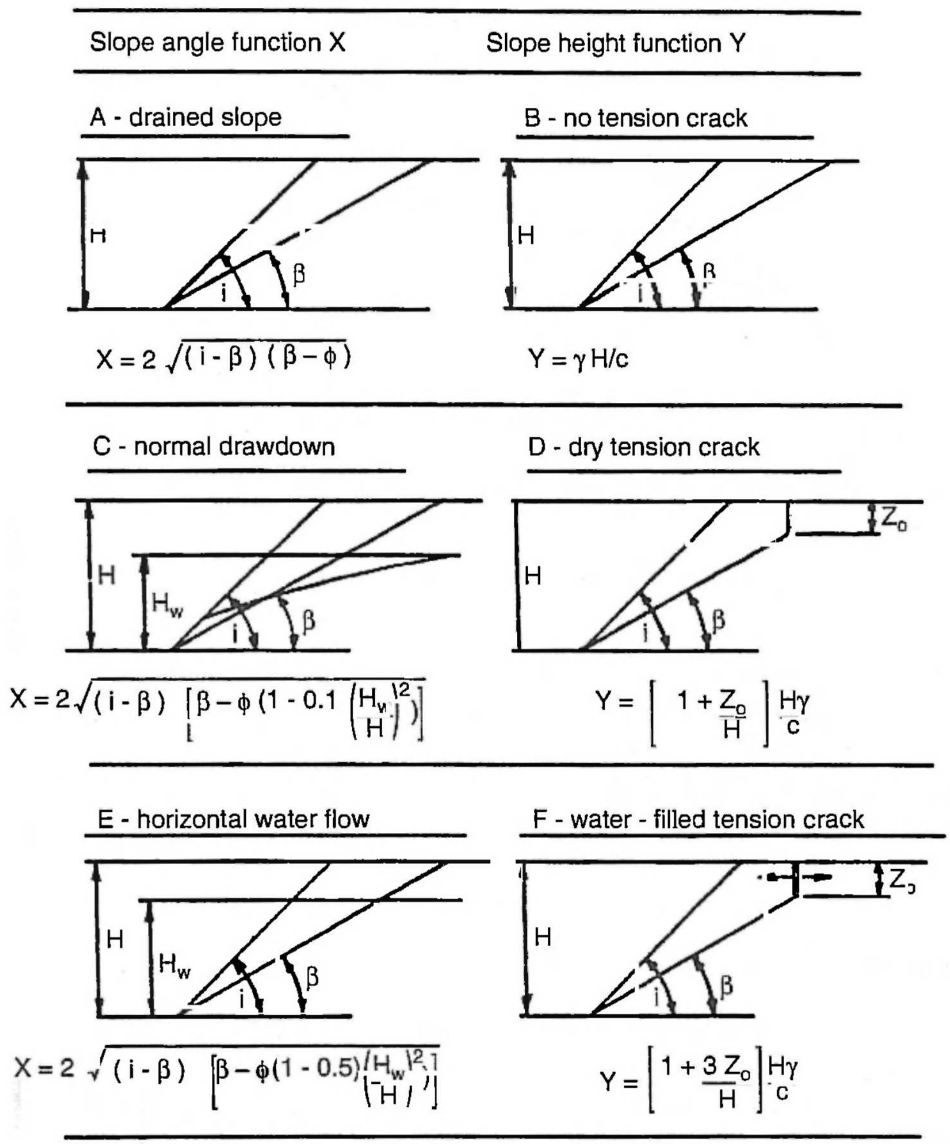


Figure 4.66. Slope angle and slope height functions for different water and tension crack conditions (Hoek, 1970a).

The confidence placed in any value calculated as the factor of safety of a slope depends upon the accuracy with which the various factors involved can be estimated. The critical items are the selection of the most adverse surface for potential failure, the measurement of the shear strength of the materials on this surface and the estimation of the water pressures in the soil pores and in any fissures along the surface.

If one were to select a safety factor of 1.2 for the previous example, one finds that for $Y = 50$, $X = 13.5$. The slope angle becomes

$$i = 54.6^\circ$$

The example applies for the very special case of a drained slope without a tension crack. Often a tension crack will be present and there can be a variety of different slope water conditions. Hoek (1970a) has developed a simple way of handling these. Figure 4.66 provides three different expressions for X corresponding to different slope water conditions

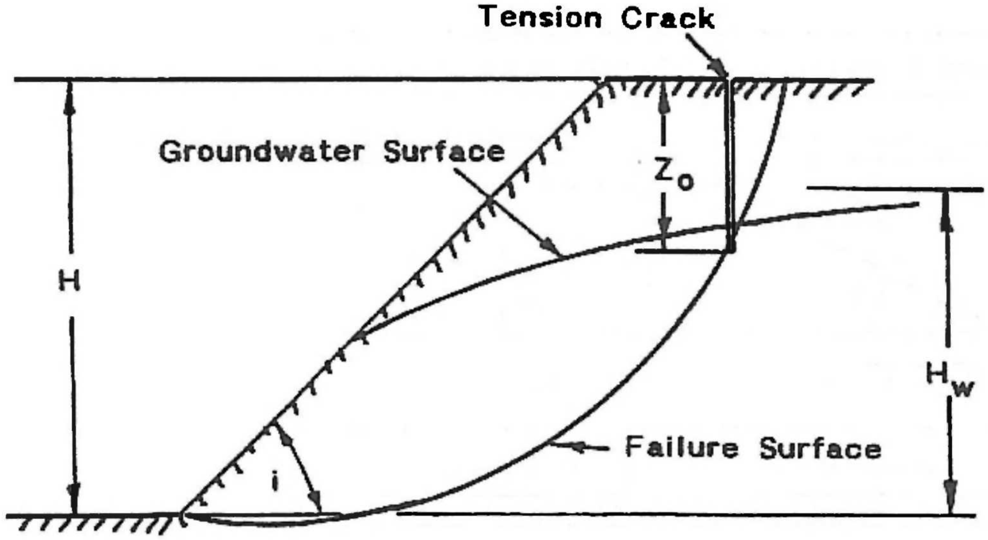


Figure 4.67. Diagrammatic representation of circular failure in a slope (Hoek, 1970a).

and three different expression for Y relating to the tension crack. Thus nine different X–Y combinations are possible. The one used in the earlier examples was combination A–B. From Figure 4.66 one finds the X–Y combination most appropriate to the problem at hand.

The known values are substituted and Figure 4.65 is used to determine the desired missing value. The interested reader is encouraged to evaluate the effect of different slope water conditions on the slope angle.

4.6.4 *Circular failure*

Hoek (1970a) has applied the same approach to the analysis of circular failures (Fig. 4.67).

Such deep seated failures occur when a slope is excavated in soil or soft rock in which the mechanical properties are not dominated by clearly defined structural features. This type of failure is important when considering the stability of:

– very high slopes in rock in which the structural features are assumed to be randomly oriented,
– benches or haul road cuts in soil,
– slimes dams,
– waste dumps.

Figure 4.68 gives the relationship between the slope height function and slope angle function for circular failure in drained slopes without a tension crack (F – 1). The corresponding chart, including different safety factor values, is given in Figure 4.69. To accommodate different tension crack and slope water conditions, Figure 4.70 has been developed. This set of curves is used in exactly the same way as described earlier.

4.6.5 *Stability of curved wall sections*

The approaches discussed to this point have applied to pit wall sections which can be approximated by two-dimensional slices. Open pits often take the form of inverted cones or have portions containing both convex and concave wall portions (Figure 4.71).

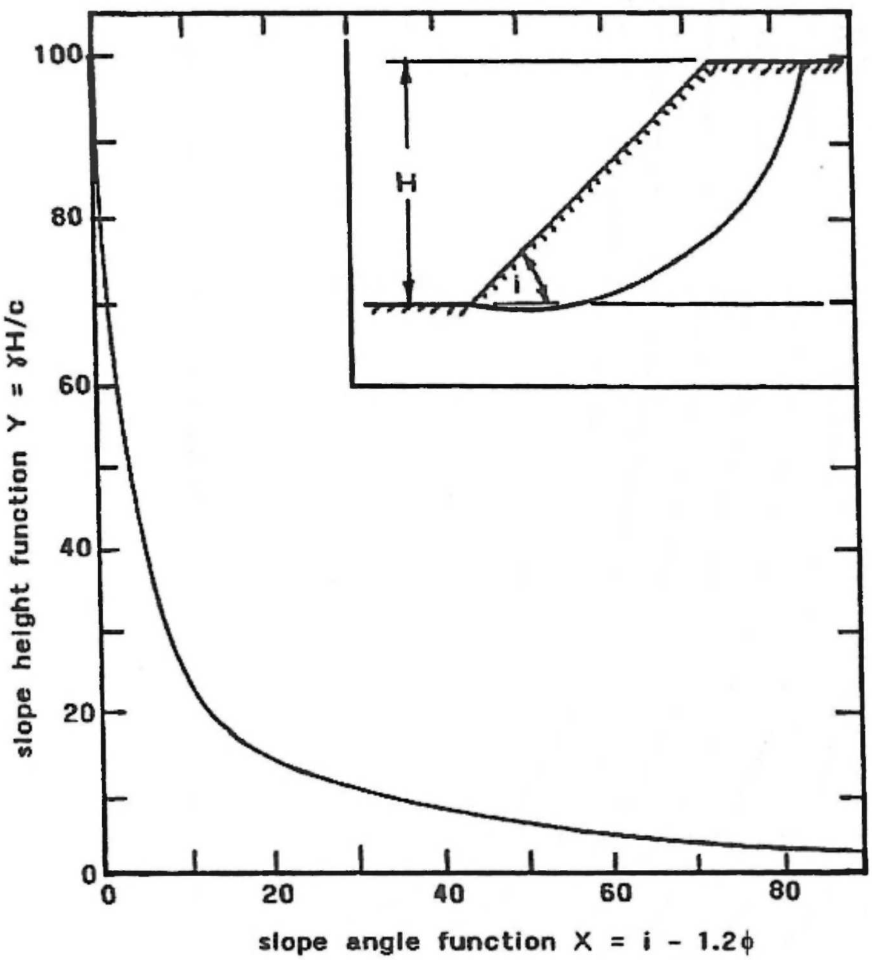


Figure 4.68. Plot of slope height versus slope angle functions for circular failure analysis (Hoek, 1970a).

Very little quantitative information on the effect of pit wall curvature on stability is available from the literature. Convex portions of a pit wall (noses which stick out into the pit) frequently suffer from unstable slopes. The relaxation of lateral stresses give rise to a reduction in the normal stress across potential failure planes and vertical joint systems can open. For concave portions of the pit, the arch shape of the slope tends to induce compressive lateral stresses which increase the normal stress across potential failure planes. The slopes are more stable due to the increased frictional resistance.

Hoek (1970a) suggests that curvature of the slope in plan can result in critical slope differences of approximately 5° from that suggested by the planar analyses. A concave slope, where the horizontal radius of curvature is of the same order of magnitude as the slope height, may have a stable slope angle 5° steeper than for a straight wall (infinite radius of curvature). On the other hand, a convex slope may require flattening by about 5° in order to improve its stability.

However, improved drainage in the convex slopes over that available with the pinched concave shape may provide a stability advantage. Thus, there may be some cancelling of advantages/disadvantages. Hence, each pit curvature situation must be carefully examined.

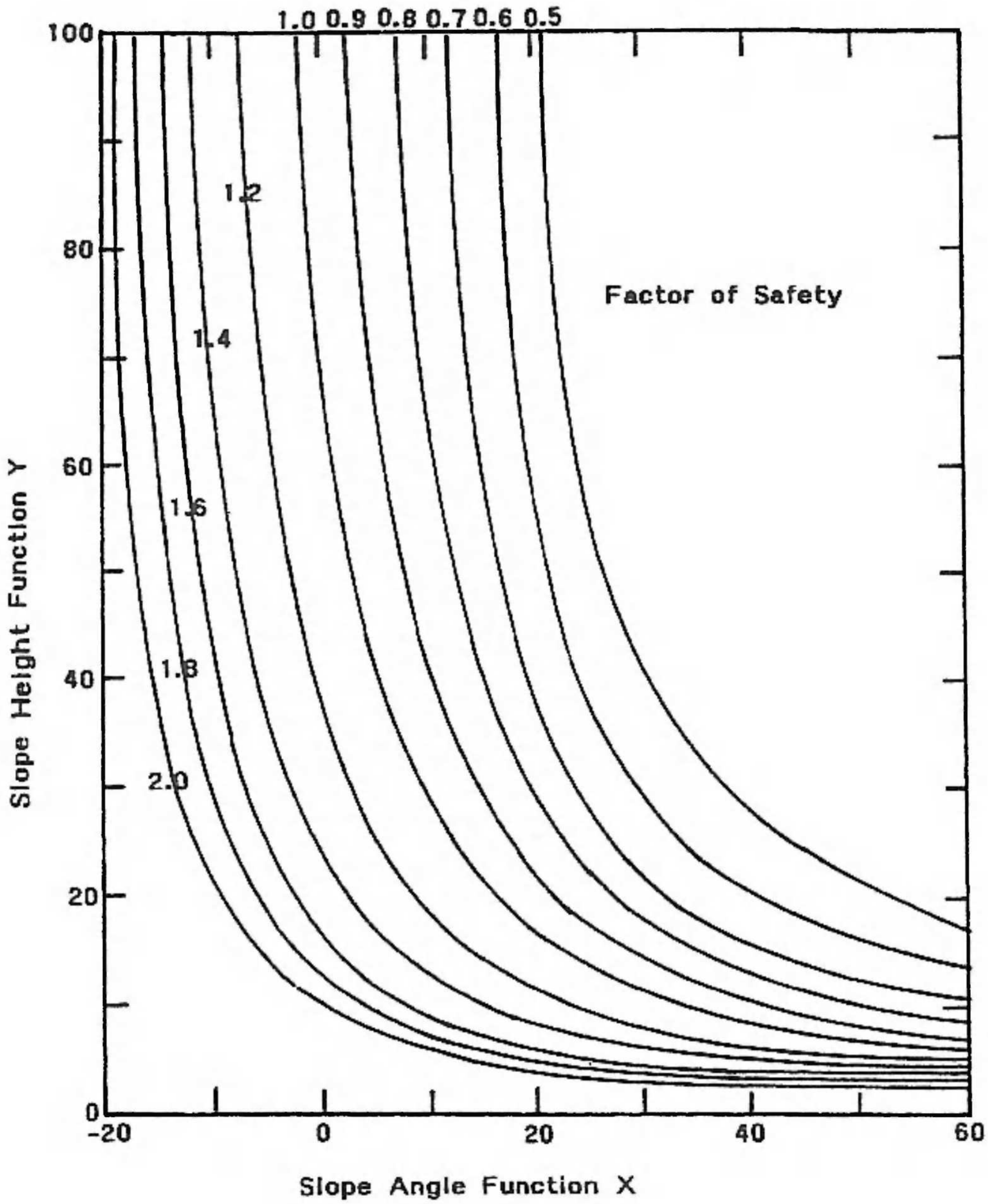


Figure 4.69. Slope design chart for circular failure including various safety factors (Hoek, 1970a).

4.6.6 *Slope stability data presentation*

Figure 4.72 developed by Hoek & Bray (1977) is a good example of how structural geology information and preliminary evaluation of slope stability of a proposed open pit mine can be presented. A contour plan of the proposed open pit mine is developed and contoured stereoplots of available structural data are superimposed. In this particular case two distinct structural regions denoted by A and B have been identified and marked on the plan. Based simply on geometry (of the pit slopes and structures), the potential failure types are identified. Each of these would then be examined using appropriate material properties and ground water conditions. Required design changes, additional data collection, etc. will emerge.

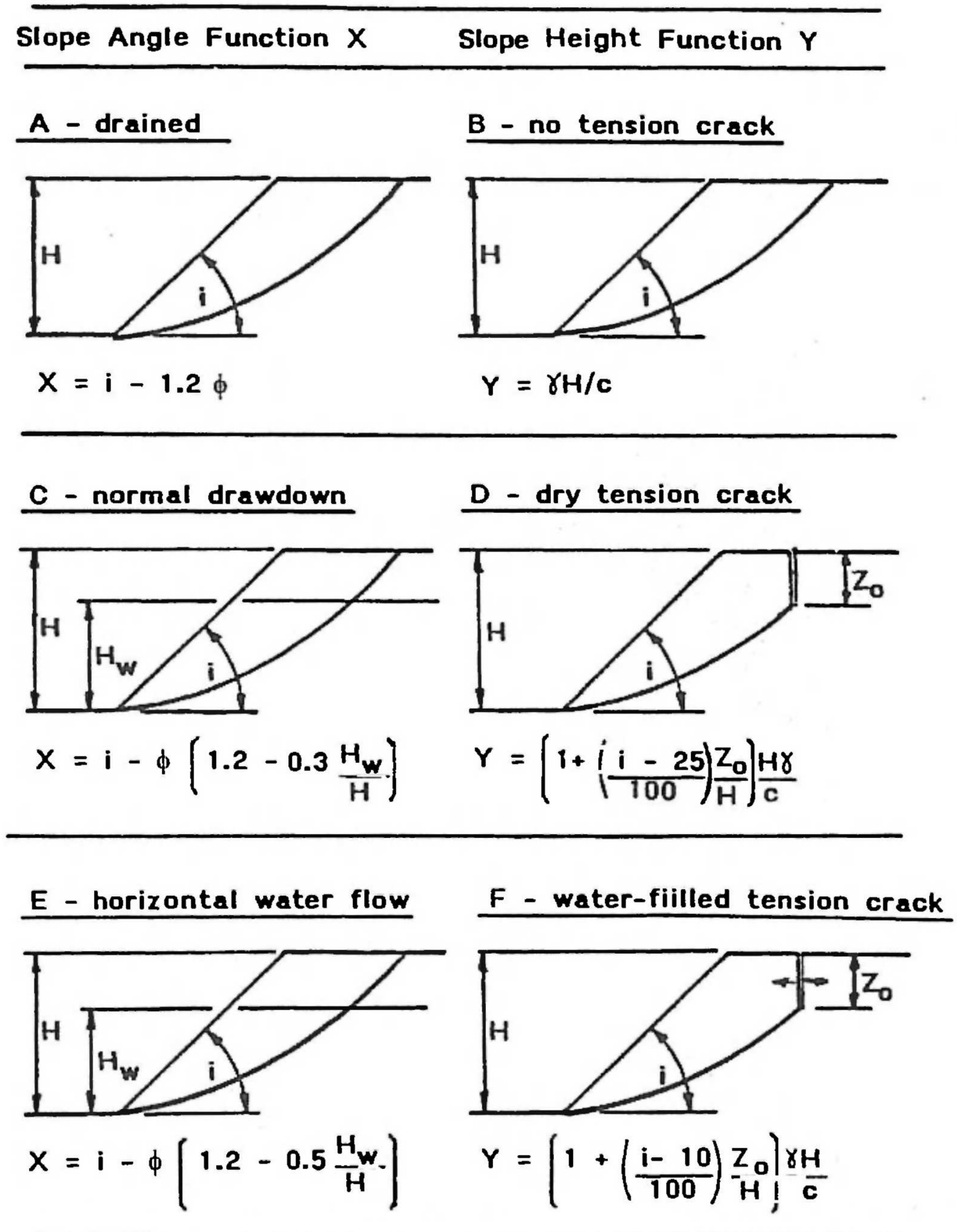


Figure 4.70. Slope angle and slope height functions for different water and tension crack conditions (Hoek, 1970a).

4.6.7 *Slope analysis example*

Reed (1983) has reported the results of applying the Hoek & Bray (1977) approach to the Afton copper-gold mine located in the southern interior of British Columbia. For the purpose of analyzing the stability of the walls of the open pit, it was divided into 9 structural domains (Fig. 4.73).

For each structural domain, a stability analysis was made of:

– the relative frequency of the various fault and bedding plane orientations, and
– the orientation of the pit wall in that particular domain.

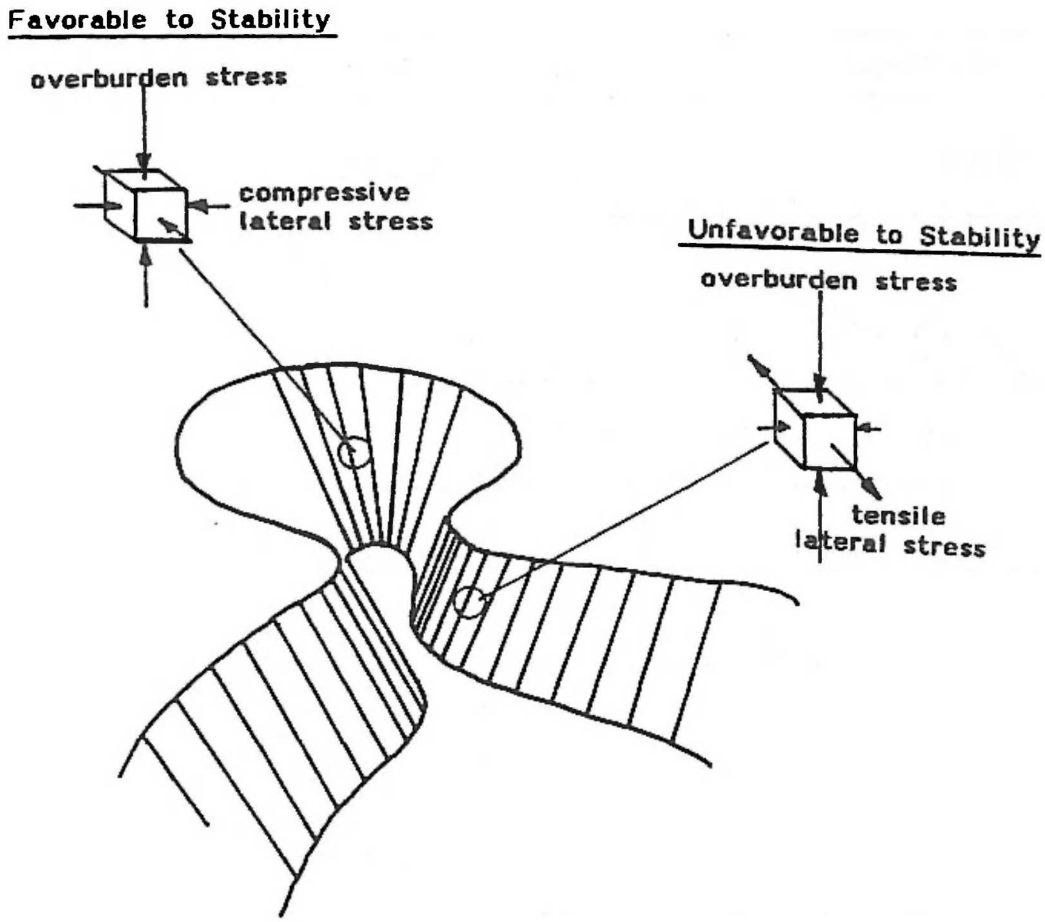


Figure 4.71. Influence of three-dimensional pit shape upon slope stability (Hoek, 1970a).

The safety factors were calculated for plane failure, wedge failure and circular failure in each domain. Table 4.5 shows the results of these stability analyses.

The 'maximum safe slope angle' for the pit wall in each domain corresponds to a calculated safety factor of 1.2. The results in Table 4.5 predict wall failure in all domains if the slopes are wet. The mine, however, lies in a semi-arid area and expected ground water quantities were small. In addition, horizontal drainholes would be used to reduce ground water pressures in domains 3 and 6. Problems would still be expected in domains 3 and 6. Since domain 3 is a relatively narrow domain and the probability of a major slide occurring was small, the design slope of the wall in that area was not flattened. At the time the paper was written (1983), the pit had reached a depth of 480 ft. Two failures had been experienced in domain 3 and several berm failures in domain 6. There was no indication of impending major failures. Final pit depth was planned to be 800 ft.

4.6.8 *Economic aspects of final slope angles*

Figure 4.74 illustrates the volume contained in a conical pit as a function of final slope angle and depth.

For a depth of 500 ft and an overall final pit angle of 45°, 1.4×10^7 tons of rock must be moved. Within the range of possible slopes (20° to 70°) at this depth the volume to be

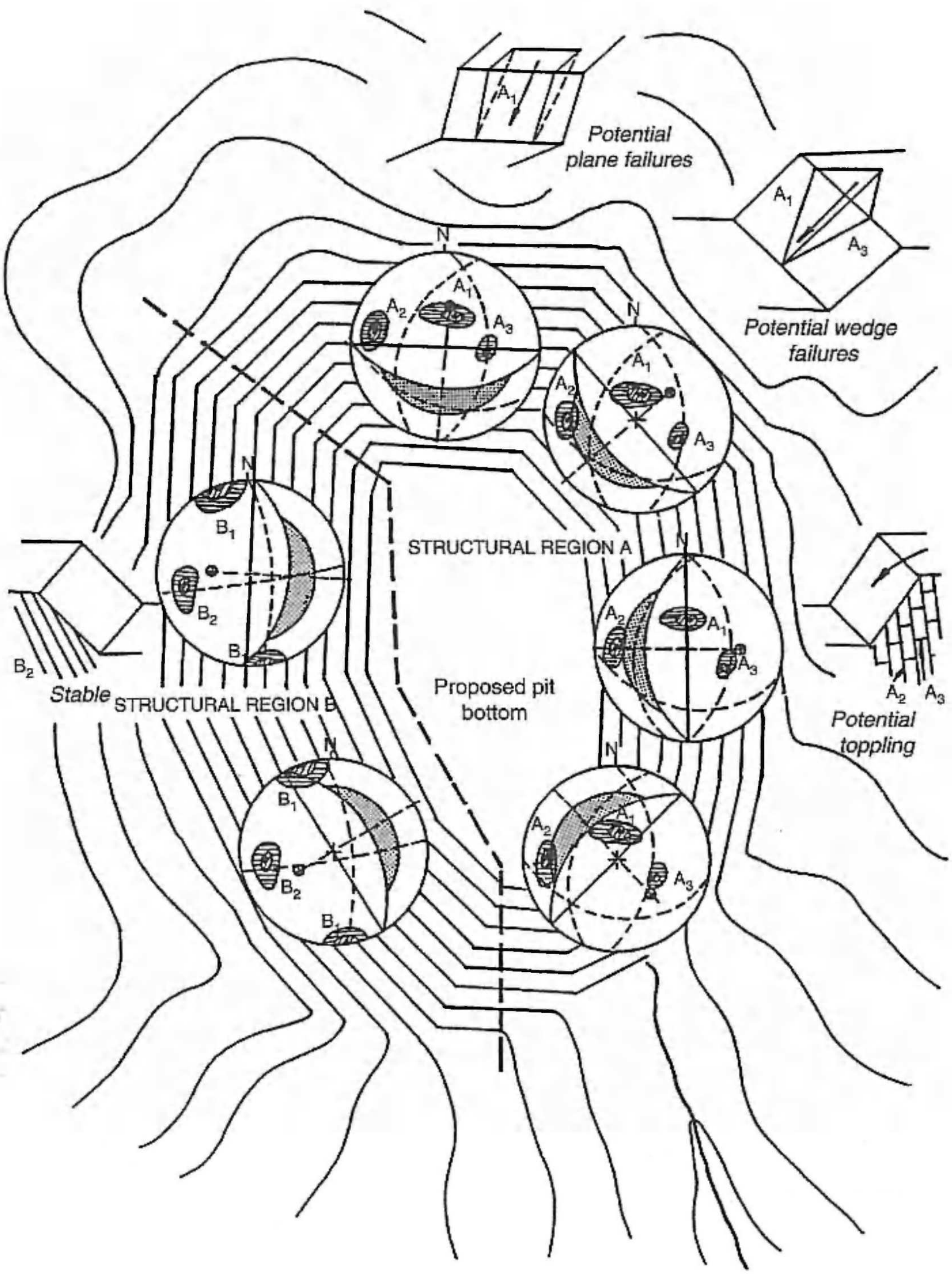


Figure 4.72. Presentation of structural geology information and preliminary evaluation of slope stability of a proposed open pit mine (Hoek & Bray, 1977).

moved approximately doubles for every 10° flattening of the slope. Flattening the slope of the 500 ft deep conical pit from 50° to 40° increases the mass of rock from 1.0×10^7 to 2.0×10^7 tons. This simple example shows that the selection of a particular slope can have a significant impact on the scale of operations and depending upon the shape, size and grade of the ore contained within the pit, on the overall economics.

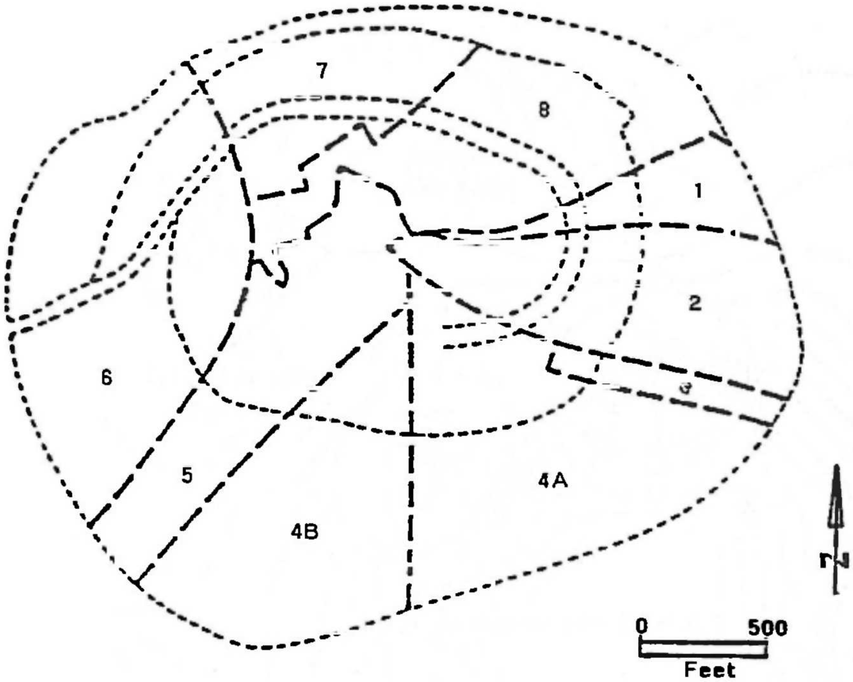


Figure 4.73. Division of the Afton open pit into 9 structural domains (Reed, 1983).

Table 4.5. Calculated and design slope angles for the Afton mine (Reed, 1983).

Domain	Maximum safe slope angle		Design slope angle
	Wet	Dry	
1	24°	54°	45°
2	52°	52°	45°
3	24°	41°	45°
4A	27°	49°	45°
4B	45°	42°	45°
5	22°	42°	45°
6	28°	39°	40°
7	33°	42°	40°
8	32°	43°	40°

4.7 PLAN REPRESENTATION OF BENCH GEOMETRY

Figure 4.75 is a cross-sectional representation of an open pit mine. Figure 4.76 is a 'birds-eye' (plan) view of the same pit. No attempt has been made to distinguish between the toes and crests (which are marked in Fig. 4.77) and hence the figure is difficult to interpret.

Several different techniques are used by the various mines to assist in plan representation and visualization. In Figure 4.78 the bank slopes have been shaded and the benches labelled with their elevations.

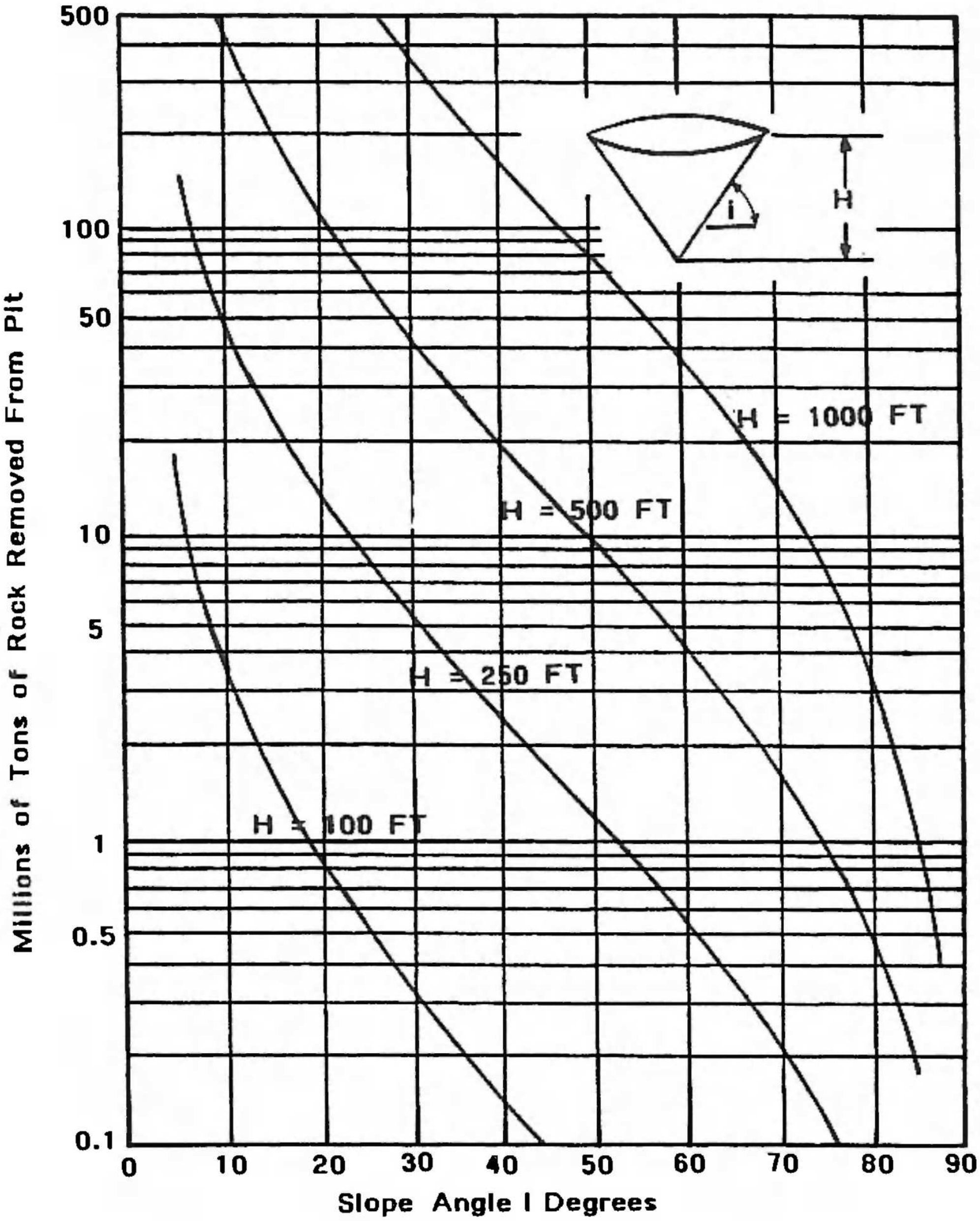


Figure 4.74. Influence of pit depth and slope angle on the amount of rock removed in mining a conical open pit (Hoek & Pentz, 1970).

Figure 4.79 is an example of this type of representation for an actual mine. An alternative is to draw the crests with solid lines and the toes with dashed lines. The result is shown in diagrammatic form in Figure 4.80 and for an actual property in Figure 4.81. Note that the banks have also been shaded. This is however seldom done. This system of identifying toes and crests is recommended by the authors.

Some companies use the opposite system labelling the crests with dashed lines and the toes by solid lines (Fig. 4.82). The Berkeley pit shown in Figure 4.83 is one such example where this system has been applied.

If there are a great number of benches and the scale is large, there can be difficulties in representing both the toes and the crests. Knowing the bench height and the bench face angle

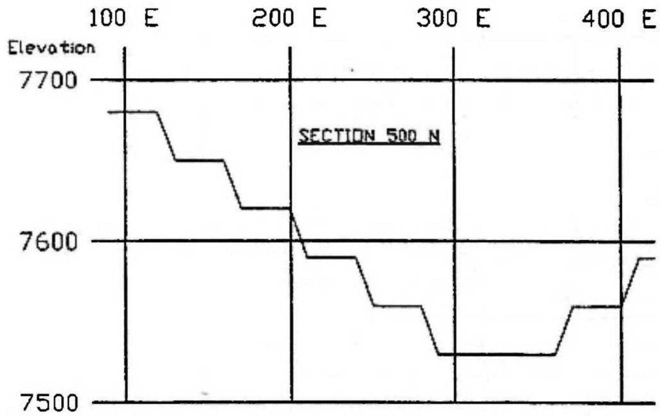


Figure 4.75. Cross-section through an open pit mine.

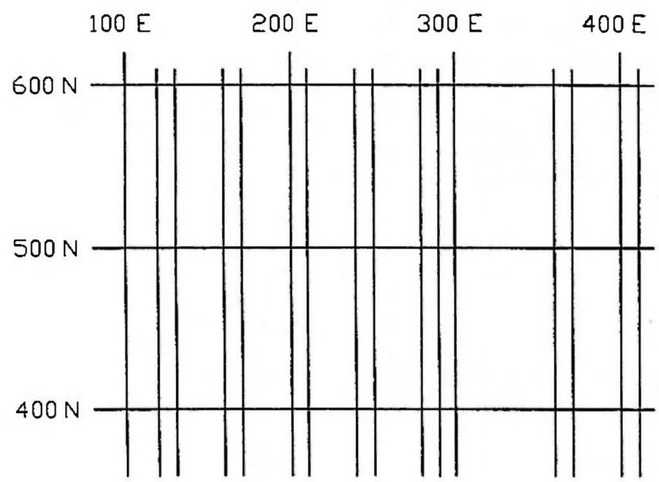


Figure 4.76. Plan view through the portion of the pit shown in cross-section in Figure 4.75 (toes and crests depicted by solid lines).

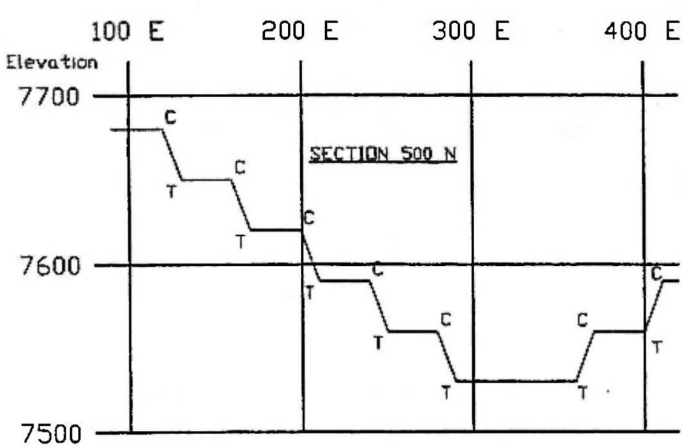


Figure 4.77. Cross-section through a portion of an open pit with toes and crests labelled.

it is a simple matter to construct, if needed, the toes presuming that the crests are given or vice versa. Hence only one set of lines (crests or toes) is actually needed. When only one line is used to represent a bench, the most common technique is to draw the median (mid bench) elevation line at its plan location on the bench face. This is shown in section and plan in Figures 4.84 and 4.85, respectively.

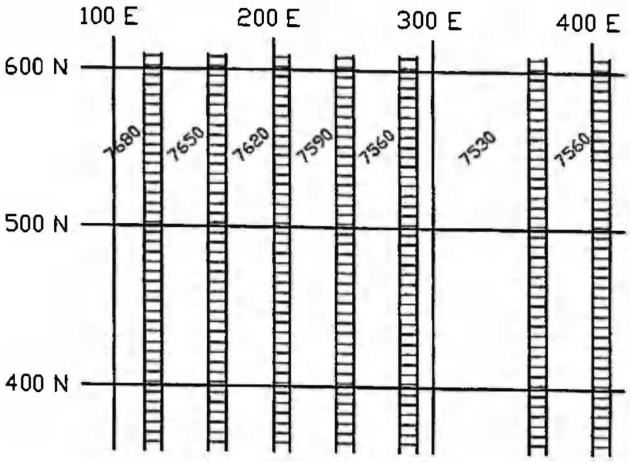


Figure 4.78. Plan view with the bench faces shaded and the flat segment elevations labelled.

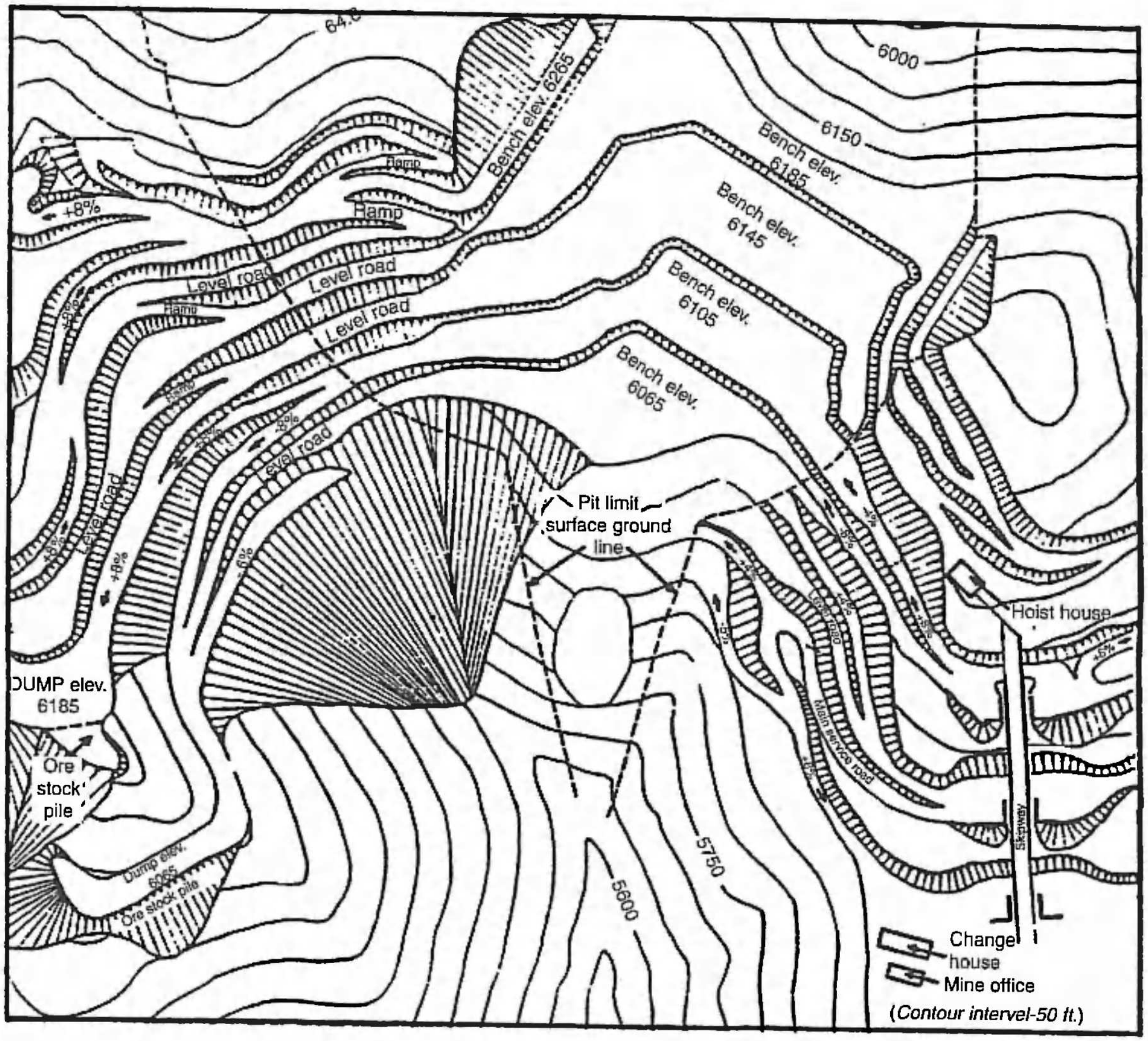


Figure 4.79. Example of slope surface shading described in Figure 4.78 (Ramsey, 1944).

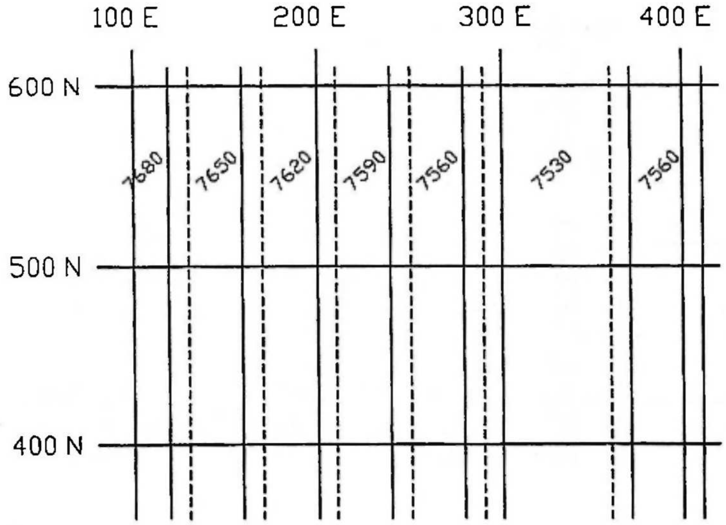


Figure 4.80. Plan view of a portion of the open pit (crests are denoted by solid lines and toes by dashed lines).

An actual example of its use is given in Figure 4.86. An enlarged view of a section of the pit is shown in Figure 4.87. The elevation label is located half way between the median contour lines. This is the actual location for this elevation and corresponds to the bench elevation at that point.

It is a relatively simple matter to go from median lines to actual bench representation (toes and crests) and vice versa. This process is depicted in Figure 4.88. The median contour line in the center will be replaced by the toe-crest equivalent. The road is 100 ft wide and has a grade of 10%. The bench height is 40 ft, the bank width is 30 ft and the width of the safety bench is 50 ft. The process begins by adding the center lines halfway to the next contour lines (Fig. 4.88b). Toe and crest lines are added (Fig. 4.88c) and the edge of road is drawn (Fig. 4.88d). Finally the construction lines are removed (Fig. 4.88e). The reader is encouraged to try this construction going back and forth from toes and crests to median lines.

4.8 ADDITION OF A ROAD

4.8.1 *Introduction*

Roads are one of the more important aspects of open pit planning. Their presence should be included early in the planning process since they can significantly affect the slope angles and the slope angles chosen have a significant effect on the reserves. Most of the currently available computerized pit generating techniques discussed in the following chapter do not easily accommodate the inclusion of roads. The overall slope angles without the roads may be used in the preliminary designs. Their later introduction can mean large amounts of unplanned stripping or the sterilization of some planned reserves. On the other hand a flatter slope angle can be used which includes the road. This may be overly conservative and include more waste than necessary.

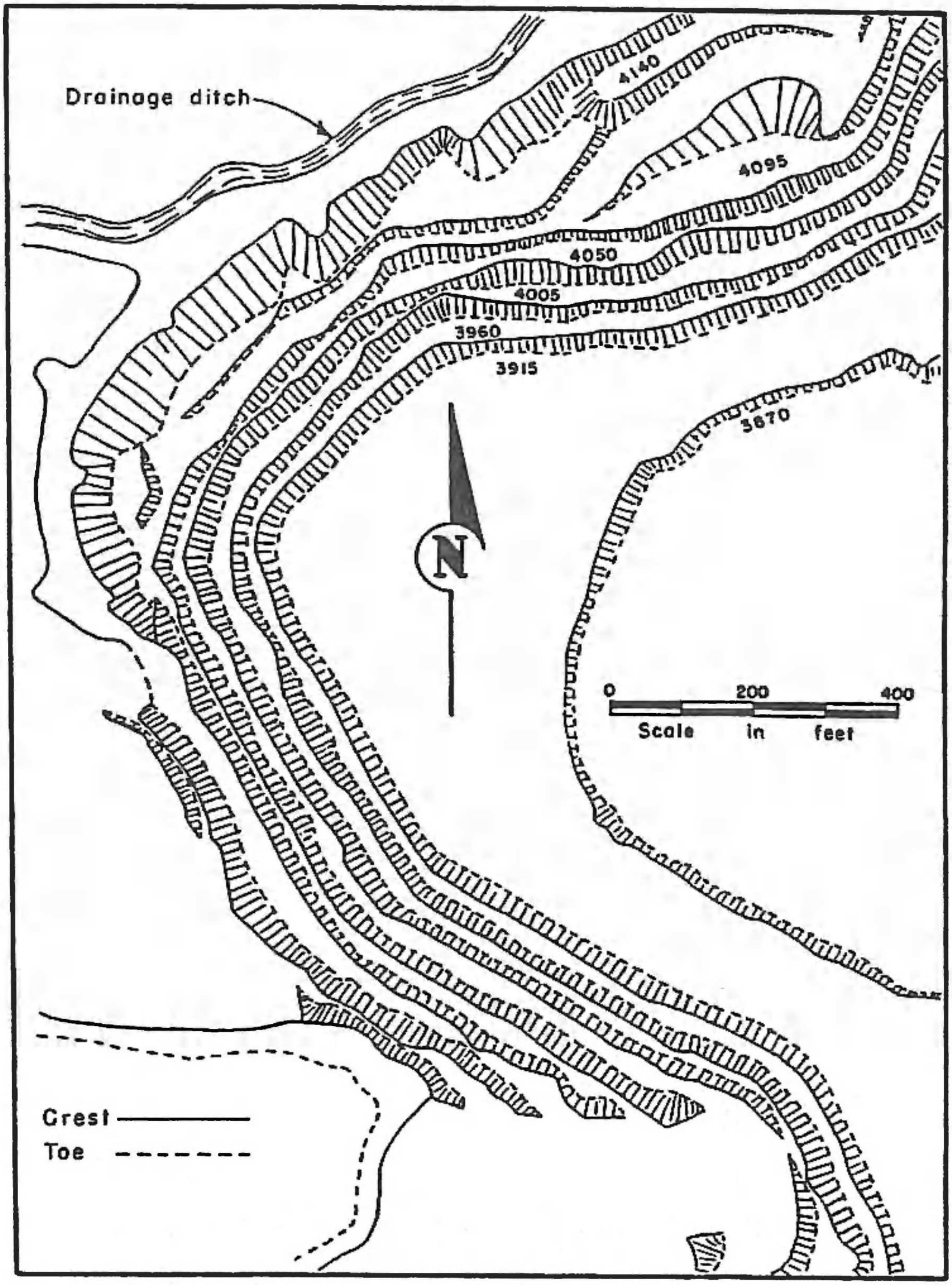


Figure 4.81. Example of the mapping procedure described in Figure 4.80 (Hardwick & Stover, 1960).

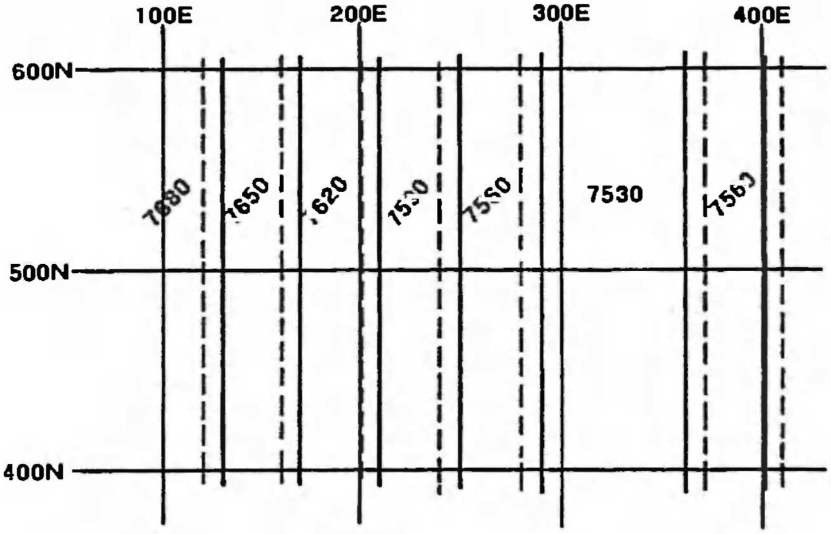


Figure 4.82. Plan view of a portion of the open pit (crests denoted by dashed lines and toes by solid lines).

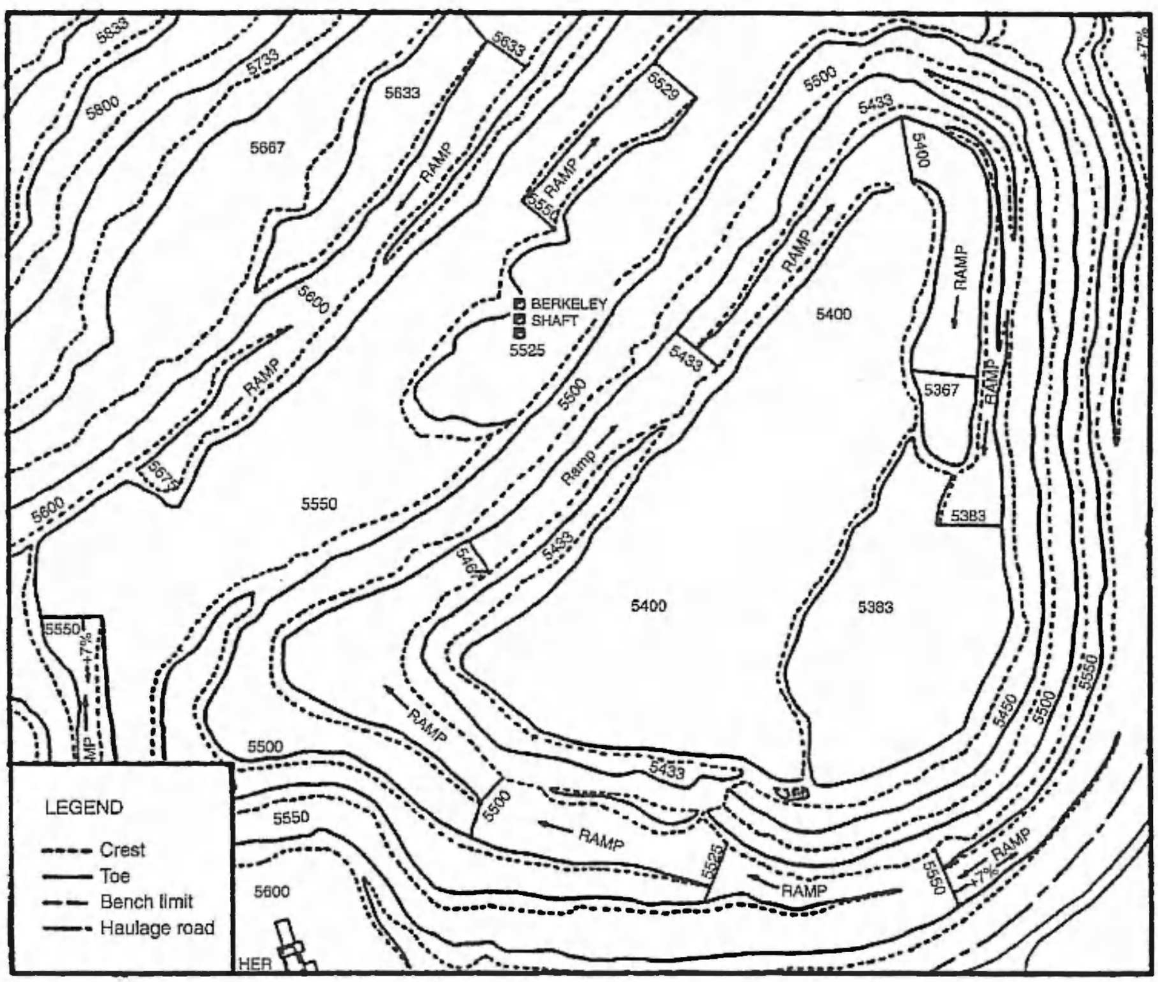


Figure 4.83. Example of the mapping procedure described in Figure 4.82 (McWilliams, 1959).

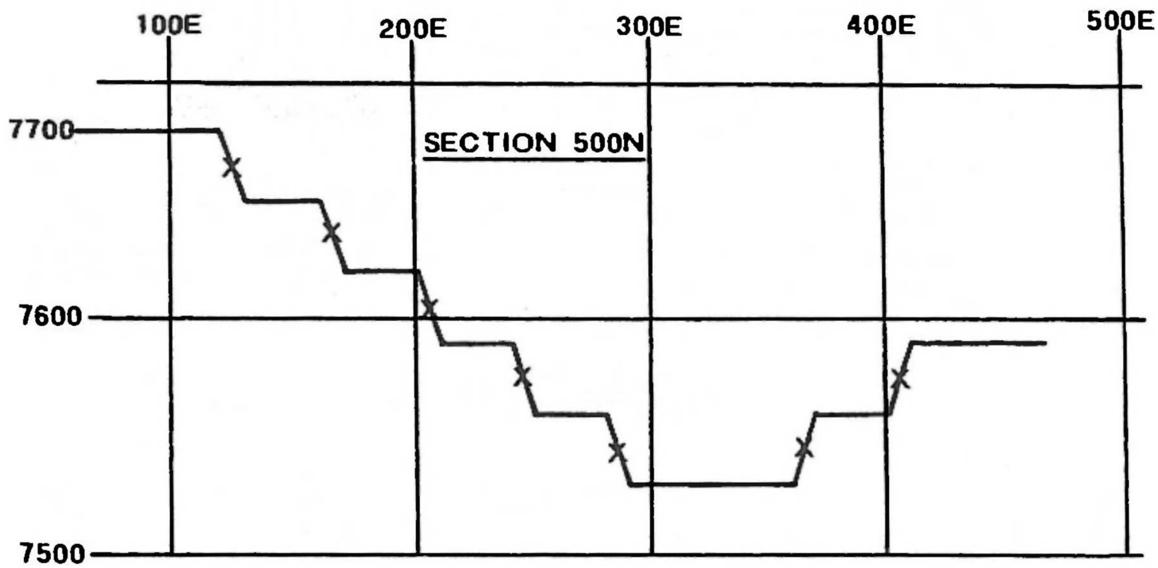


Figure 4.84. Procedure of denoting the median midbench elevation line on the bench face.

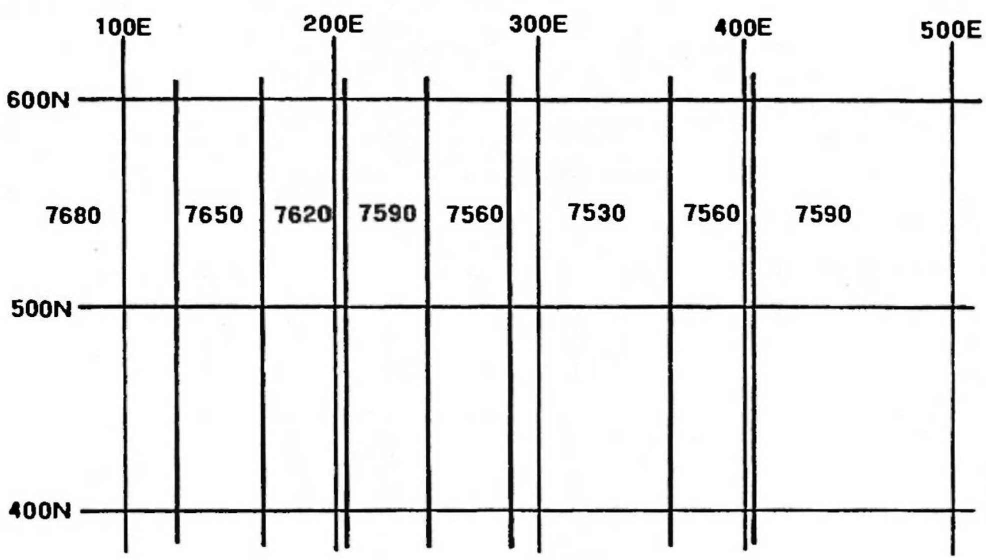


Figure 4.85. Resulting plan view corresponding to the midbench representation of Figure 4.84. The given elevations are bench toe elevations.

Until rather recently, rail haulage was a major factor in open pit operations. Because of the difficulties with sharp turns and steep grades, a great deal of time was spent by mine planners in dealing with track layout and design. Rubber tired haulage equipment has presented great flexibility and ability to overcome many difficulties resulting from inadequate or poor planning in today's pits. However as pits become deeper and the pressure for cost cutting continues, this often neglected area will once again be in focus.

There are a number of important questions which must be answered when siting the roads (Couzens, 1979).

1. The first decision to be taken is where the road exit or exits from the pit wall will be. This is dependent upon the crusher location and the dump points.

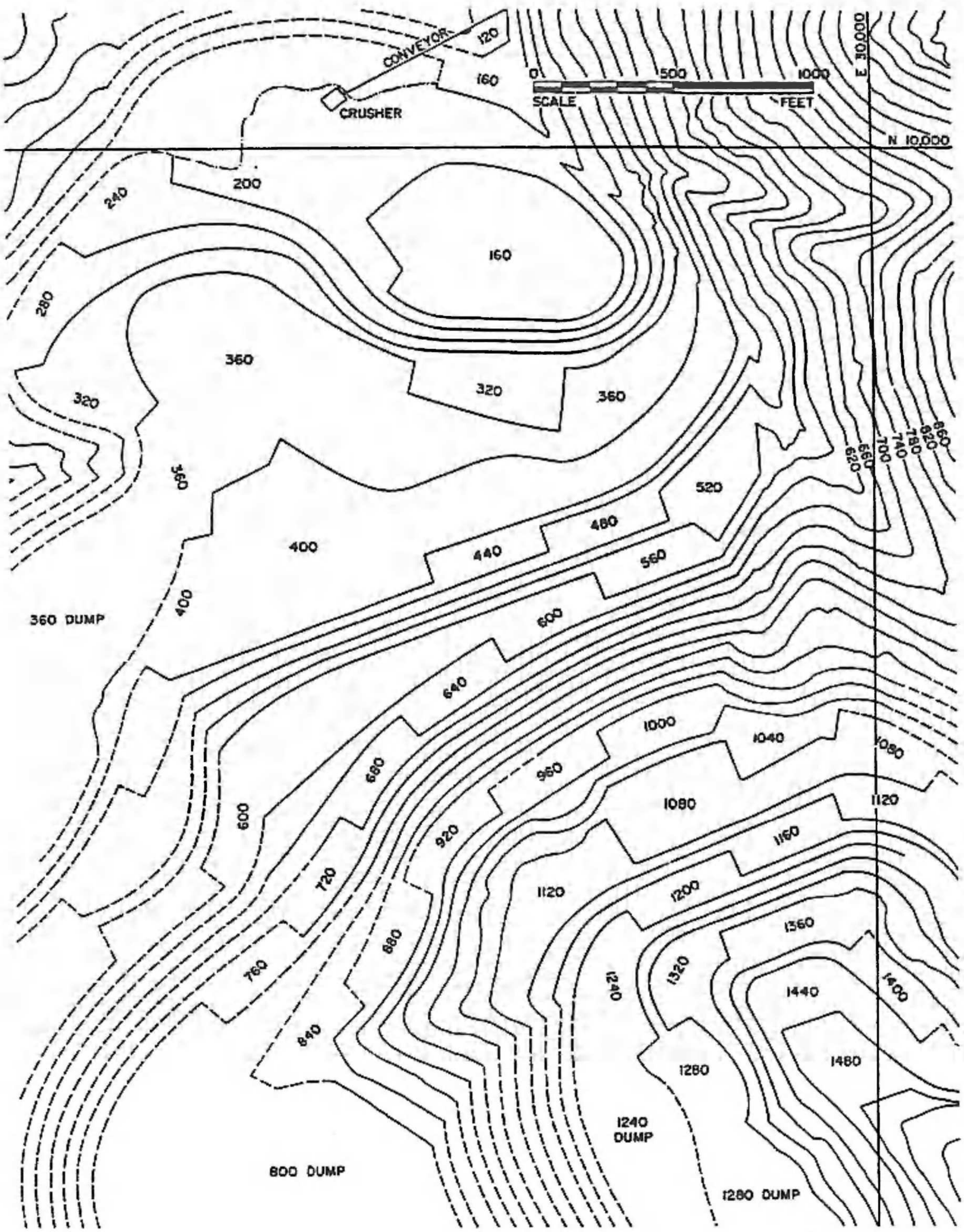


Figure 4.86. Example mining plan composite map based on midbench contours (Couzens, 1979).

2. Should there be more than one means of access? This allows certain flexibility of operation but the cost of added stripping can be high.

3. Should the roads be external or internal to the pit? Should they be temporary or semi-permanent?

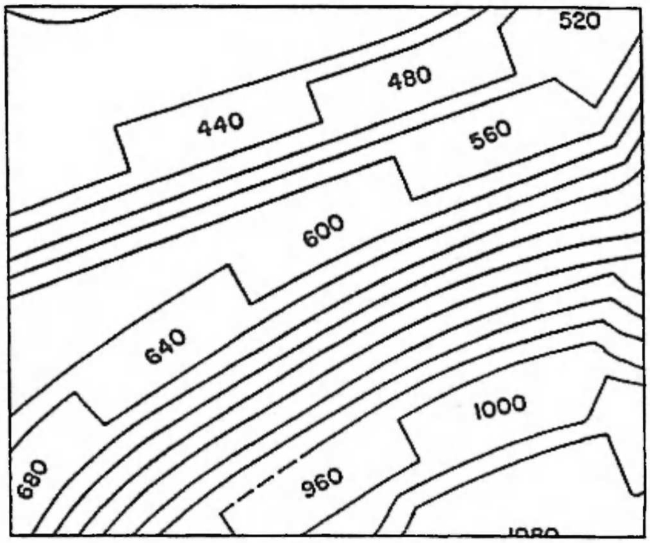


Figure 4.87. An enlarged portion of Figure 4.86 (Couzens, 1979).

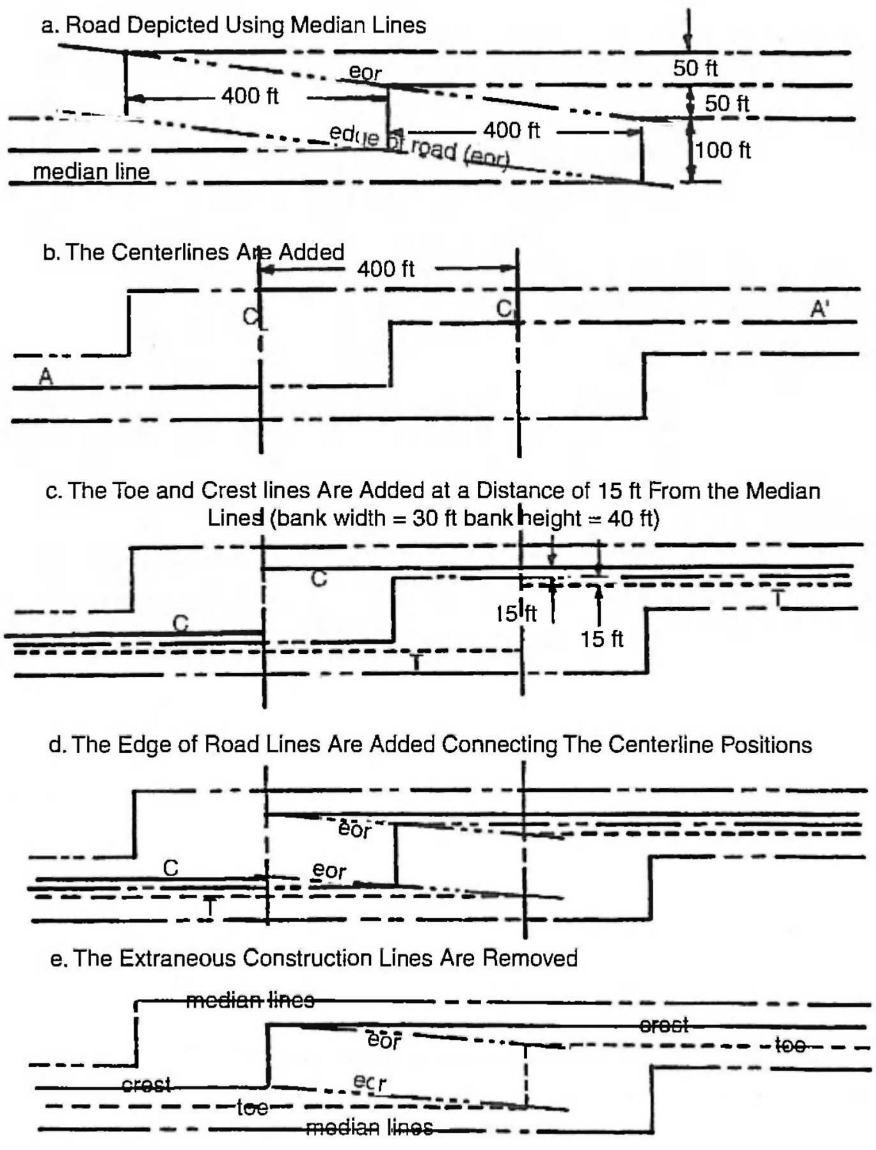


Figure 4.88. Going from midbench contours to a toe and crest representation.

4. Should the road spiral around the pit? Have switchbacks on one side? Or a combination?

5. How many lanes should the road have? The general rule of thumb for 2 way traffic is: road width $\geq 4 \times$ truck width. Adding an extra lane to allow passing may speed up the traffic and therefore productivity but at an increased stripping cost.

6. What should the road grade be? A number of pits operate at 10% both favorable and unfavorable to the haul. A grade of 8% is preferable since it provides more latitude in building the road and fitting bench entries. That is, providing it does not cause too much extra stripping or unduly complicate the layout.

7. What should be the direction of the traffic flow? Right hand or left hand traffic in the pit?

8. Is trolley assist for the trucks a viable consideration? How does this influence the layout?

This section will not try to answer these questions. The focus will be on the procedure through which haulroad segments can be added to pit designs. The procedures can be done by hand or with computer assist. Once the roads have been added then various equipment performance simulators can be applied to the design for evaluating various options.

4.8.2 *Design of a spiral road – inside the wall*

As has been discussed in Section 4.3, the addition of a road to the pit involves moving the wall either into the pit and therefore losing some material (generally ore) or outward and thereby adding some material (generally waste). This design example considers the first case (inside the original pit wall). The second case will be discussed in the following section. This pit consists of the four benches whose crests are shown in Figure 4.89. Both toes and crests are shown in Figure 4.90. The crest-crest dimension is 60 ft, the bench height is 30 ft

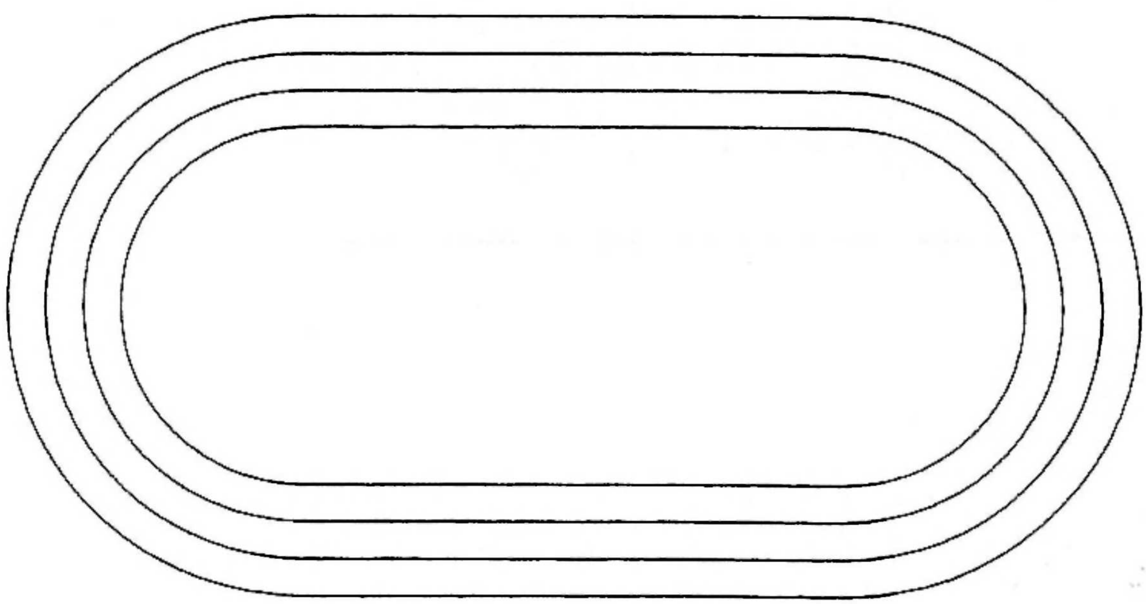


Figure 4.89. The four bench pit with crests shown.

and a road having a width of 90 ft and a grade of 10% is to be added to the north wall. The bench face has an angle of 56°.

Step 1. The design of this type of road begins at the pit bottom. For reasons to be discussed later, the point where the ramp meets the first crest line is selected with some care. In this case, the ramp will continue down to lower mining levels along the north and east walls, thus point A in Figure 4.91 has been selected.

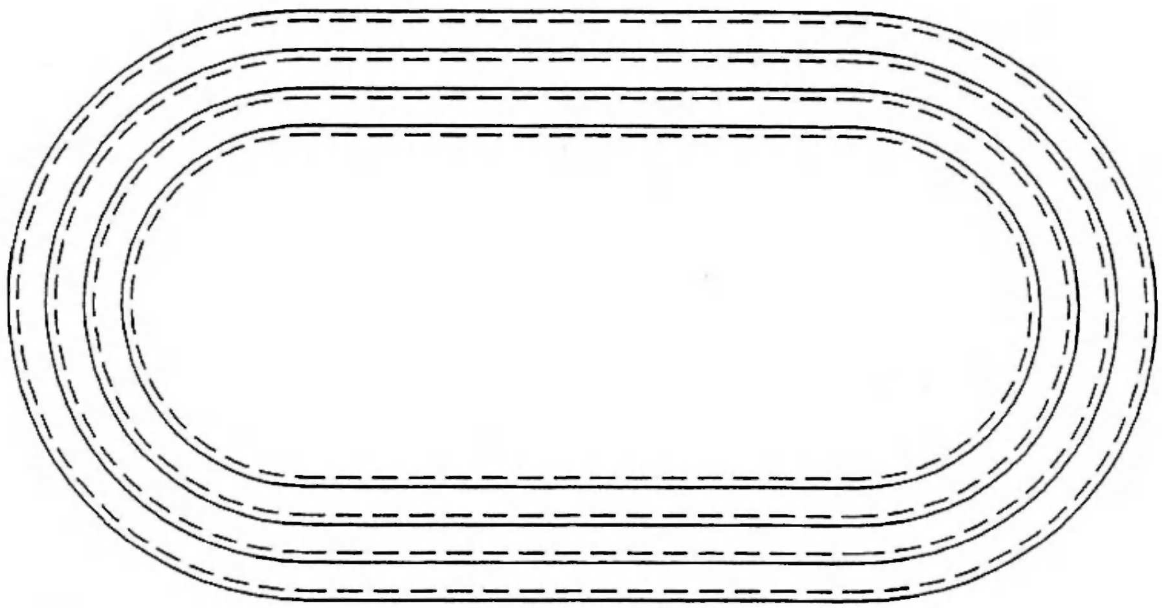


Figure 4.90. The four bench pit with toes added.

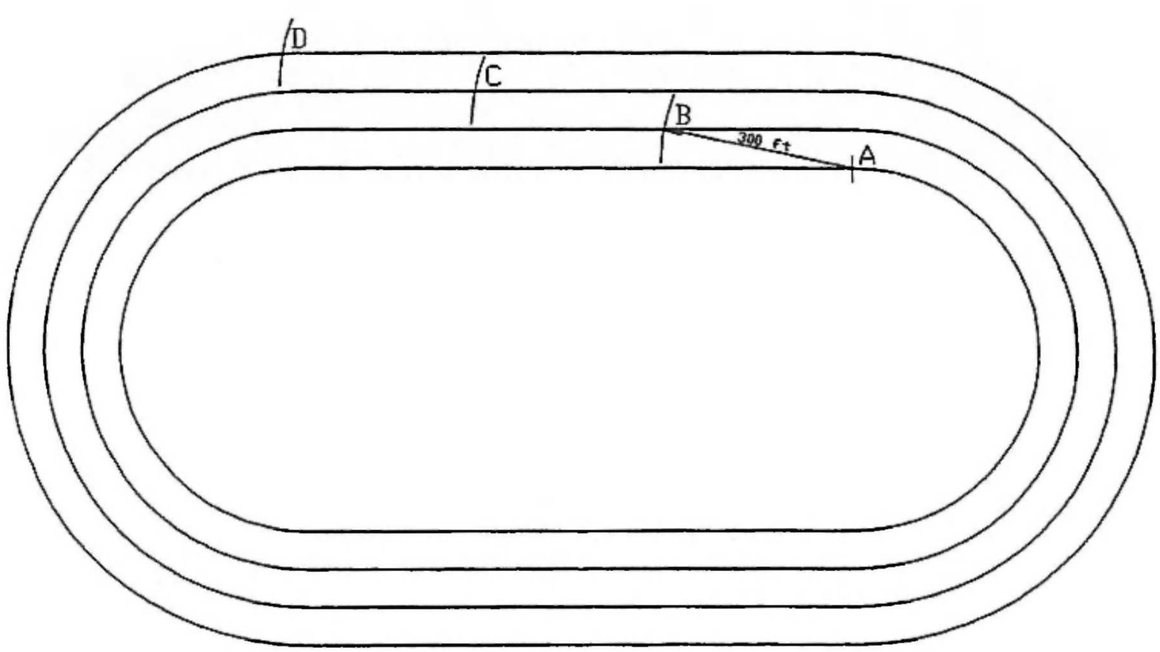


Figure 4.91. Point of ramp initiation and crest intercepts.

Step 2. The locations where the ramp meets the succeeding crests are now determined. Since the bench height H is 30 ft and the road grade G is 10%, the horizontal distance D travelled by a truck going up to the next level is

$$D = \frac{100H}{G(\%)} = \frac{100 \times 30}{10} = 300\text{ ft}$$

Point B on the crest of the next bench is located by measuring the 300 ft distance with a ruler or by swinging the appropriate arc with a compass. Points C and D are located in a similar way.

Step 3. The crest line segments indicating the road location will be added at right angles to the crest lines rather than at right angles to the line of the road. Hence they have a length (W_a) which is longer than the true road width (W_t). As can be seen in Figure 4.92, the angle (Θ) that the road makes with the crest lines is

$$\Theta = \sin^{-1}\frac{600}{300} = 11.5°$$

Hence the apparent road width W_a (that which is laid out), is related to the true road width by

$$W_a = \frac{W_t}{\cos\Theta} = 1.02\,W_t = 1.02 \times 90 = 92\text{ ft}$$

For most practical purposes, little error results from using

$$W_a \cong W_t = W$$

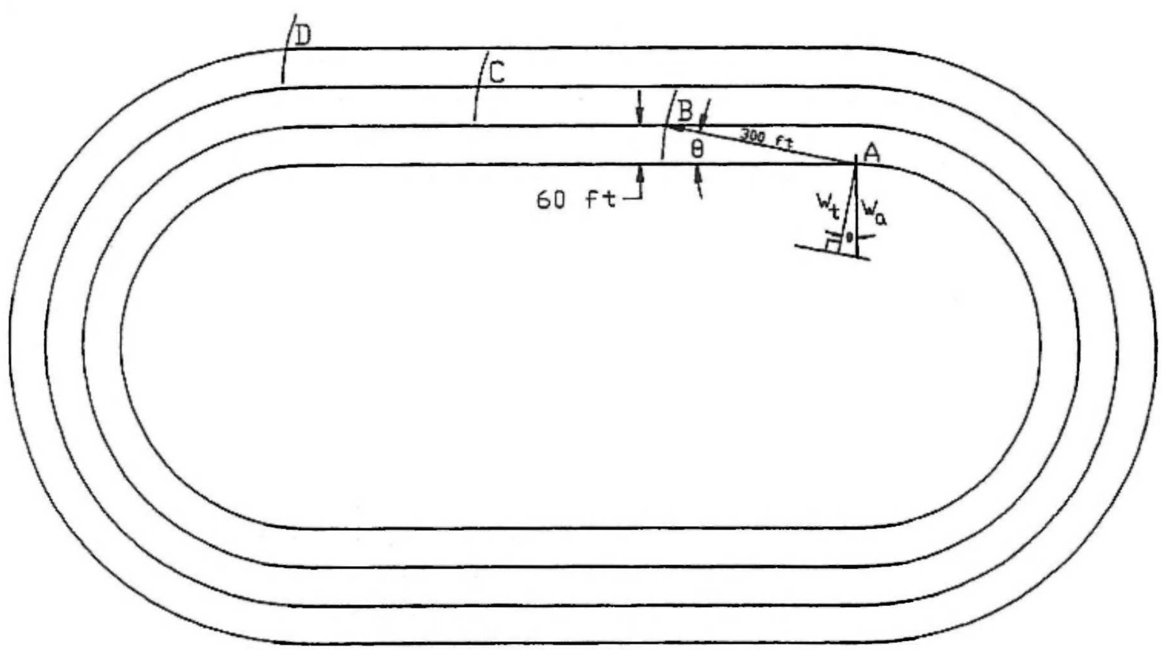


Figure 4.92. Addition of ramp width (Step 3).

Lines of length W drawn perpendicular to the crest lines from points A, B, C and D have been added to Figure 4.93a. In addition short lines running parallel to the crest starting at the ends of these lines have been added. Line a-a' is one such line.

Step 4. Line a-a' is extended towards the west end of the pit. It first runs parallel to the previous crest line but as the pit end approaches it is curved to make a smooth transition with the original crest line. This is shown in Figure 4.93b. The designer has some flexibility

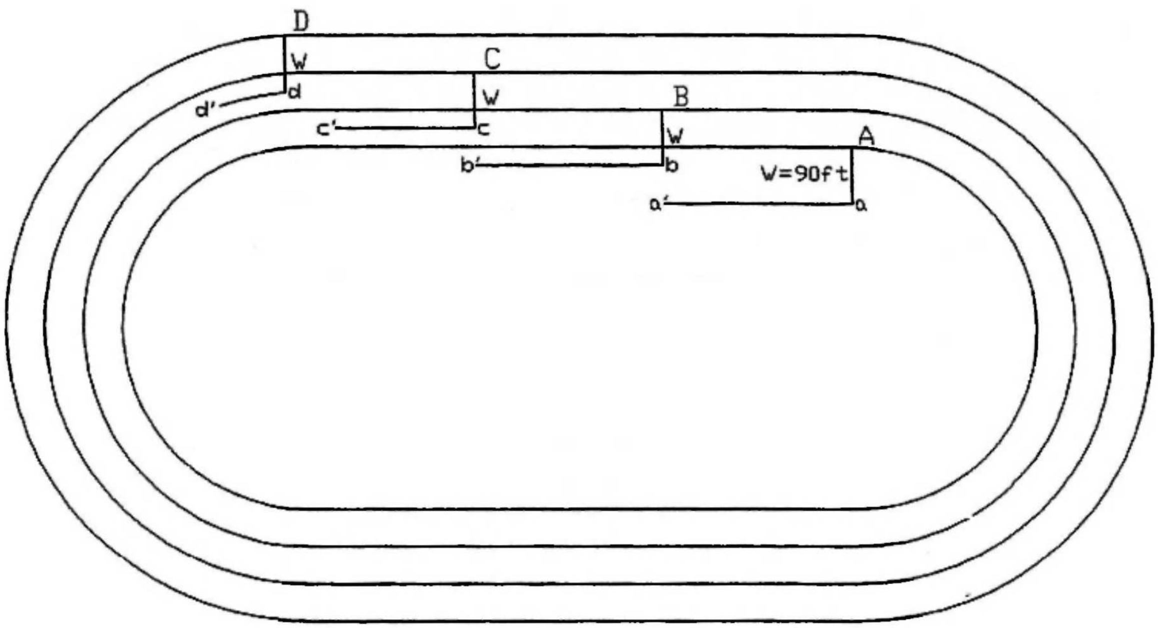


Figure 4.93a. Completing the new crest lines (Step 4).

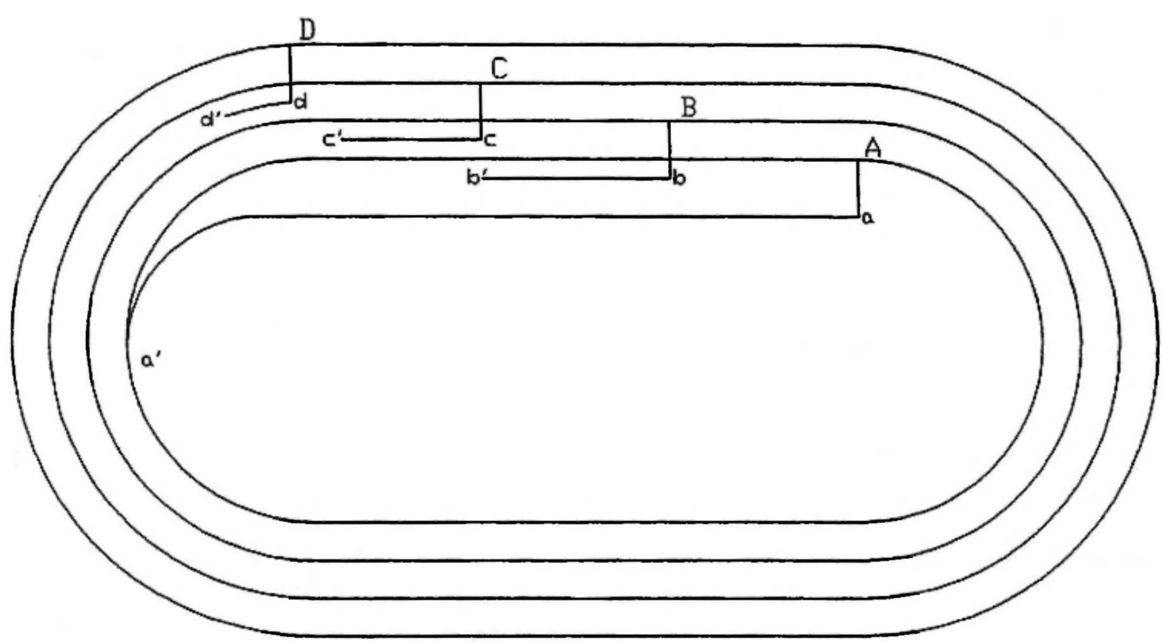


Figure 4.93b. Completing the new crest lines (Step 4).

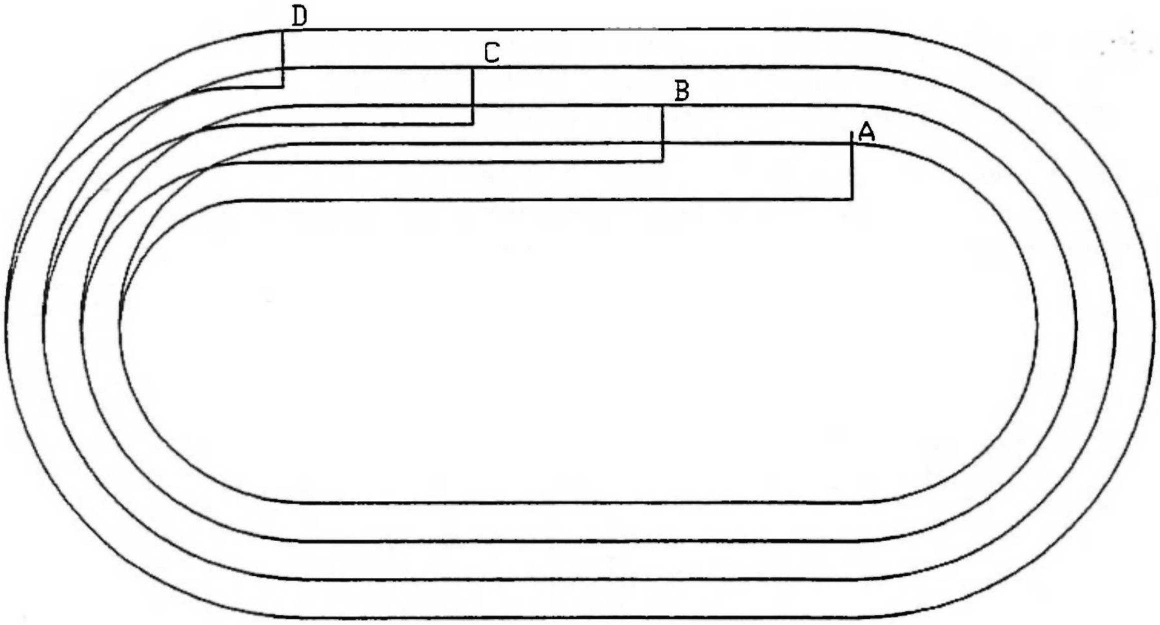


Figure 4.93c. Completing the new crest lines (Step 4).

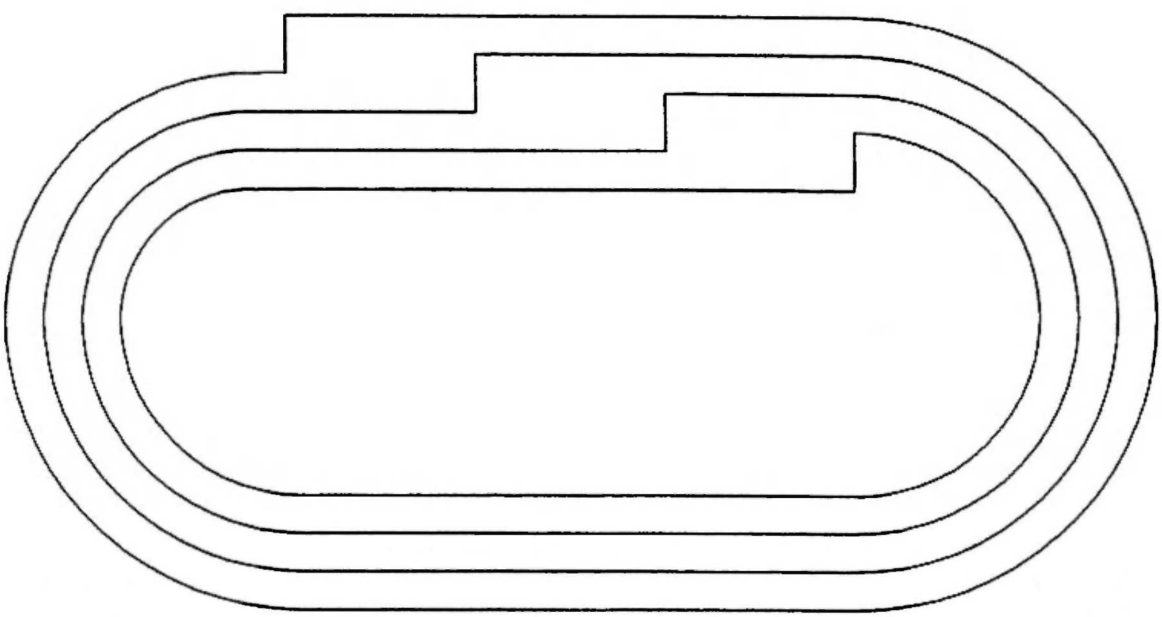


Figure 4.94. The pit as modified by the ramp (Step 5).

on how this transition occurs. Once this decision is made then the remaining crest lines are drawn parallel to this first one. The results are shown in Figure 4.93c.

Step 5. The extraneous lines remaining from the original design are now removed. The resulting crest lines with the included ramp are shown in Figure 4.94.

Step 6. The ramp is extended from the crest of the lowest bench to the pit bottom. This is shown in Figure 4.95. The toe lines have been added to assist in this process. In Figure 4.95,

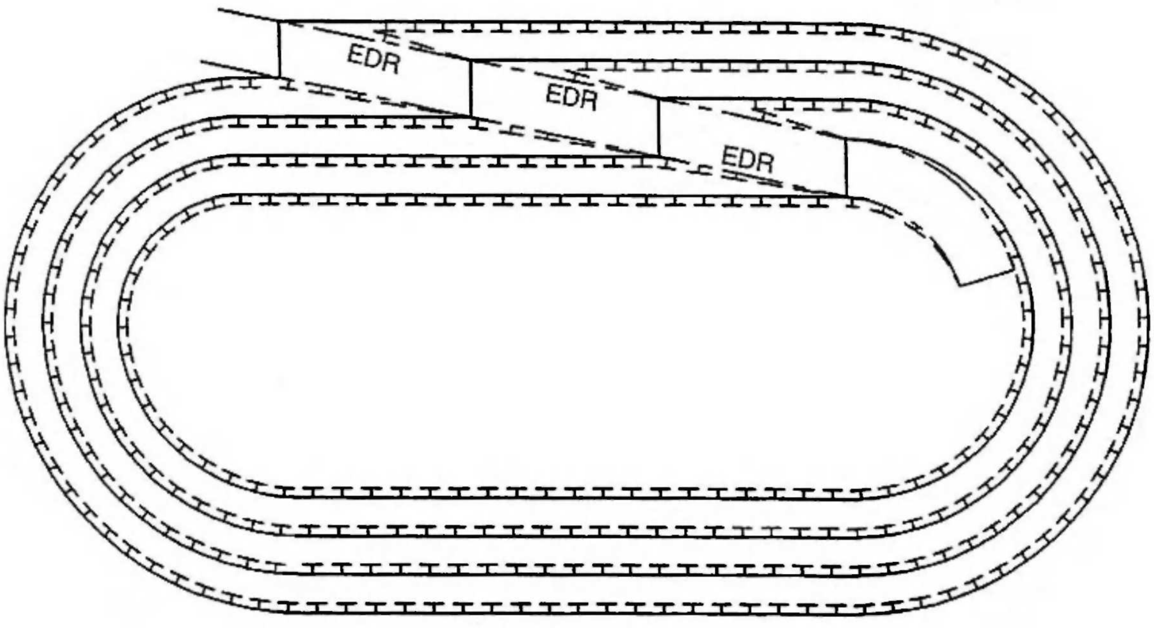


Figure 4.95. Addition of entrance ramp and toe lines (Step 6).

the slopes have been shaded to help in the visualization. The edge of road (EOR) lines shown are also crest lines.

4.8.3 *Design of a spiral ramp – outside the wall*

In the previous section the addition of a spiral ramp lying inside the original pit contours was described. It's addition meant that some material initially scheduled for mining would be left in the pit. For the case described in this section where the ramp is added outside the initial pit shell design, additional material must be removed. The same four bench mine as described earlier will be used:

Bench height = 30 ft
Crest-crest distance = 60 ft
Road width = 90 ft
Road grade = 10%
Bench slope angle = 56°

Step 1. The design process begins with the crest of the uppermost bench. A decision must be made regarding the entrance point for the ramp as well as direction. As shown in Figure 4.96, the entrance should be at point A in the direction shown. Mill and dump locations are prime factors in selecting the ramp entrance point. From this point an arc of length L equal to the plan projection of the ramp length between benches is struck. This locates point B. From point B an arc of length L is struck locating point C, etc.

Step 2. From each of the intersection points A, B, C and D, lines of length W_a (apparent road width) are constructed normal to their respective crest lines. This is shown in

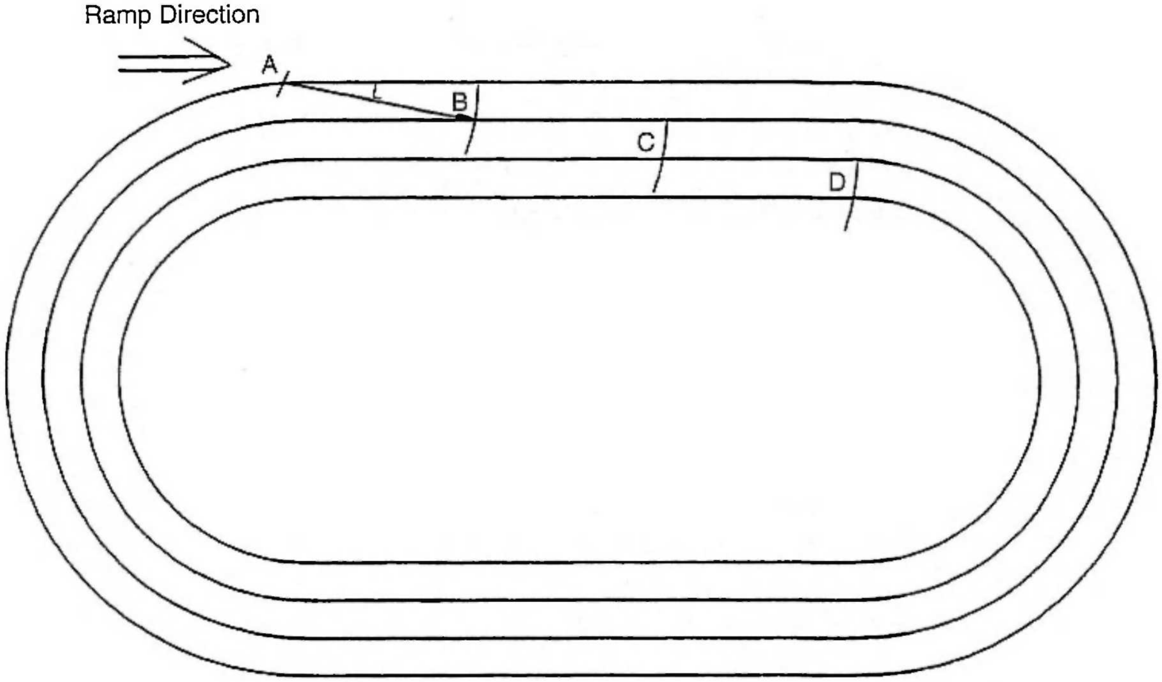


Figure 4.96. Point of ramp initiation and crest intercepts (Step 1).

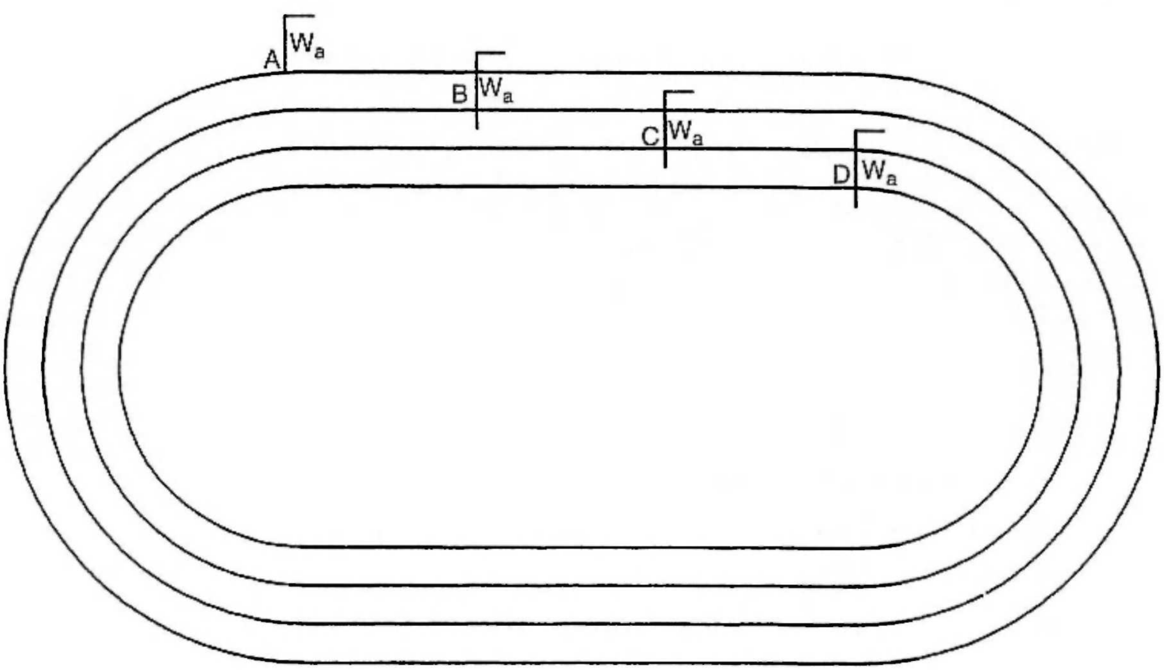


Figure 4.97. Addition of ramp width (Step 2).

Figure 4.97. A short length of line is drawn parallel to the crest line from the end in the ramp direction.

Step 3. Beginning with the lowermost crest, a smooth curve is drawn connecting the new crest with the old. This is shown in Figure 4.98.

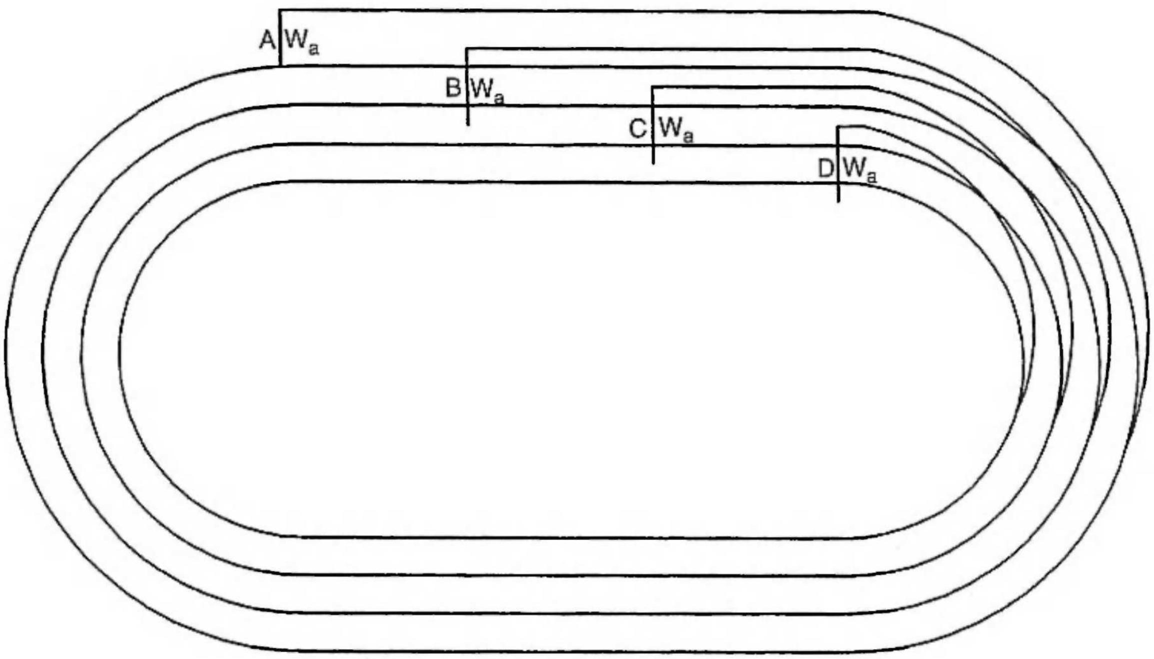


Figure 4.98. Drawing the new crest lines (Steps 3 and 4).

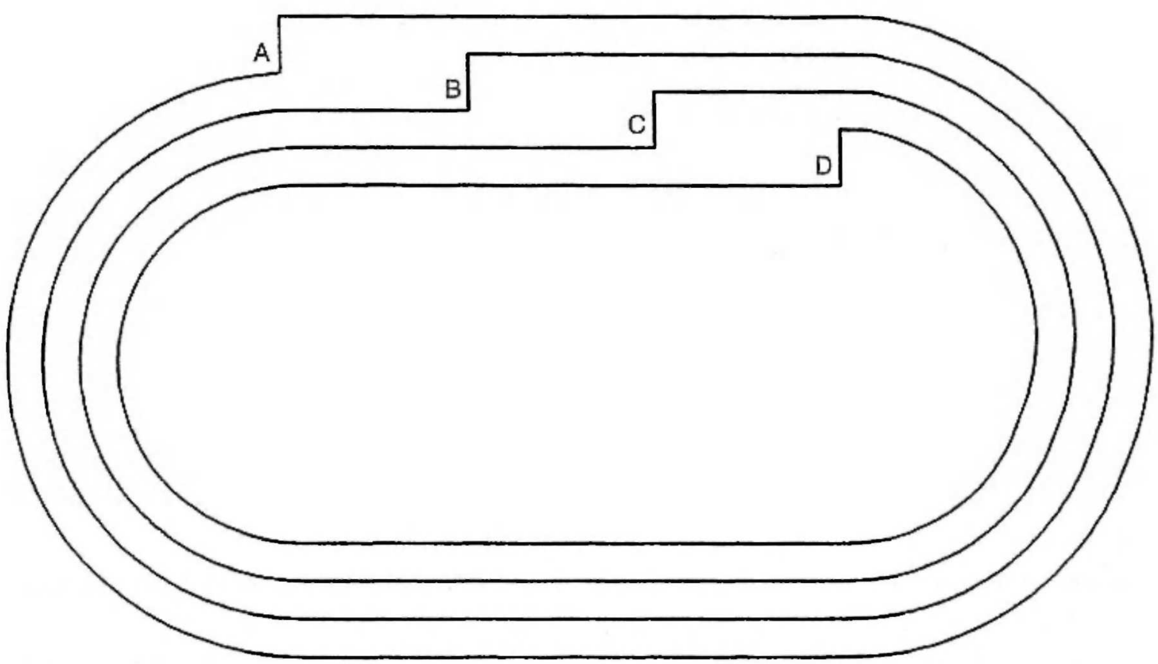


Figure 4.99. The pit as modified by the ramp (Step 5).

Step 4. The remaining new crest line portions are drawn parallel to the first crest working upwards from the lowest bench.

Step 5. The extraneous lines are removed from the design (Fig. 4.99).

Step 6. The toe lines at least for the lowest bench are added and the ramp to the pit bottom added. In Figure 4.100, the slopes have been shaded to assist in viewing the ramp.

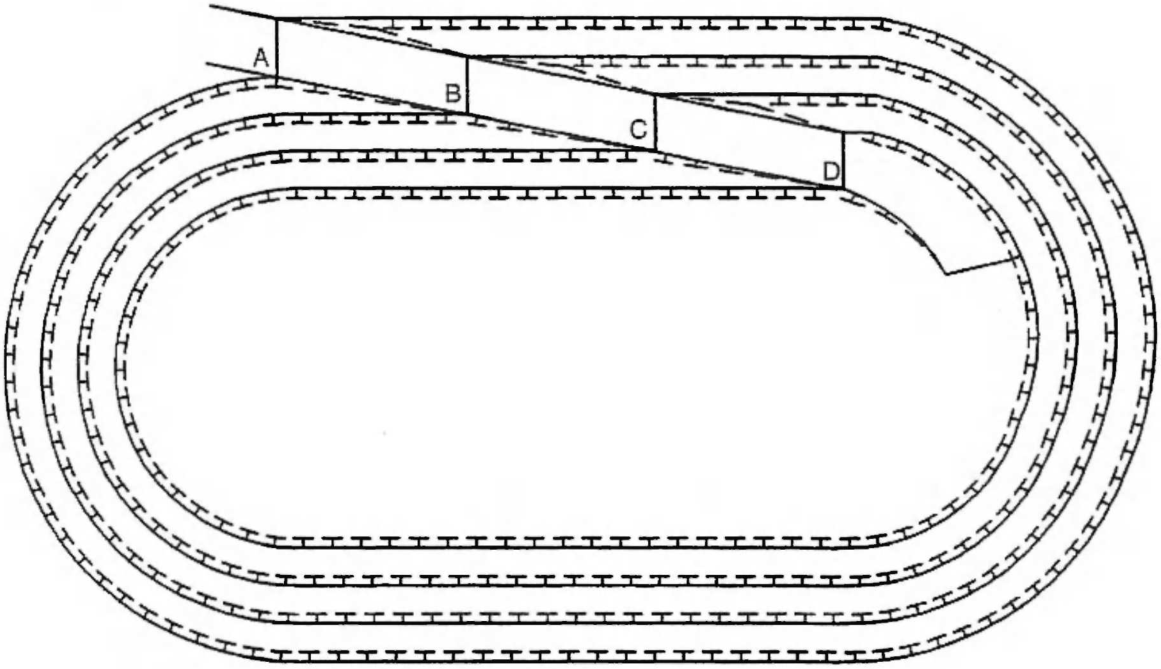


Figure 4.100. Addition of entrance ramp and toe lines (Step 6).

4.8.4 *Design of a switchback*

In laying out roads the question as to whether to:
 (a) spiral the road around the pit,
 (b) have a number of switchbacks on one side of the pit, or
 (c) use some combination.

Generally (Couzens, 1979) it is desirable to avoid the use of switchbacks in a pit. Switchbacks:
- tend to slow down traffic,
- cause greater tire wear,
- cause various maintenance problems,
- probably pose more of a safety hazard than do spiral roads (vision problems, machinery handling, etc.).

Sometimes the conditions are such that switchbacks become interesting:
- when there is a gently sloping ore contact which provides room to work in switchbacks at little stripping cost;
- it may be better to have some switchbacks on the low side of the pit rather than to accept a lot of stripping on the high side.

The planner must take advantage of such things. The general axiom should be to design the pit to fit the shape of the deposit rather than vice versa. If switchbacks are necessary the planner should:
- leave enough length at the switchbacks for a flat area at the turns so that trucks don't have to operate on extremely steep grades at the inside of curves,
- consider the direction of traffic,
- consider problems the drivers may have with visibility,
- consider the effect of weather conditions on the design (ice, heavy rain, etc.).

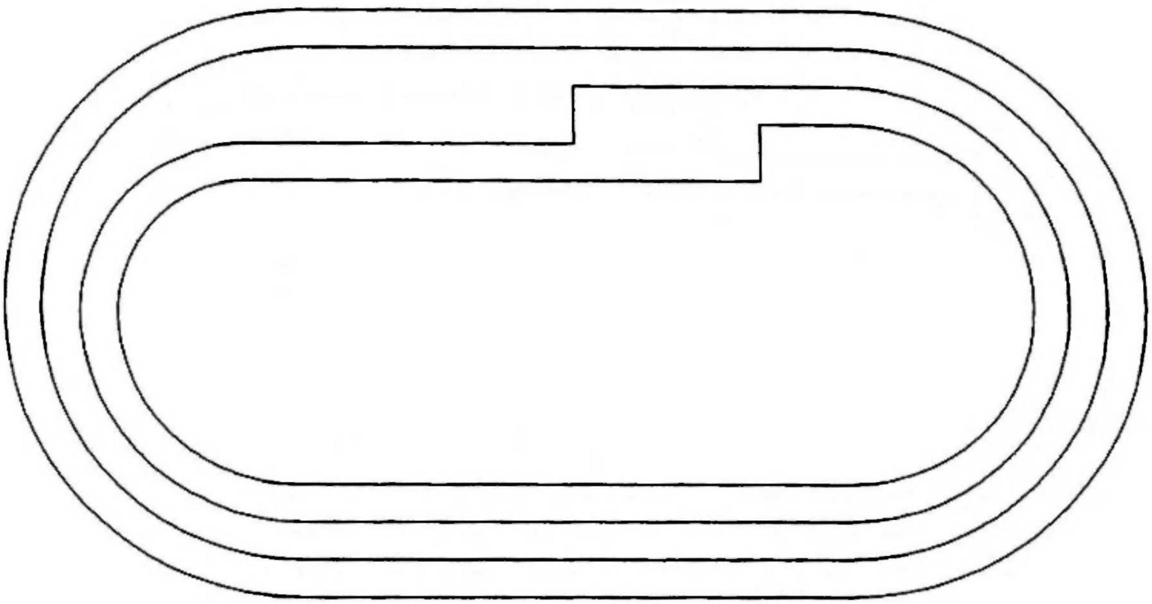


Figure 4.101. The starter pit for switchback addition to the north wall (Step 1).

In this section the steps required to add a switchback to the pit shown in Figure 4.89 will be described. The switchback will occur between the second and third benches on the north pit wall.

Step 1. The design will begin from the pit bottom. In this case the ramp moves into the as-designed pit wall. Figure 4.101 shows the modified pit with the crest lines drawn for benches 4 (lowermost) and 3. This is the same procedure as with the spiral ramp. The bench height has been selected as 30 ft and the road gradient is 10%. Hence the plan distance R is 300 ft.

Step 2. The center C used to construct the switchback is now located as shown in Figure 4.102. There are three distances involved L_1, L_2 and L_3. L_2 is the given crest-crest distance. Distances L_1 and L_3 must now be selected so that

$$L_1 + L_3 = R - L_2$$

In this particular case $L_1 = 0.5R = 150$ ft. Since $L_2 = 60$ ft, then $L_3 = 90$ ft. The center C is located at $L_2/2 = 30$ ft from the 3 construction lines. A vertical line corresponding to road width W is drawn at the end of L_3.

Step 3. In Figure 4.103 the curve with radius $R_2 = L_2/2$ is drawn from C. This becomes the inner road radius. It should be compared with the turning radius for the trucks being used. A second radius $R_3 = 2W$ is also drawn from C. The intersection of this curve with the horizontal line drawn from C becomes a point on the bench 2 crest. It is noted that actual designs may use values of R_3 different from that recommended here. This is a typical value. Portions of the bench 2 crest lines have been added at the appropriate distances.

Step 4. A smooth curve is now added going from line $a - b$ through crest point CP to line $c - d$. The designer can use some judgement regarding the shape of this transition line.

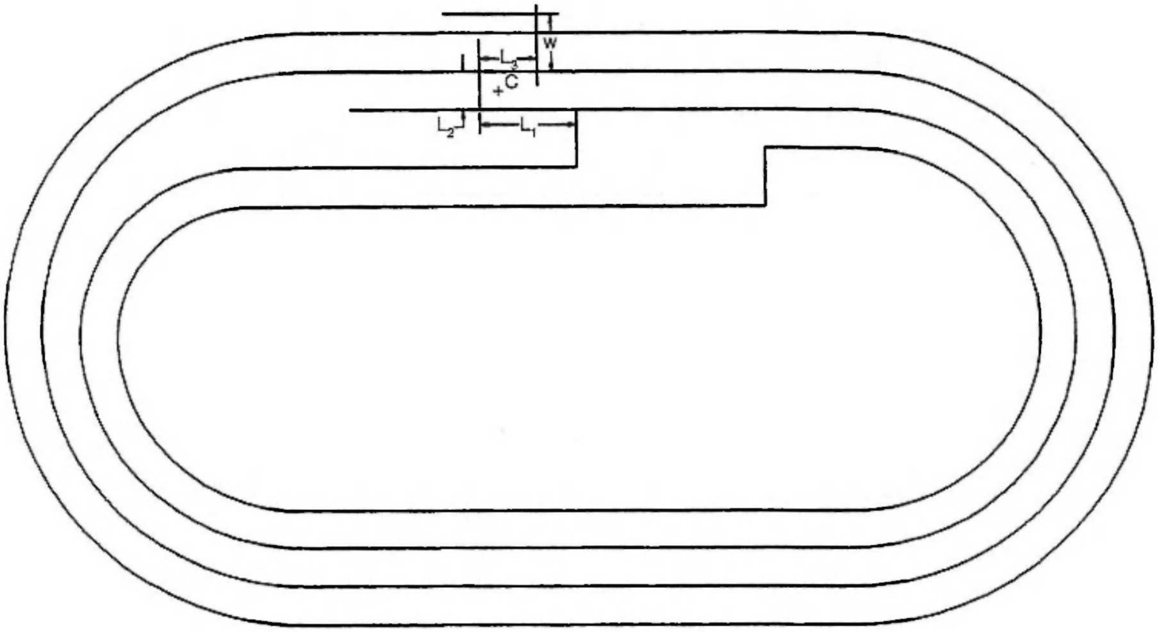


Figure 4.102. Construction lines for drawing the switchback (Step 2).

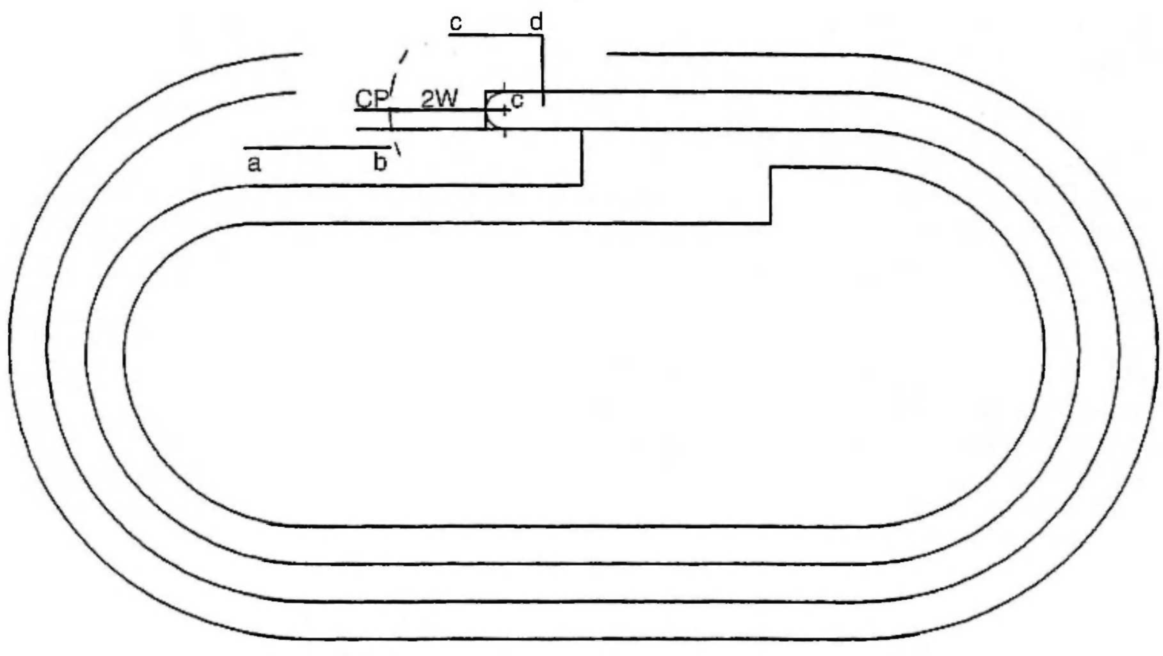


Figure 4.103. Crest lines and the crest point for bench 2 (Step 3).

Figure 4.104 shows the results. The lines surrounding point C simply represent edge of road (EOR).

Step 5. The crest line for bench 1 is then added parallel to that drawn for bench 2 (Fig. 4.105).

Step 6. The final crest line representation of the pit is drawn (Fig. 4.106). As can be seen the switchback occupies a broad region over a relatively short length. Thus it can be logically placed in a flatter portion of the overall pit slope.

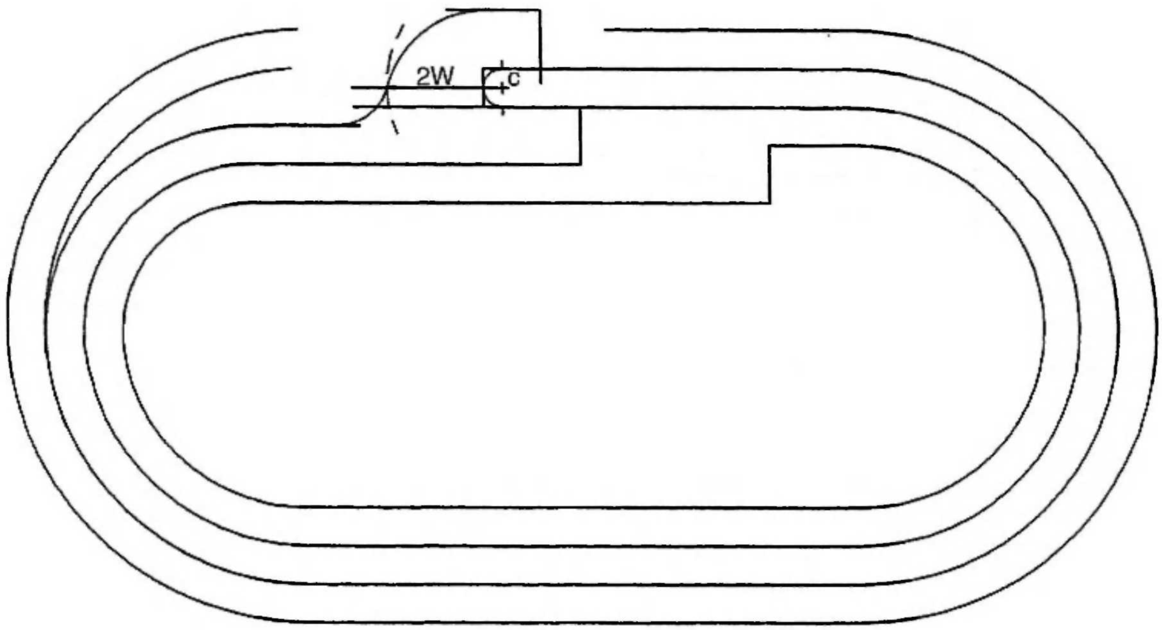


Figure 4.104. The transition curve has been added (Step 4).

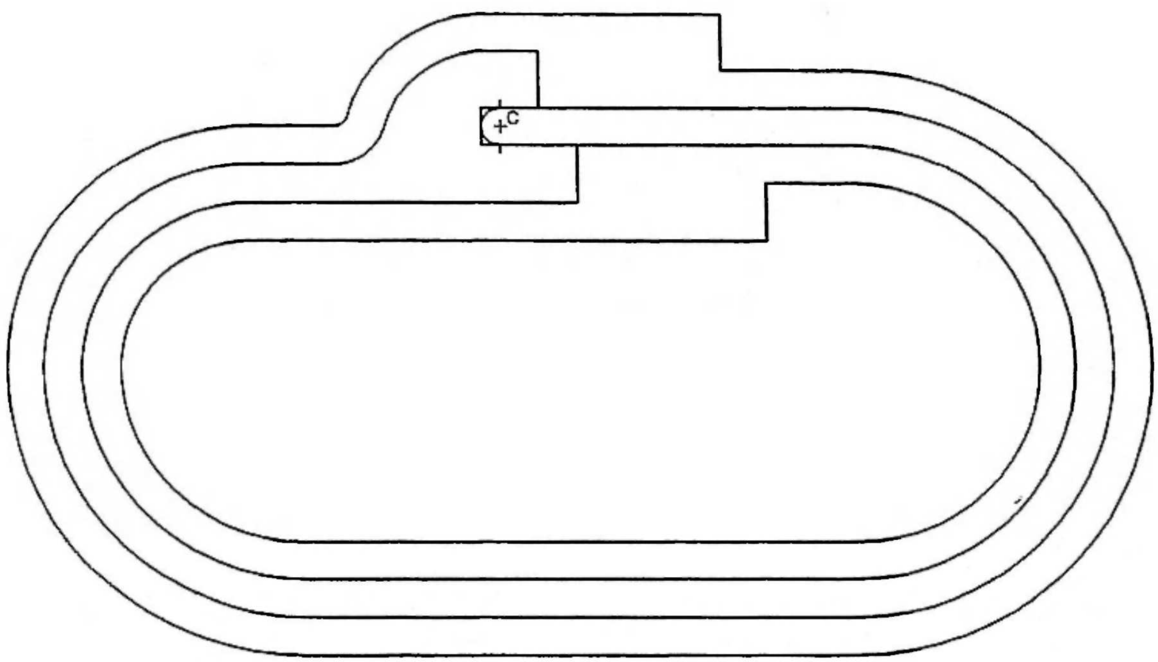


Figure 4.105. The crest line for bench 1 is added (Step 5).

Step 7. The toes are drawn and the lower section of the ramp (between bench 4 crest and the pit floor) added (Fig. 4.107).

Two examples of switchbacks are shown in Figure 4.108.

4.8.5 *The volume represented by a road*

The addition of a haulroad to a pit results in a large volume of extra material which must be removed or a similar volume in the pit which is sterilized (covered by the road). Thus even

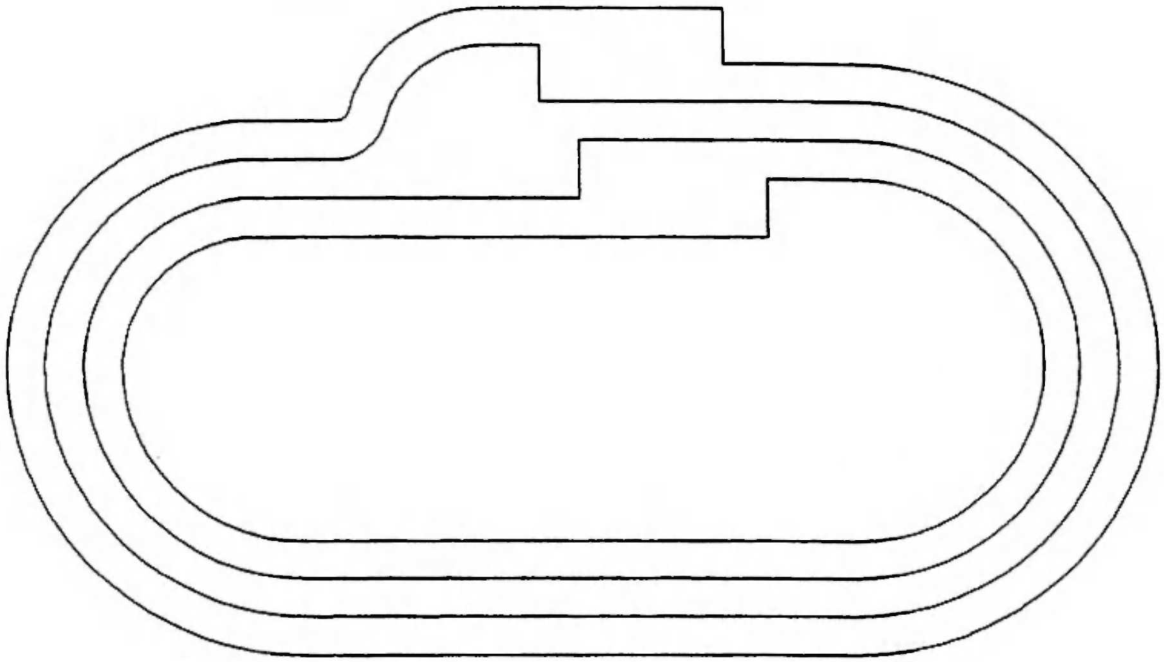


Figure 4.106. The final pit crest lines with the switchback (Step 6).

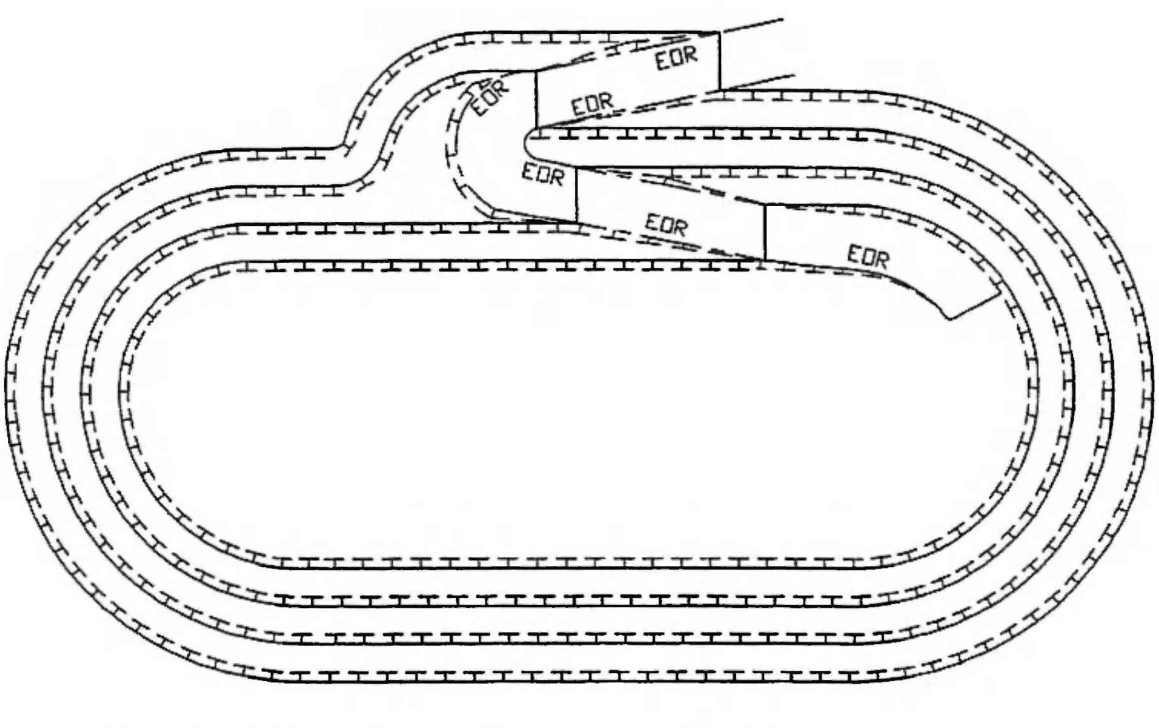


Figure 4.107. The lower entrance ramp and the toes are added (Step 7).

though production flexibility can be improved and the security of having several accesses to the pit can lead to other savings such as steeper interramp slopes, the additional haul roads are associated with significant expense. To demonstrate this, consider the pit shown in Figure 4.109 which contains no haulroad.

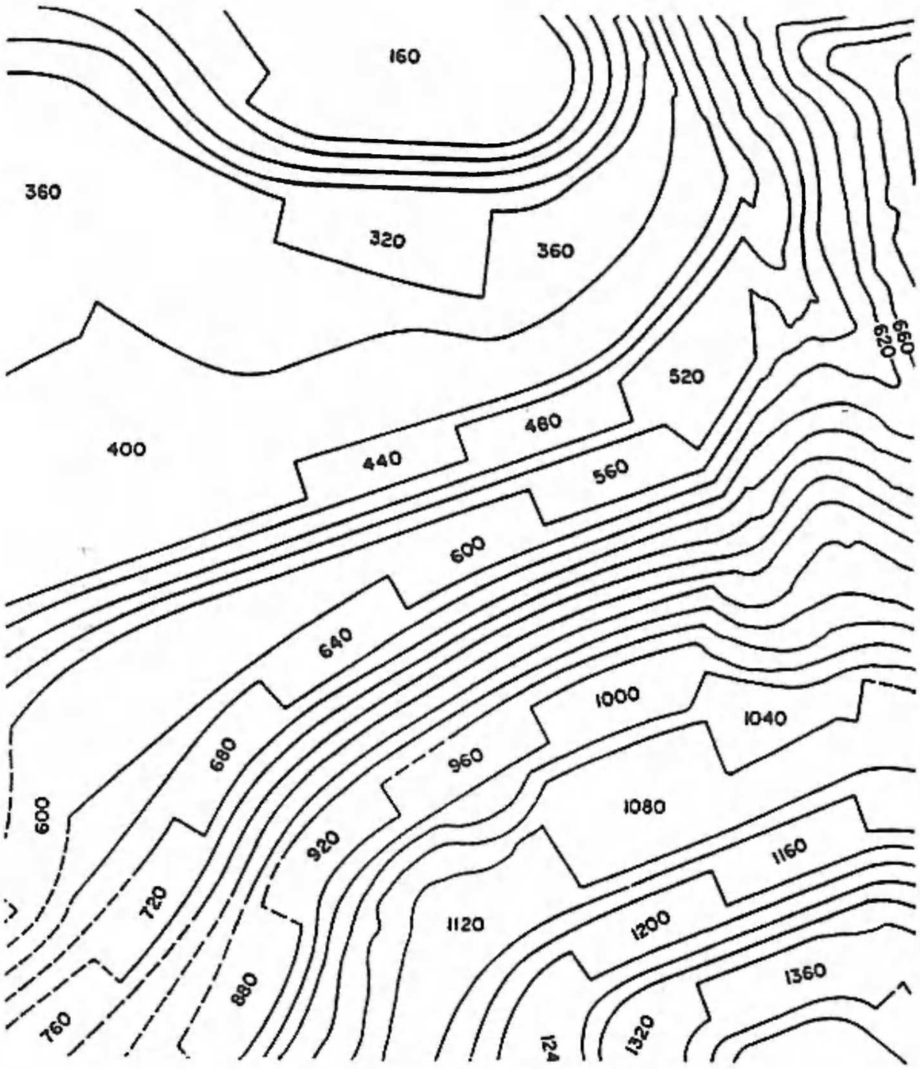


Figure 4.108. An example showing two different switchback (520 region and 1040 region) situations (Couzens, 1979).

The same pit with the road added, is shown in Figure 4.110. The shaded regions show the differences between sections A, B, C, D and E with and without the road.

In plan the length L of the road is

$$L = \frac{(\text{No. of benches} \times \text{Bench height})\ 100}{\text{Road grade (\%)}} = \frac{4 \times 30 \times 100}{10} = 1200\,\text{ft} \tag{4.4}$$

Because the road is oriented at angle Θ to the pit axis, the length projected along the axis is

$$L_2 = L \cos \Theta = 1176\,\text{ft} \tag{4.5}$$

The sections are made normal to this axis. They are spaced every 294 ft.

The road areas for each section are shown in Figure 4.111. The shaded boxes are of the same area

$$A = W_A \times \text{Bench height}$$

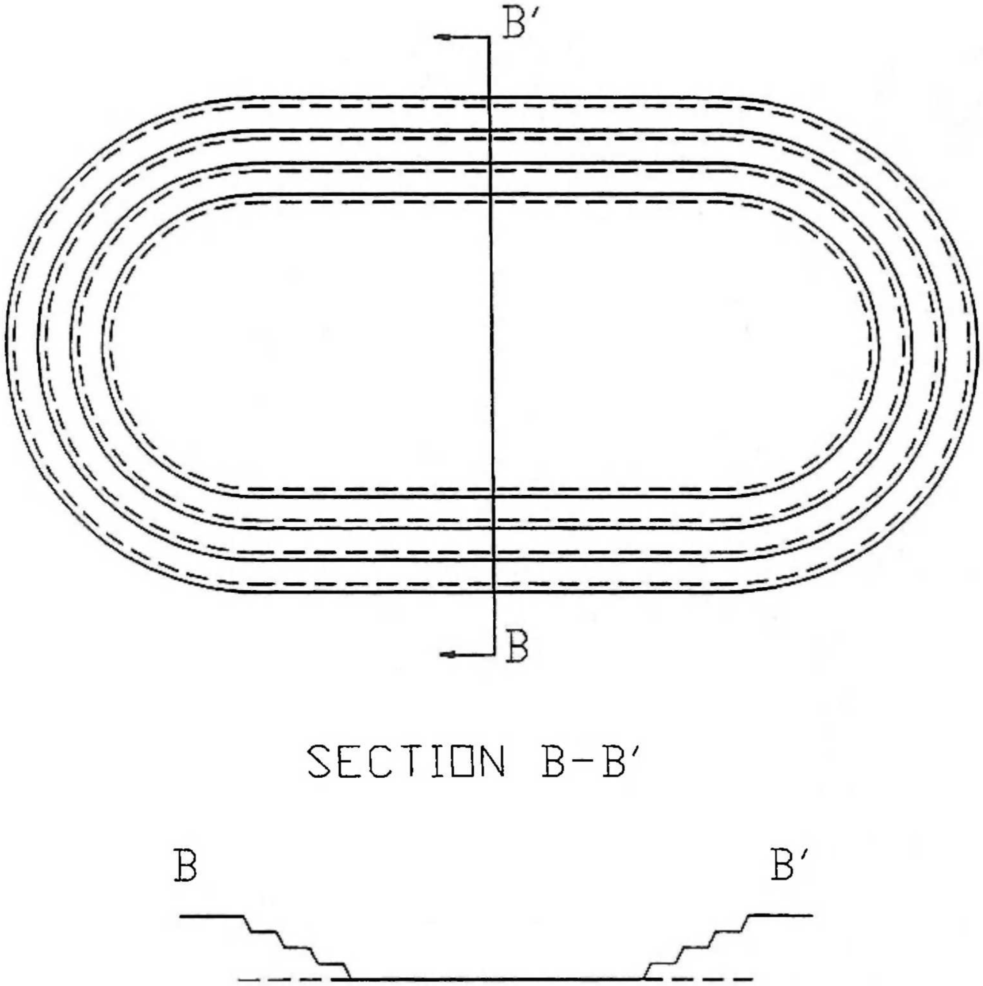


Figure 4.109. Plan and section views of a four bench pit without ramp.

They can be lined up as shown in Figure 4.111a. These in turn can be plotted such as shown in Figure 4.111b.

The volume contained in the ramp is that of a triangular solid of width W_A, length L_2 and height varying linearly from 0 to the pit depth (Fig. 4.112). This can be expressed as

$$V = \frac{1}{2} W_A L_2 \times \text{Pit depth} = \frac{1}{2} W_A L \cos\Theta \times \text{Pit depth} \tag{4.6}$$

which can be simplified to

$$V = \frac{1}{2} W_A \frac{(\text{Pit depth})^2}{\text{Grade }(\%)} 100 \cos\Theta \tag{4.7}$$

Since the apparent road width W_A is equal to

$$W_A = \frac{W_t}{\cos\Theta}$$

The simplified road volume formula becomes

$$V = \frac{1}{2} \frac{100 \times (\text{Pit depth})^2}{\text{Grade }(\%)} W_T \tag{4.8}$$

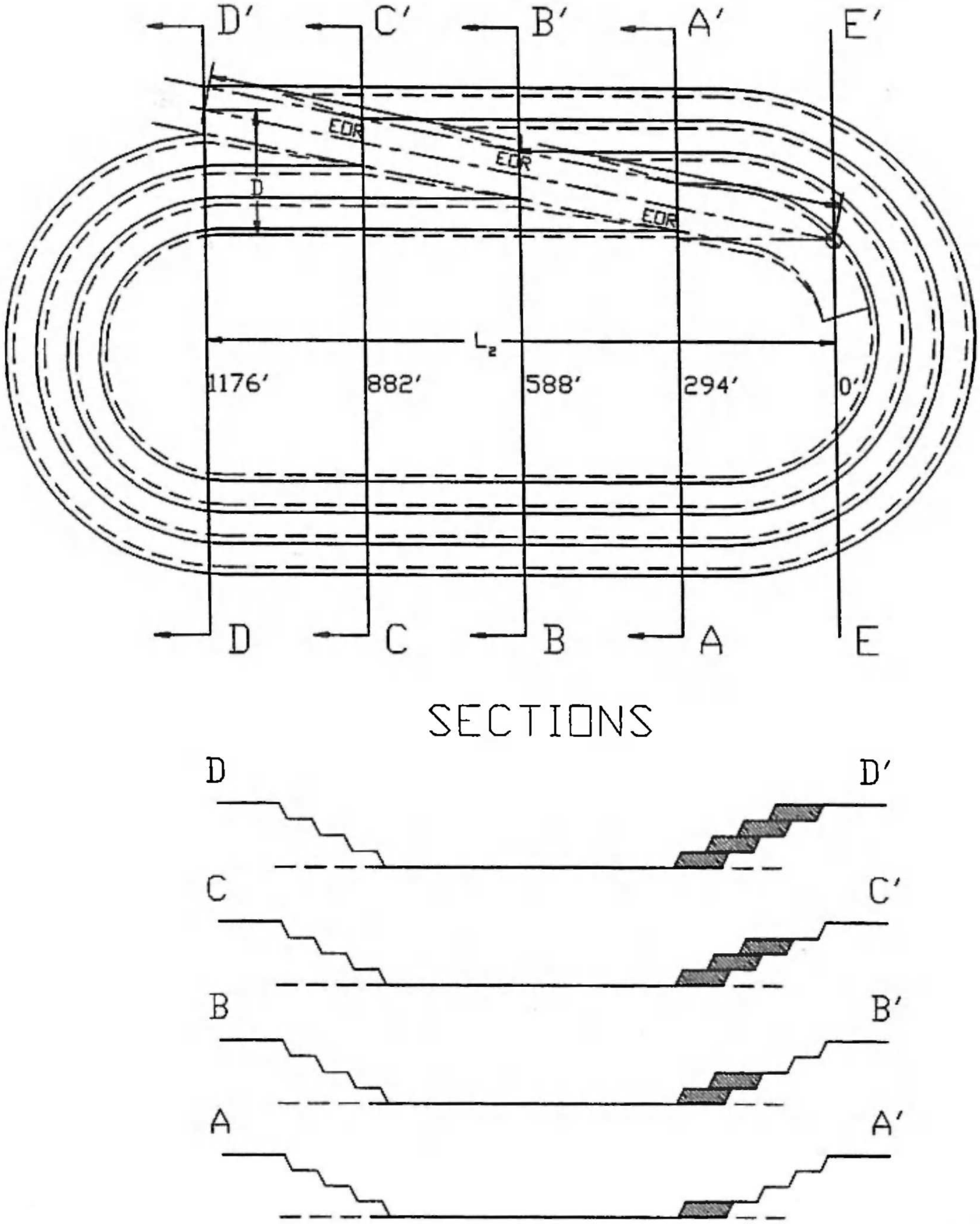


Figure 4.110. Plan and section views of a four bench pit with ramp.

In the present case the volume is

$$V = \frac{1}{2}\frac{100}{10}(120)^2 \times 90 = 6,480,00\,\text{ft}^3 = 240,000\,\text{yd}^3$$

For a tonnage factor of 12.5 ft^3/st, there are 518,400 st involved in the road.

The overall length of the road (L_{ov}) is given by

$$L_{ov} = \sqrt{L^2 + (\text{Pit depth})^2} \tag{4.9}$$

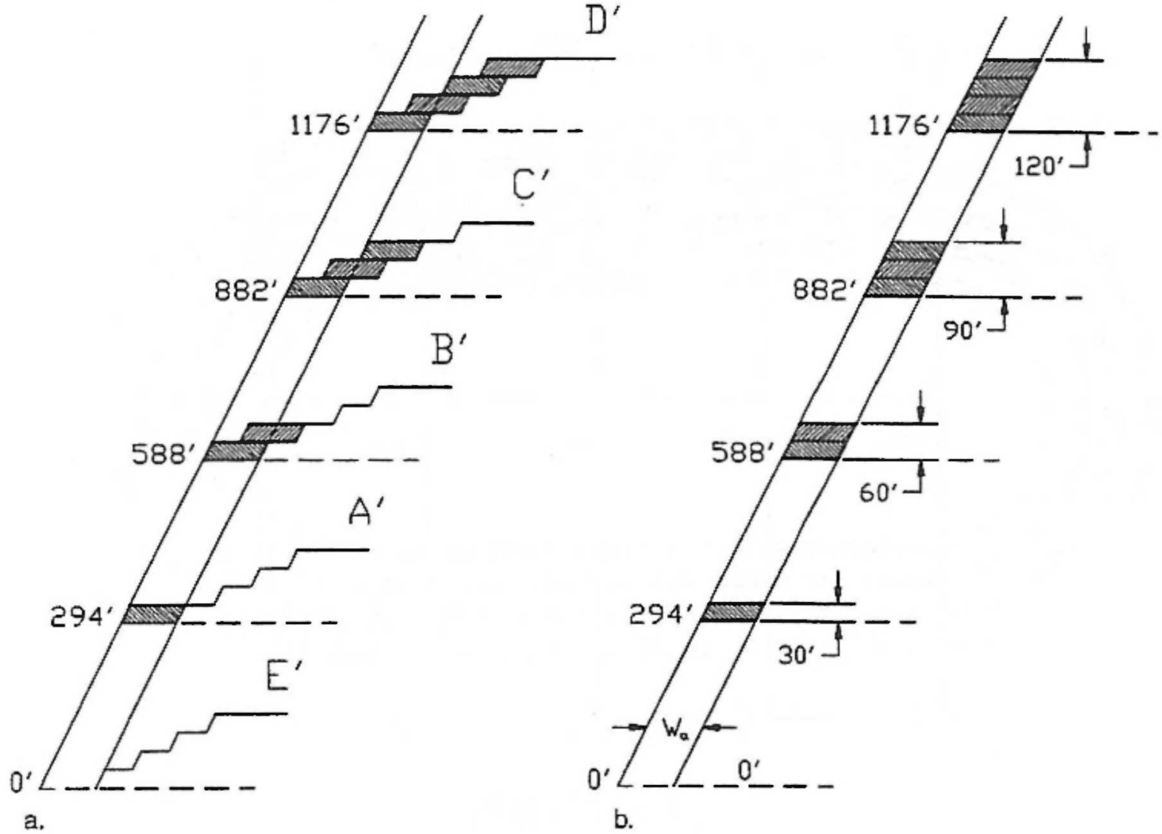


Figure 4.111. Construction to show road volume on each section.

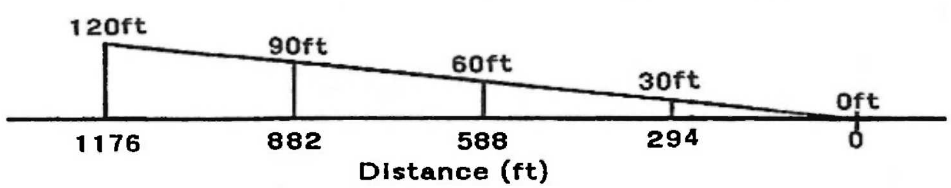


Figure 4.112. The volume involved in the ramp.

In this case it is

$$L_{ov} = \sqrt{(1200)^2 + (120)^2} = 1206\,\text{ft}$$

4.9 ROAD CONSTRUCTION

4.9.1 *Introduction*

Good haulroads are a key to successful surface mining operations. Poorly designed, constructed and maintained roads are major contributors to high haulage costs and pose safety hazards. In this section some of the basic design aspects will be discussed. Figure 4.113 shows a typical cross section through a road.

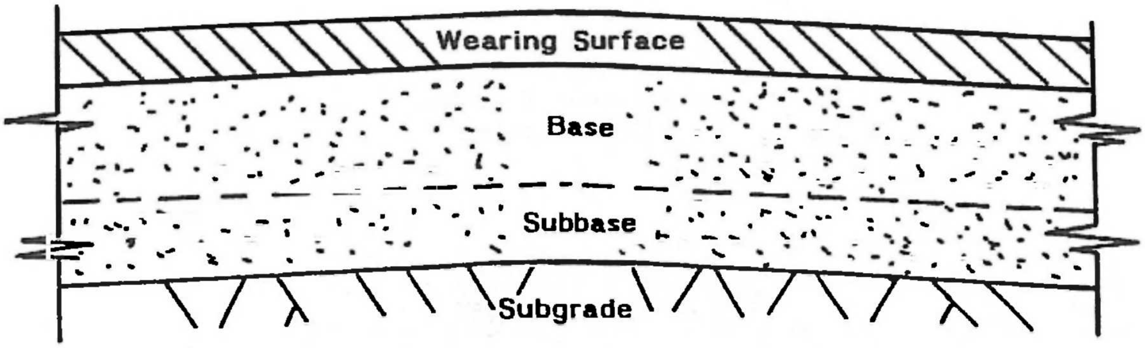


Figure 4.113. Simplified flexible pavement structure (Seelye, 1945).

Generally there are four different layers involved:
- subgrade,
- subbase,
- base,
- wearing surface.

The subgrade is the foundation layer. It is the structure which must eventually support all the loads which come onto the wearing surface. In some cases this layer will simply be the natural earth surface. In other and more usual instances, it will be the compacted rock or soil existing in a cut section or the upper layer of an embankment section.

The wearing surface provides traction, reduces tractive resistance, resists abrasion, ravelling and shear, transmits tire load to the base and seals the base against penetration of surface water. Although this surface may be asphalt or concrete, most typically it is crushed rock.

The base is a layer of very high stability and density. It's principal purpose is to distribute or 'spread' the stresses created by wheel loads acting on the wearing surface, so that they will not result in excessive deformation or the displacement of the subgrade. In addition it insulates the subgrade from frost penetration and protects the working surface from any volume change, expansion and softening of the subgrade.

The subbase which lies between the base and subgrade, may or may not be present. It is used over extremely weak subgrade soils or in areas subject to severe frost action. They may also be used in the interest of economy when suitable subbase materials are cheaper than base materials of a higher quality. Generally the subbase consists of a clean, granular material. The subbase provides drainage, resists frost heave, resists shrinkage and swelling of the subgrade, increases the structural support and distributes the load.

4.9.2 *Road section design*

In designing the road section, one begins with the maximum weight of the haulage equipment which will use the road. To be as specific as possible, assume that the haulage trucks have a maximum gross vehicle weight of 200,000 lbs including their 58 st payload. The load is distributed as follows:
- 33% on the front tires, and
- 67% on the dual rear tires.

The load on each of the front tires is 33,000 lbs. For each of the four rear tires (2 sets of duals) the load is 33,500 lbs. Thus the maximum loading to the wear surface is applied by

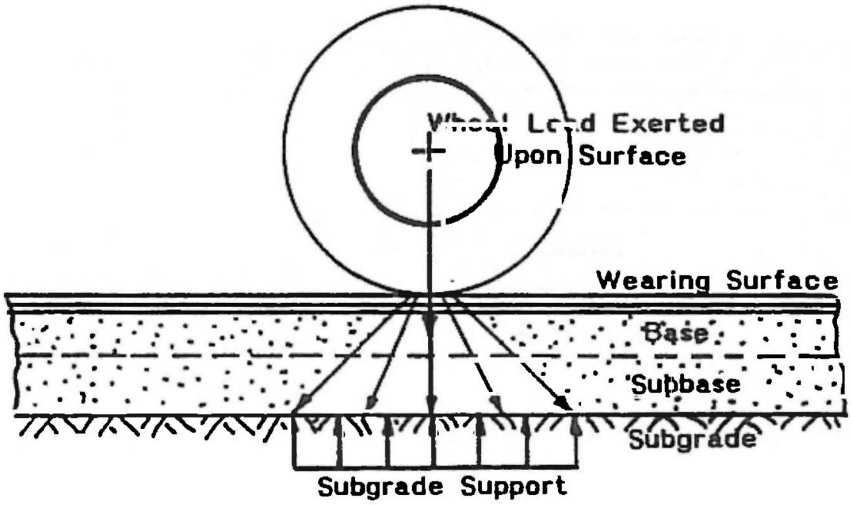


Figure 4.114. Load distribution beneath a tire (Seelye, 1945).

the rear tires. Although the contact pressure between the wheel and the road depends on the tire inflation pressure and the stiffness of the tire side walls, for practical purposes, the contact pressure is assumed to be equal to the tire pressure. Since for this truck, the inflation pressure is about 90 psi, the bearing pressure on the road surface is 90 psi or 12,960 psf. In lieu of knowing or assuming an inflation pressure, Kaufman & Ault (1977), suggest that a value of 16,000 psf (110 psi), will rarely be exceeded. The tire contact area is

$$\text{Contact area (in}^2) = \frac{\text{Tire load (lbs)}}{\text{Tire inflation pressure (psi)}} \tag{4.10}$$

For the rear tires

$$\text{Contact area (in}^2) = \frac{33{,}500}{90} = 372\,\text{inch}^2$$

Although the true contact area is approximately elliptical, often for simplicity the contact area is considered to be circular in shape. The contact pressure is usually assumed to be uniformly distributed. Because

$$\pi r^2 = 372\,\text{inch}^2$$

the radius of the tire contact area is

$$r \cong 11\,\text{inch}$$

and the average applied pressure is 90 psi (12,960 psf). As one moves down, away from the road surface, the force of the tire is spread over an ever increasing area and the bearing pressure is reduced. For simplicity, this load 'spreading' is assumed to occur at 45°. This is shown in Figure 4.114. Thus at a depth of 10 inches beneath the tire, the pressure radius would have increased to 21 inches and the pressure has dropped to 24.7 psi (3560 psf). However, for this truck there are dual rear wheels. Tire width is about 22 inches and the centerline spacing for the tires in each set is about 27 inches. This is shown diagrammatically in Figure 4.115.

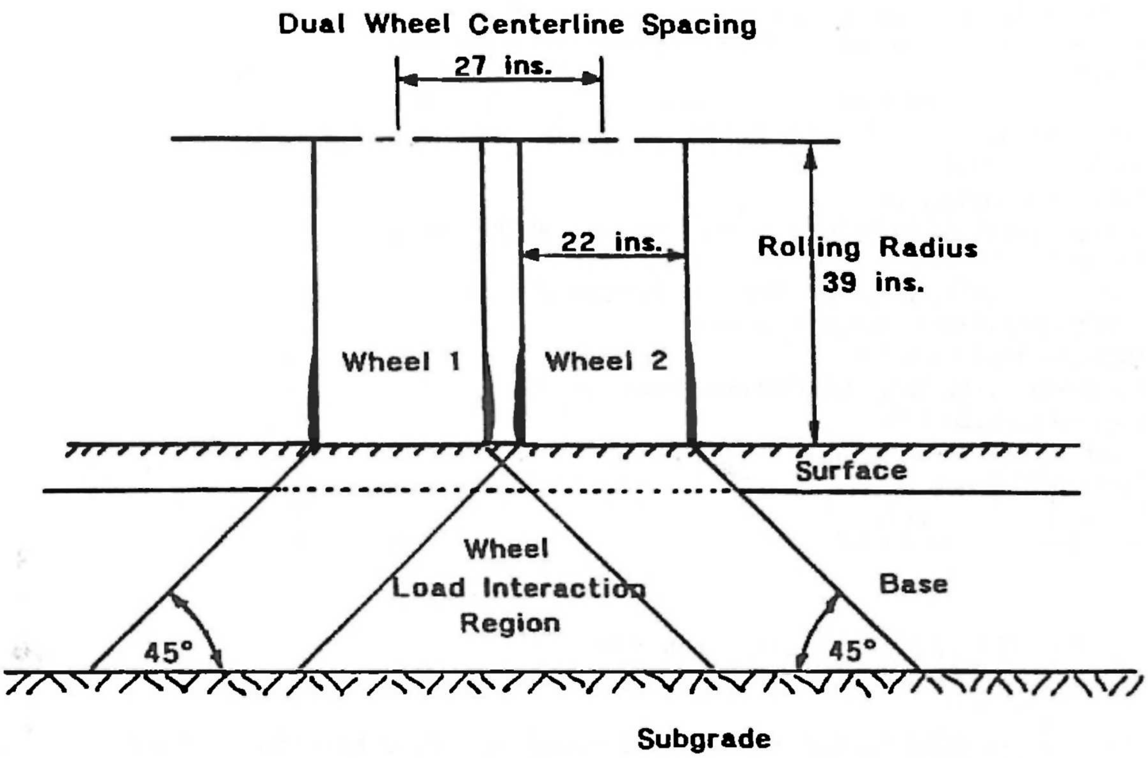


Figure 4.115. Load interaction with dual wheels.

As can be seen, the bearing pressure bulbs from each tire overlap. The greatest effect is observed along the line separating the tires. This interaction changes with tire width, tire separation and depth below the wear surface. To take this into account, Kaufman & Ault (1977) suggest using an equivalent single tire wheel load (L_E) which is 20% higher than the single tire load (L_T). Thus,

$$L_E = 1.20 \times L_T \tag{4.11}$$

In the case of the 58 st capacity truck

$$L_E = 1.20 \times 33,500 \cong 40,000\,\text{lb}$$

The combined subbase, base and wearing surface thickness must be sufficiently large so that the stresses occurring in the subgrade will not cause excessive distortion or displacement of the subgrade soil layer.

As a first guide, one can compare the required wear surface pressure to the bearing capacity of various subgrade materials. These are given in Table 4.6.

As can be seen, any subgrade that is less consolidated than soft rock will require additional material in order to establish a stable base. If, for example, the subgrade is a compact sand-clay soil with a bearing capacity of 6000 psf, then base/subbase materials of suitable strength would have to be placed down to increase the distance between the wear surface and the subgrade. Using the approach described earlier

$$\pi(11 + t)^2 \times 6000 = \pi(11)^2 \times 12,960$$

Table 4.6. Bearing capacities of subgrade materials (Kaufman & Ault, 1977).

Material	1000 psf
Hard, sound rock	120
Medium hard rock	80
Hard pan overlying rock	24
Compact gravel and boulder-gravel formations; very compact sandy gravel	20
Soft rock	16
Loose gravel and sandy gravel; compact sand and gravelly sand; very compact sand – inorganic silt soils	12
Hard dry consolidated clay	10
Loose coarse to medium sand; medium compact fine sand	8
Compact sand-clay soils	6
Loose fine sand; medium compact sand – inorganic silt soils	4
Firm or stiff clay	3
Loose saturated sand clay soils, medium soft clay	2

the minimal required thickness (t) would be

$$t \cong 5 \text{ inch}$$

The technique often applied to determine the working surface, base and subbase thicknesses involves the use of California bearing ratio (CBR) curves. The CBR test is an empirical technique for determining the relative bearing capacity of the aggregate materials involved in road construction. In this test the aggregate material with a maximum size of ¾ inch is placed in a 6 in diameter metal mold. The material is compacted by repeatedly dropping a 10 lb weight through a height of 18 in. After compaction, a cylindrical piston having an end area of 3 inch^2 is pushed into the surface at a rate of 0.05 inch/minutes. The CBR is calculated by dividing the piston pressure at 0.1 or 0.2 inch penetration by reference values of 1000 psi for 0.1 inch and 1500 psi for 0.2 inch. These standard values represent the pressures observed for a high quality, well graded crushed stone reference material. The calculated pressure ratios are multiplied by 100 to give the CBR value expressed as a percent. Figure 4.116 shows design curves based upon the use of CBR values. The subbase thickness has been plotted against CBR/soil type for various wheel loads.

To demonstrate the use of these curves, consider the 58-st capacity truck travelling over a haulroad which the subgrade material is a silty clay of medium plasticity (CBR = 5). One finds the intersection of CBR = 5 and the 40,000 lb equivalent single wheel load. Moving horizontally it is found that the required distance between the wear surface and the subgrade must be a minimum of 28 inches.

Fairly clean sand with a CBR of 15 is available to serve as subbase material. Repeating the process, one finds that this must be kept 14 inches away from the wear surface. The base material is well graded, crushed rock with a CBR rating of 80. The intersection of the 40,000 lb curve and CBR = 80 occurs at 6 inches. This 6 inch gap between the top of the base and the wear surface is intended to accommodate the wear surface thickness. If the actual wear surface is thinner than this, the remaining space is simply added to the base thickness (CBR equal to at least 80). Figure 4.117 shows the final results (Kaufman & Ault, 1977).

In most open pit mines, the wear surface is formed by well graded, crushed rock with a maximum dimension smaller than that used as base. Since traffic loading is directly applied

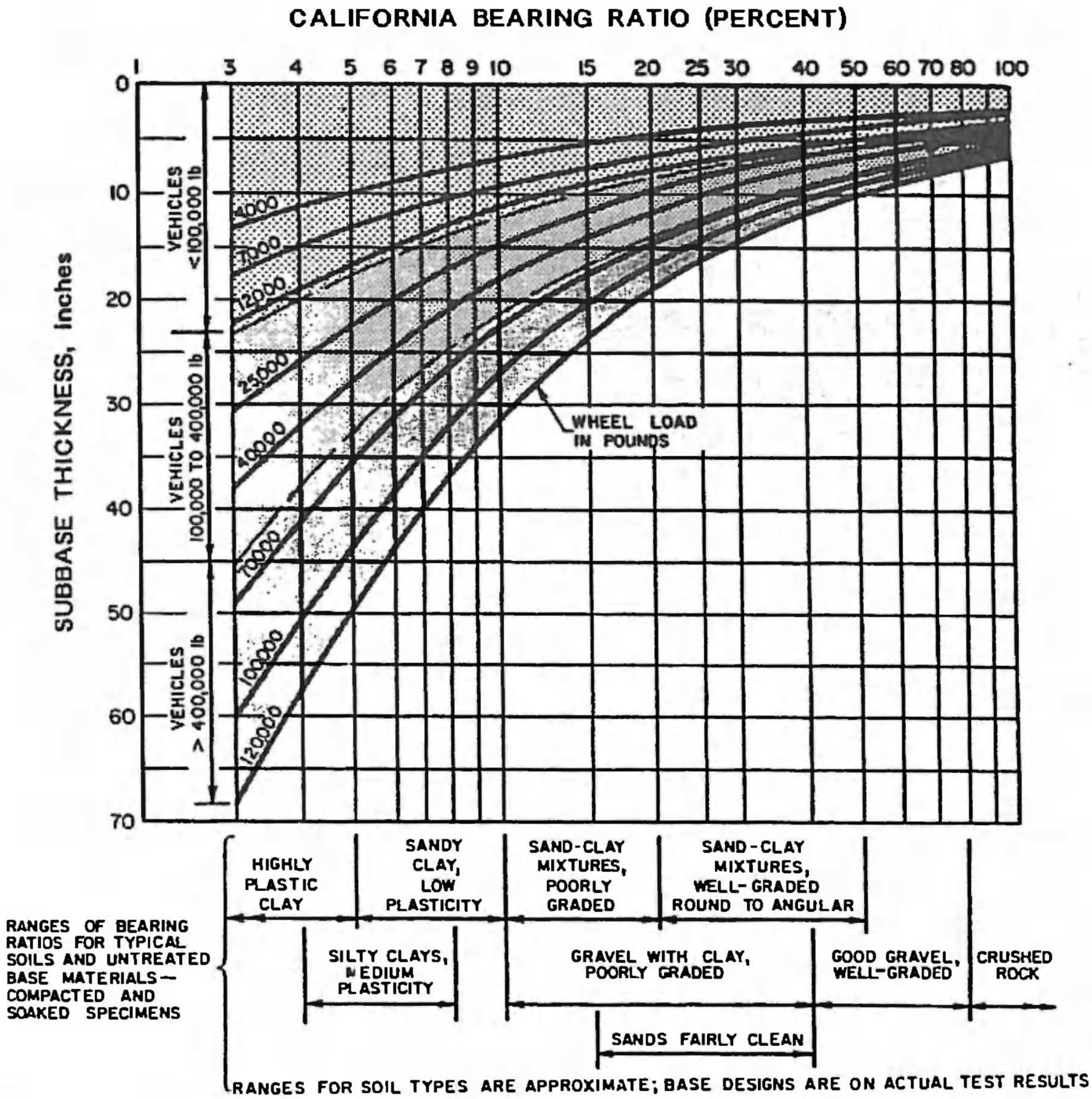


Figure 4.116. CBR curves (Kaufman & Ault, 1977).

to the aggregate layer, the upper most aggregate layer must possess sufficient strength and rutting resistance to minimize both

- bearing capacity failure, and
- rutting failure

within the layer.

The aggregate layer must also possess good wear resistance to minimize attrition under traffic. Table 4.7 indicates an acceptable aggregate size distribution (gradation) for this wearing surface.

Particle gradation is the distribution of the various particle size fractions in the aggregate. A well graded aggregate has a good representation of all particle size fractions from the maximum size through the smaller sizes. This is needed so that particles lock together forming a dense, compact surface. The opposite of a well graded aggregate is one which is poorly graded. Here the particles are all about the same size. Such a distribution might

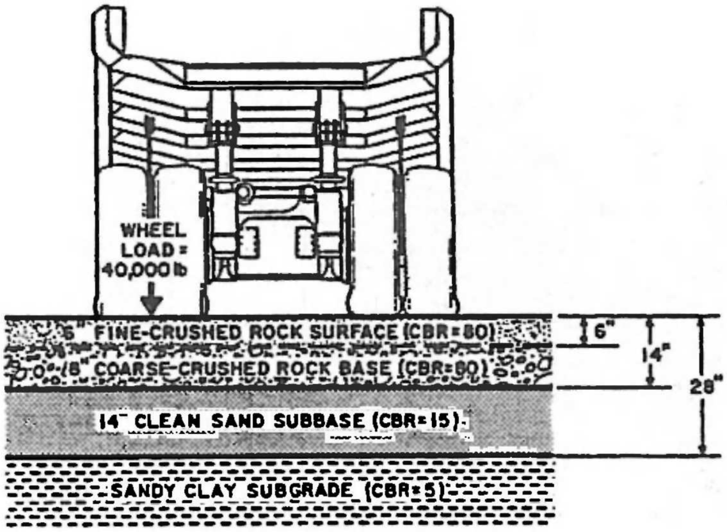


Figure 4.117. Example of mine road construction (Kaufman & Ault, 1977).

Table 4.7. Desired characteristics for a crushed stone running surface (Kaufman & Ault, 1977).

Screen size	Material passing (%)
$1\frac{1}{2}$ inches	100
1 inch	98
$\frac{3}{4}$ inch	92
$\frac{3}{8}$ inch	82
No. 4 mesh	65
No. 10 mesh	53
No. 40 mesh	33
No. 200 mesh	16
Liquid limit	25.2
Plasticity limit	15.8
Plasticity index	9.4
Optimum moisture content during placing	12.2

be used as part of a runaway ramp with the objective being that of creating a high rolling resistance.

The use of CBR curves requires laboratory tests or the assumption of CBR values of subgrade, and available base or subbase materials. The most economical combination is used. The CBR curves show directly the total thickness needed over any subgrade soil. The total subbase and the base thickness is created by putting down a series of relatively thin layers of the correct moisture content. Compaction is done between layers.

4.9.3 *Straight segment design*

Figure 4.118 shows a typical cross-section through a mine haul road carrying two way traffic. As can be seen there are three major components to be considered:

a) travel lane width,
b) a safety berm,
c) a drainage ditch.

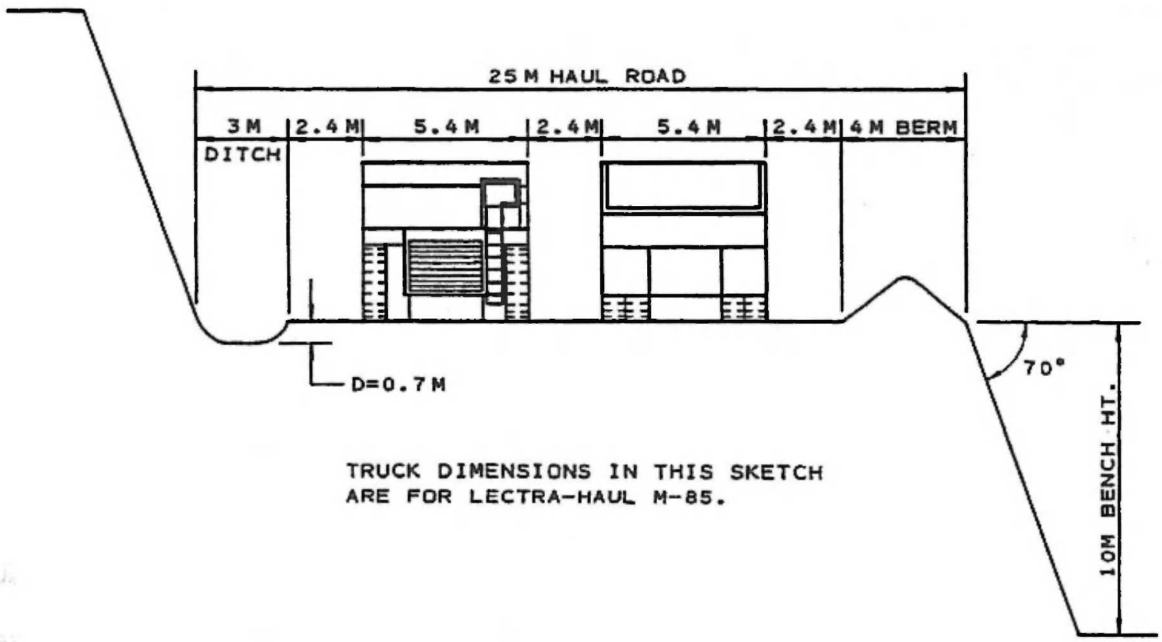


Figure 4.118. Typical design haulroad width for two-way traffic using 85 st capacity trucks (Couzens, 1979).

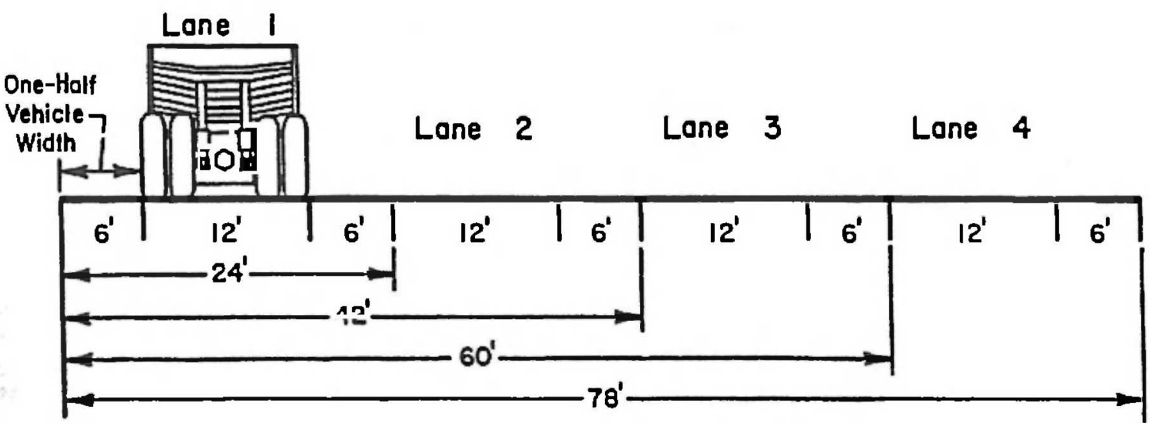


Figure 4.119. Multi-lane road design widths (Kaufman & Ault, 1977).

The width of each is added together to obtain the total roadway width.

The width criteria for the traveled lane of a straight haul segment should be based on the widest vehicle in use.

The 1965 *AASHO Manual for Rural Highway Design* recommends that each lane of travel should provide clearance to the left and right equal to one-half of the vehicle width. This is shown in Figure 4.126 for a 12-ft wide truck.

Values for other truck widths are given in Table 4.8. Typical widths of haulage trucks used in open pit mines are listed in Table 4.9.

For the two-way traffic which is most common in open pit mines, the rule of thumb is that roadway width should be no less than four times the truck width (Couzens, 1979):

$$\text{Roadway width} \geq 4 \times \text{Truck width} \tag{4.12}$$

Table 4.8. Recommended lane widths for tangent sections (Kaufman & Ault, 1977).

Vehicle width (ft)	1 lane	2 lanes	3 lanes
8	16	28.0	40
9	18	31.5	45
10	20	35.0	50
11	22	38.15	55
12	24	42.0	60
13	26	45.5	65
14	28	49.0	70
15	30	52.5	75
16	32	56.0	80
17	34	59.5	85
18	36	63.0	90
19	38	66.5	95
20	40	70.0	100
21	42	73.5	105
22	44	77.0	110
23	46	80.5	115
24	48	84.0	120
25	50	87.5	125
26	52	91.0	130
27	54	94.5	135
28	56	98.0	140

Table 4.9. Widths for various size rear dump trucks (this width includes the safety berm).

Truck size	Approx. width (m)	4 × width (m)	Design width	
			m	ft
35 st	3.7	14.8	15	50
85 st	5.4	21.6	23	75
120 st	5.9	23.6	25	85
170 st	6.4	25.6	30	100

Some mines have two lanes of traffic in one direction to allow passing for loaded uphill traffic. The downhill empty traffic travels in a single lane. A rule of thumb for the width of such a three lane road is 5 times the truck width.

The steps to be followed in selecting a design width are (Kaufman & Ault, 1977):

1. Define the width of all equipment that may have to travel the haulage road.
2. Solicit dimensional data for any anticipated new machines.
3. Determine the overall width of any equipment combinations that may be involved in a passing situation.
4. Delineate the location of road segments requiring a greater than normal width.

There may be wider stretches of road where there is merging of traffic streams such as near a crusher. Curves and switchbacks require special consideration. These will be discussed later.

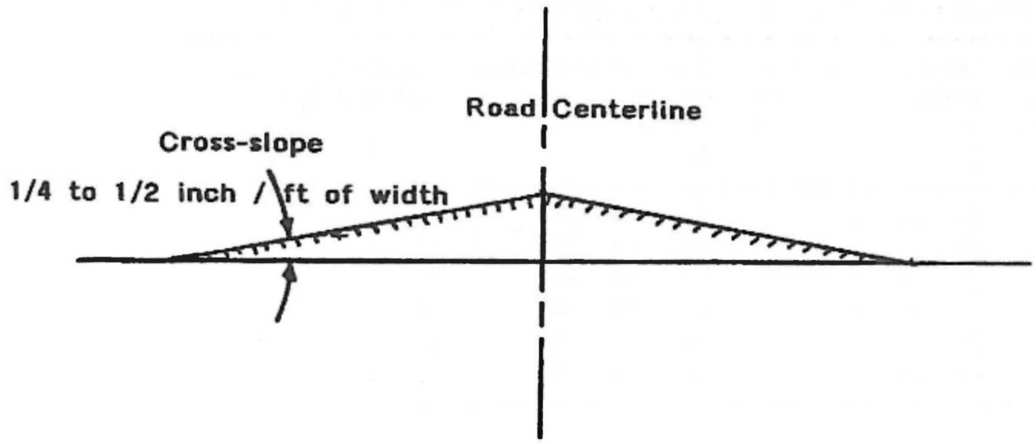


Figure 4.120. Cross slope design.

Table 4.10. Design widths (ft) for curves – single unit vehicles (Kaufman & Ault, 1977).

Radius (R) on inner edge of pavement (ft)	One-lane haulageway, vehicle category				Two-lane haulageway, vehicle category				Three-lane haulageway, vehicle category			
	1	2	3	4	1	2	3	4	1	2	3	4
Minimum	29	34	45	70	51	60	79	123	73	86	113	176
25	27	34	44	68	48	60	76	119	68	86	109	170
50	25	31	41	63	44	54	72	110	63	77	103	158
100	24	29	39	59	42	51	69	103	60	73	99	147
150	24	29	39	58	41	50	68	101	59	72	97	145
200	23	29	38	57	41	50	67	101	59	712	96	144
Tangent	23	28	37	56	40	48	65	98	57	69	93	140

The road surface is often slightly crowned such as shown in Figure 4.120, to facilitate water runoff. The cross slope is expressed in inches per foot of width. Most mine roads are constructed of gravel and crushed rock. In this case, except where ice/mud is a problem, the cross slope should be ½ inch per foot (0.04 ft/ft). For relatively smooth road surfaces such as asphaltic concrete which can rapidly shed water or roads which have ice/mud problems, a cross slope of ¼ inch per foot (0.02 ft/ft) is appropriate.

For single lanes, it is necessary to decide whether the left edge should be higher than the right or vice-versa. For three-lane surfaces, there should be a continuous cross slope for the two lanes having traffic in the same direction. It should be noted that the use of a cross-slope increases the steering effort by the driver. Thus there must be a balance between steerability and water drainage.

4.9.4 *Curve design*

For straight sections it was recommended that the left and right vehicle clearances should be half of the vehicle width. In the case of curves this distance must be increased both due to vehicle overhang and increased driving difficulty.

Tables 4.10 and 4.11 provide the design widths as a function of the inner pavement radius for various combinations of vehicle size, vehicle type and roadway types. For

Table 4.11. Design widths (ft) for curves – articulated vehicles (Kaufman & Ault, 1977).

Radius (R) on inner edge of pavement (ft)	One-lane haulageway, vehicle category			Two-lane haulageway, vehicle category			Three-lane haulageway, vehicle category		
	2	3	4	2	3	4	2	3	4
25	38	68	86	66	119	151	95	170	215
50	32	57	71	56	99	124	80	142	177
100	28	48	58	50	83	101	71	119	144
150	27	44	52	47	76	91	68	109	130
200	26	42	49	46	73	85	66	104	122
Tangent	25	41	41	44	71	72	63	102	103

Table 4.12. Minimum single unit haulage truck turning radius (Kaufman & Ault, 1977).

Vehicle weight classification	Gross vehicle weight (GVW) (lb)	Minimum turning radius (ft)
1	$< 100,000$	19
2	100–200,000	24
3	200–400,000	31
4	$> 400,000$	39

reference approximate turning radii are indicated by gross vehicle weight categories in Table 4.12.

For example, if a single unit haulage truck of weight classification 3 is to traverse a 100 ft minimum radius curve, the two lane width should be 69 ft. For a straight road segment the corresponding width is 65 ft. Hence the effect of the curve is to add 4 ft to the width.

Vehicles negotiating curves are forced outward by centrifugal force. For a flat surface this is counteracted by the product of the vehicle weight and the side friction between the roadway and the tires (Fig. 4.121).

For certain combinations of velocity and radius the centrifugal force will equal or exceed the resisting force. In such cases, the vehicle skids sideways. To assist the vehicles around the curves, the roadways are often banked. This banking of curves is called superelevation. The amount of superelevation (cross slope) can be selected to cancel out the centrifugal force. The basic equation is

$$e + f = \frac{V^2}{15R} \tag{4.13}$$

where e is the superelevation rate (ft/ft); f is the side friction factor; V is the vehicle speed (mph); R is the curve radius (ft). If $f = 0$, then the vehicle would round the curve without steering effort on the part of the operator. If however the operator would maintain a speed different from that used in the design, then he would have to steer upslope (in the case of too low a speed) of downslope (too high a speed) to maintain the desired path. Under ice and snow conditions, too slow a speed on such super elevated curves could lead to sliding down the slope.

Table 4.13 gives recommended superelevation rates as a function of curve radius and vehicle speed. The table can also be used to suggest a safe speed for a given radius and superelevation rate.

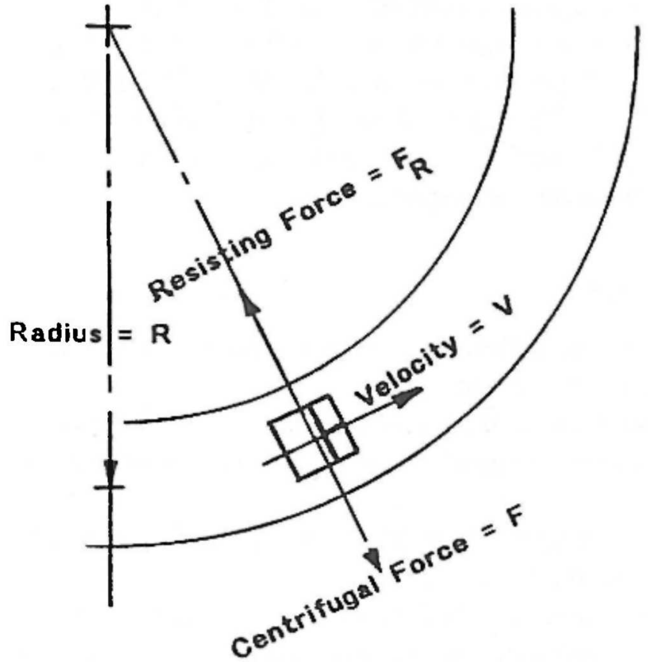


Figure 4.121. Centrifugal force effects on curves.

Table 4.13. Recommended superelevation rates (feet per foot of width) (Kaufman & Ault, 1977).

Radius of curve (ft)	Speed of vehicle (mph)					
	10	15	20	25	30	35 and over
50	0.04	0.04				
100	0.04	0.04	0.04			
150	0.04	0.04	0.04	0.05		
250	0.04	0.04	0.04	0.04	0.06	
300	0.04	0.04	0.04	0.04	0.05	0.06
600	0.04	0.04	0.04	0.04	0.04	0.05
1000	0.04	0.04	0.04	0.04	0.04	0.04

Table 4.14. Recommended rate of cross-slope change (Kaufman & Ault, 1977).

Vehicle speed (mph)	10	15	20	25	30	35 and above
Cross-slope change in 100-ft length of haulageway (ft/ft)	0.08	0.08	0.08	0.07	0.06	0.05

There is a certain distance required to make the transition from the normal cross-slope section to the superelevated portion and back again. This is called the superelevation runout. The purpose is to help ease the operator into and out of the curve. Part of the transition can be placed in the straight (tangent) portion and part in the curve. The design criteria of 1/3 inch curve and 2/3 inch the tangent is used here. The recommended rate of cross-slope change as a function speed is given in Table 4.14.

To illustrate the use of this table, assume a vehicle is traveling at 35 mph on tangent with normal cross slope 0.04 ft/ft to the right. It encounters a curve to the left necessitating a superelevation rate of 0.06 ft/ft to the left. The total cross-slope change required is 0.10 ft/ft (0.04 + 0.06). The table recommends a 0.05 ft/ft cross-slope change in 100 ft. Thus the total runout length is computed as 200 ft [(0.10/0.05) × 100 = 200]. One-third of this length should be placed in the curve and two-thirds on the tangent.

4.9.5 *Conventional parallel berm design*

U.S. federal law (MSHA, 1992) contains the following guidance regarding the need for berms/guardrails in open pit mines (Section 57.9300):

(a) Berms or guardrails shall be provided and maintained on the banks of roadways where a drop-off exists of sufficient grade or depth to cause a vehicle to overturn or endanger persons in equipment.

(b) Berms or guardrails shall be at least mid-axle height of the largest self-propelled mobile equipment which usually travels the roadway.

(c) Berms may have openings to the extent necessary for roadway drainage.

(d) Where elevated roadways are infrequently traveled and used only by service or maintenance vehicles, berms or guardrails are not required (when certain very specific conditions are met).

The principal purpose of these berms is to redirect the vehicle back onto the roadway and away from the edge. Their effectiveness in this regard is controlled by berm face angle, berm facing, the angle of incidence, and primarily by berm height. The stopping of runaway vehicles is accomplished by median berms (Subsection 4.9.6) or special escapeways. One negative effect of berms is the possibility of the vehicles overturning due to climbing the sides.

There are two principal berm designs in common use today. The triangular or trapezoidal shaped berm is generally formed from blasted materials. The sides stand at the angle of repose of the material. The second type is the boulder-faced berm. Here, large boulders, lined up along the haulage road, are backed with earthen material or blasted rock.

For the triangular berms, the design rule of thumb is that the height must be equal to or greater than the static rolling radius (SRR) of the vehicle's tire. For boulder-faced berms, the height of the berm should be approximately equal to the tire height. Figure 4.122 shows the relationship between the static rolling radius and haulage vehicle carrying capacity. Tire height (TH) is about equal to:

$$TH = 1.05 \times 2 \times SRR \tag{4.14}$$

4.9.6 *Median berm design*

Some means should be provided on haulroads to reduce truck speed or handle the truck that loses its brakes. This is particularly true when long, downhill loaded hauls are involved. Currently the most successful technique is through the use of median berms, also known as 'straddle berms' or 'whopper stoppers' (Winkle, 1976a,b). These are constructed of sand or some other fine grained material. The height of these berms is designed to impinge on the under-carriage of the truck. Since the typical distance between the road surface to the undercarriage is of the order of 2 to 3 ft for the range of available haulers, it is not necessary to build a big barrier providing just another crash hazard.

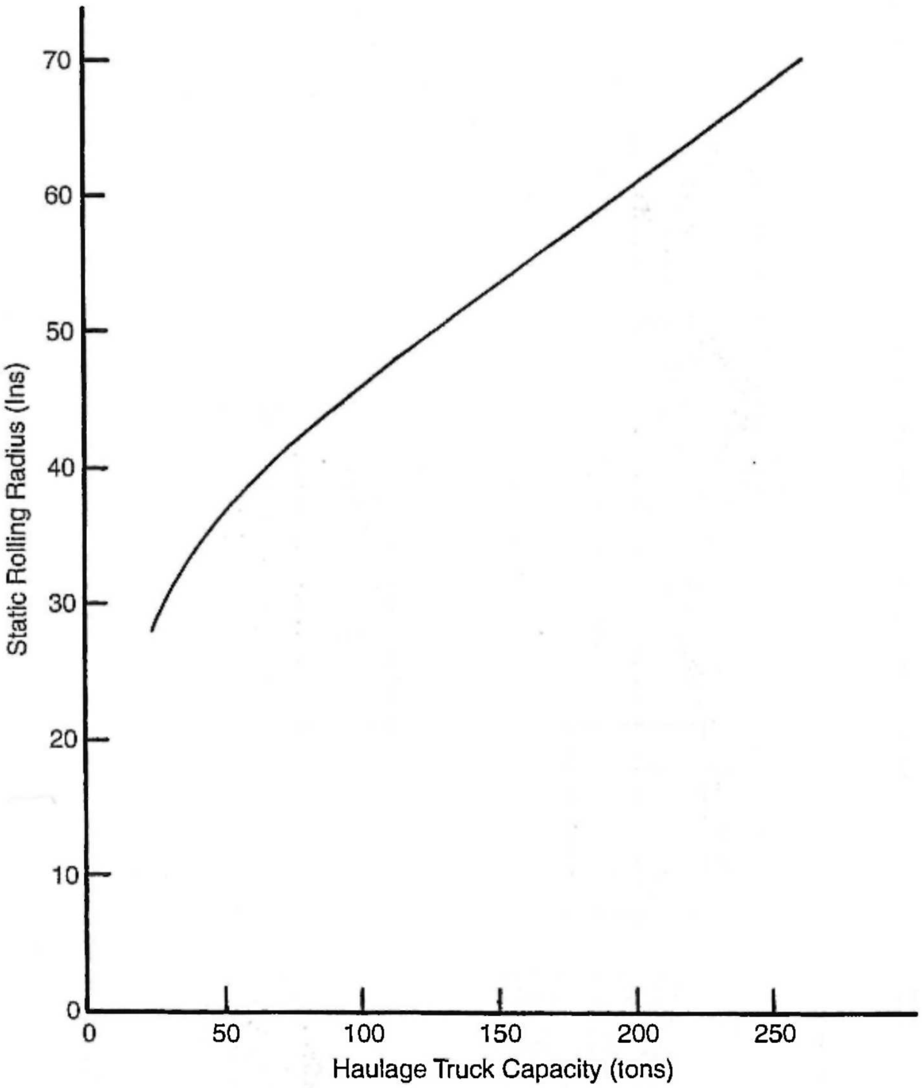


Figure 4.122. Static rolling radius as a function of haulage truck capacity (Goodyear, 1992).

Guidance in median berm design provided by Kaufman & Ault (1977) is given in Figure 4.123. The dimensions corresponding to the letters in the figure are given in Tables 4.15 and 4.16. The vehicle categories are based upon gross vehicle weights.

Training the driver to get onto the berm or into the bank just as soon as they start to lose control of their truck and before they build up speed is as important, or more important, than the berm design itself (Couzens, 1979).

4.9.7 *Haulage road gradients*

A number of rules of thumb regarding haulage road gradients have been provided by Couzens (1979). These are given below:

1. In a pit where there is a considerable vertical component to the haulage requirement, the grade will have to be fairly steep to reduce the length of the road and the extra material necessary to provide the road length. The practical maximum grade is considered to be 10%.

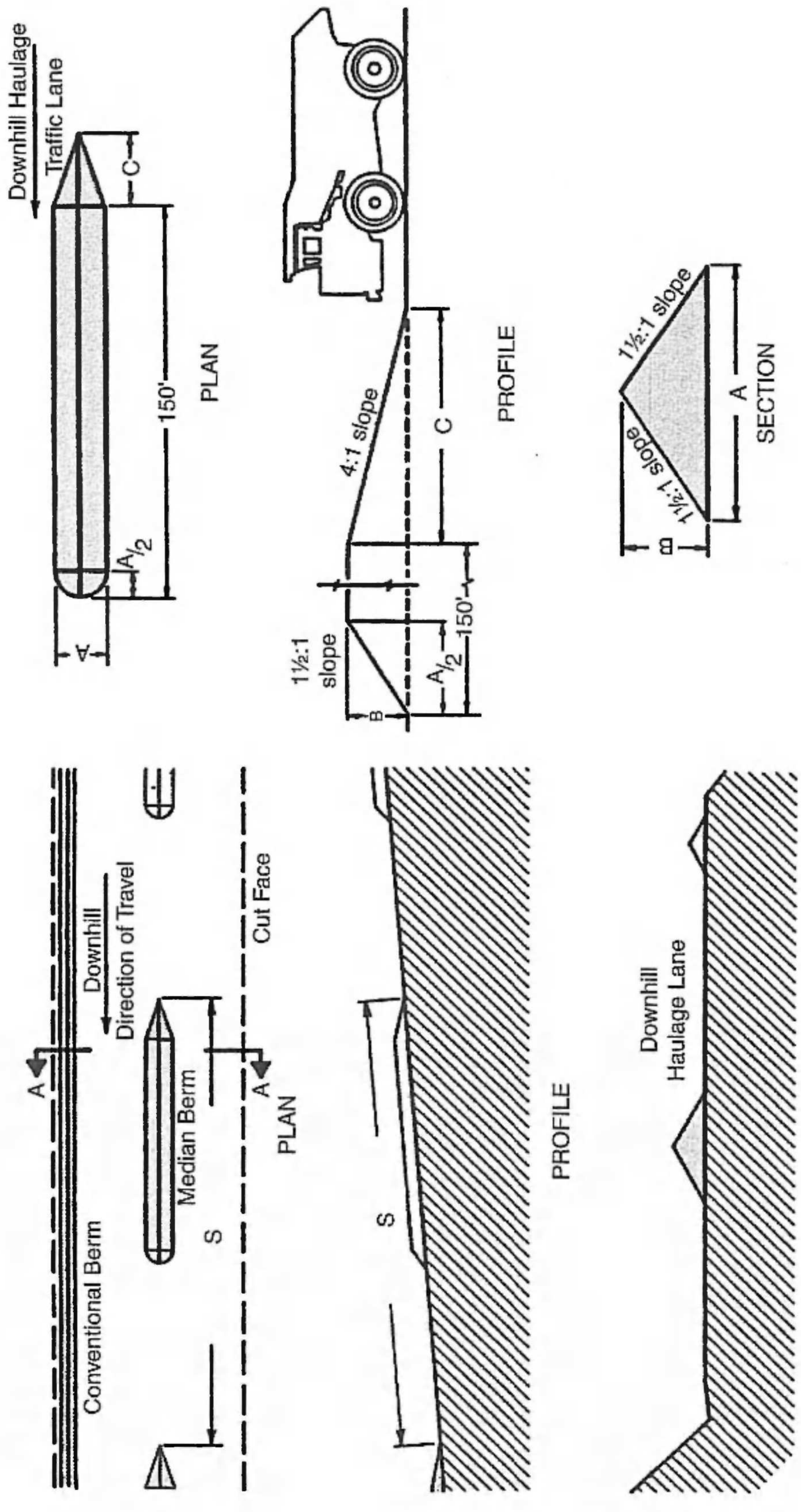


Figure 4.123. Runaway-vehicle collision berms (Kaufman & Ault, 1977)

Table 4.15. Typical median berm dimensions (see Figure 4.123) (Kaufman & Ault, 1977).

	A	B	C
Category 1 13 to 25 st <100,000 lb	11′–12′	3.5′–4′	14′–16′
Category 2 28 to 50 st 100,000–200,000 lb	12′–15′	4′–5′	16′–20′
Category 3 55 to 120 st >200,000–400,000 lb	15′–18′	5′–6′	20′–24′
Category 4 120 to 250 st > 400,000 lb	18′–32′	6′–11′	24′–44′

Table 4.16. Berm spacing (S) expressed in feet assuming that the initial speed at brake failure is 10 mph.

Equivalent downgrade, (%)	Maximum permissible vehicle speed or terminal speed at entrance to safety provision (mph)							
	15	20	25	30	35	40	45	50
1	418	1003	1755	2674	3760	5013	6433	8021
3	140	335	585	892	1254	1671	2145	2674
5	84	201	351	535	752	1003	1287	1604
7	60	144	251	382	537	716	919	1146
9	47	112	195	297	418	557	715	892
11	38	92	160	243	342	456	585	730
13	33	78	135	206	290	386	495	617
15	28	67	117	179	251	335	429	535

A number of pits operate quite well at 10% grades both favorable and unfavorable to the loads.

2. An 8% road grade is probably preferred providing that it does not cause too much extra stripping or unduly complicate the road layout. This grade provides more latitude in: (a) building the road and (b) fitting in bench entries without creating some locally over-steep places, than do steeper grades.

3. There is normally nothing to be gained by flattening the road below 8%, unless there is a long distance to travel without requiring much lift. The extra length on the grade and the complications of fitting the road into the available room or doing extra stripping would probably offset any increase in uphill haul speed.

4. Pit geometry is the prime consideration and roads are designed to fit the particular situation. Thus there often will be a number of different grade segments in haul roads.

4.9.8 *Practical road building and maintenance tips*

The preceding parts of this section have dealt with some of the general road design principles. Winkle (1976a,b) has provided a number of practical tips based upon many years of practical experience. Some of these have been included below.

1. The size of the orebody and the nature of the overlying topography will have considerable impact upon road design. When the orebody is small it will likely be advantageous to strip immediately to the projected pit limits, since some mining inefficiencies occur when mining areas overlap. This dictates an immediate final road layout on the backslope which is planned to avoid expensive modification. For very large orebodies, particularly where an outcrop of ore is exposed, it is highly unlikely that initial stripping will extend to the final planned perimeter. Careful study of the topography is required to ensure proper rapid access. The cost of rehandling material dumped within eventual pit limits must be weighed against increased haul distances, sharp curves, etc.

2. Change in equipment size frequently is a cause of road modification, particularly width. Pit design should incorporate allowance for reasonable future equipment size increases.

3. When mixed haulage fleets with varying speeds are used or where trucks hauling from two or more shovels are using the same haul roads, passing lanes on long grades should be considered.

4. Short radius curves result in reduced productivity, high tire cost, high maintenance cost (particularly electric wheels) and introduce additional safety hazards into the operation. Switchbacks are to be avoided unless a tradeoff of reduced stripping dictates their construction.

5. When curves are necessary in haulroads, superelevation must be designed into the curves. Excessive superelevation is to be avoided since trucks rounding a curve slippery from rain, ice or overwetting can slide inward and possibly overturn. Overly 'supered' curves result in excessive weight and wear on the inside tires.

6. Often curves are constructed to provide an access road into a mining bench from a steeply inclined haul road. To prevent the inside (and lower) side of this superelevated access curve from being at a steeper gradient than the main haul road, it is necessary to reduce (flatten) the center line grade of the curve. The inside grade should not be allowed to exceed the main road grade.

If enough room is available, the inside gradient of the curve should in fact be flatter than the main road grade to compensate for the increased rolling resistance. To accomplish this the design of a transition spiral is necessary.

7. Curves in the flat haul portion just as the trucks are leaving the shovel are quite critical. Due to the centrifugal forces induced by the curve, spill rock is thrown to the outside. Where possible, the return lane should be on the inside of the curve to avoid spill falling in the path of returning trucks and damaging tires. This can be accomplished by the use of crossovers to change traffic from right hand to left hand or vice versa in the necessary area. Adequate warning lights must be used at night to insured the safety at the crossovers. The costs of the warnings are small compared to savings in tire costs.

8. Waste dumps should be designed for placement at a two percent upgrade. This is done for the following reasons:

(a) The increase in dump height and volume occurs with little increase in haul speed or fuel cost. Because of the rapid dump volume increase, the haul distance is reduced.
(b) Better drainage on dumps.
(c) Some additional safety is afforded drivers backing up for dumping.
(d) If eventual dump leaching is planned, the water distribution is less expensive.

9. Within the mining areas, roads are built of the country rock at hand and are surfaced with the best material available within a reasonable haul distance. In the case of using something other than environmental rock to surface roads within the ore zone, double handling costs as well as ore dilution must be considered.

10. Main roads into the pit are usually planned for extended time of use and will justify more expenditure for subbase compaction and surfacing than temporary access roads.

11. If intended for use as a haul road, engineering layout should precede construction of even the shortest road or ramp. Mine survey crews should place desired stakes for initial cuts and fills and grade stakes including finish grade stakes.

12. When shovels are working in coarse, sharp rock faces, loading should be stopped periodically to allow fine material to be brought in and used to cap the surface of the loading area. Similar activity should be performed on waste dumps.

13. Constant attention to haul road surfaces is necessary. Soft spots, holes, 'washboard' areas, etc. should be repaired as soon as possible. Repairs usually consist of digging out the incompetent road material and replacing it with more desirable rock.

14. Grading of roads often results in a buildup of windrows on road edges. These narrow the roads and place sharp rocks in a position to damage tire sidewalls. Windrow buildup should be removed by loader or careful grader application.

15. Balding or grading of roads and dumps should be done when possible at a time when traffic can be moved to other areas. Many tires have been damaged by trucks driving through windrows created by graders assigned to improve roads and thereby reduce tire costs.

16. Maintenance of haul roads is equally important to good haulage costs as are design and construction. As more tires are damaged in shovel pits and dump areas than on actual haul roads, cleanup around an operating shovel is often assigned to the haulroad rather than the loading function. Road maintenance, to be successful, must have responsible supervision assigned to this task alone.

4.10 STRIPPING RATIOS

Consider the orebody shown in Figure 4.124 which has the shape of a right circular cylinder.

It outcrops at the surface and extends to a depth h. The volume of the contained ore is expressed by

$$V_o = \pi r^2 h \tag{4.15}$$

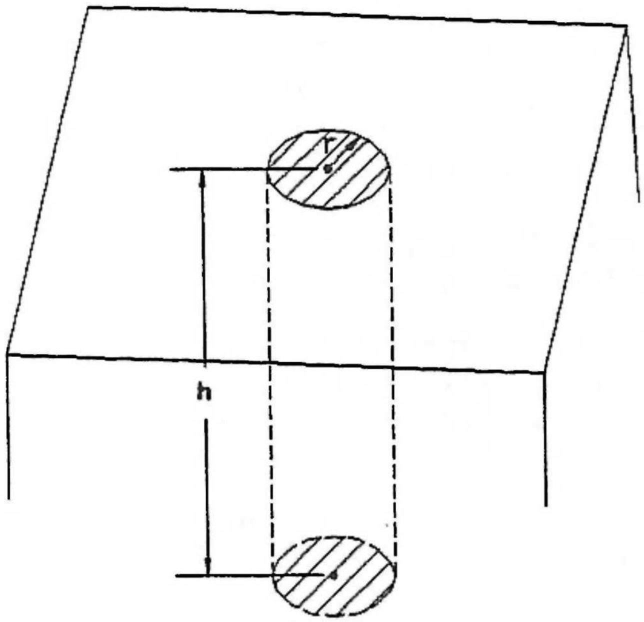


Figure 4.124. Cylindrical orebody.

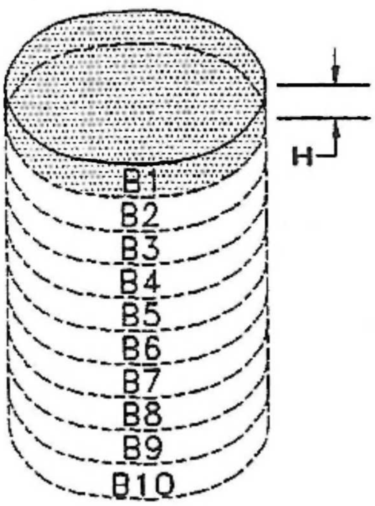


Figure 4.125. Cylindrical orebody mined as a sequence of constant diameter and thickness benches.

where r is the ore radius and h is the ore thickness. In concept, at least, one could remove the ore as a single plug and just leave the remaining hole. In practice, however, the orebody is first divided up into a series of benches of thickness H (Fig. 4.125). The volume of each ore bench B_i is

$$V_b = \pi r^2 H \tag{4.16}$$

In this case it will be assumed that each bench exactly satisfies the required annual production. Hence the pit would increase in depth by one bench per year. The surrounding waste rock has been assumed to have high strength so that these 90° pit walls can be safely achieved and maintained. In this mining scheme no waste is removed.

In reality, vertical rock slopes are seldom achieved except over very limited vertical heights. It is much more common to design using an overall slope angle Θ. As can be seen in Figure 4.126 the shape of the mined space changes from a right circular cylinder to a truncated right circular cone. The height of the truncated portion of the cone is

$$\Delta h = r \tan \Theta \tag{4.17}$$

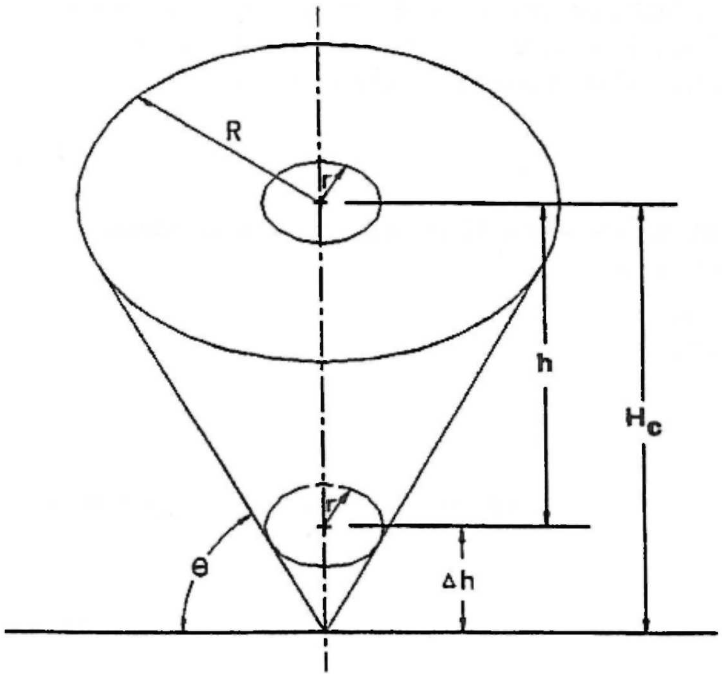


Figure 4.126. The cylindrical orebody mined via a conical pit.

where Θ is the overall slope angle. The height H_c of the cone which includes the orebody is then

$$H_c = h + \Delta h = h + r\tan\Theta \tag{4.18}$$

The base radius R of the circumscribed cone is

$$R = \frac{H_c}{\tan\Theta} = \frac{h}{\tan\Theta} + r \tag{4.19}$$

Using the volume formula for a right circular cone

$$V_{rcc} = \frac{1}{3}A_{bc}H_c \tag{4.20}$$

where A_{bc} is the base area of the cone, H_c is the height of the cone, and V_{rcc} volume of the cone, one can find the following volumes:

Truncated tip

$$V_{tip} = \frac{1}{3}\pi r^2 \Delta h \tag{4.21}$$

Fully circumscribed cone

$$V = \frac{1}{3}\pi R^2 H_c \tag{4.22}$$

Mined volume (ore + waste)

$$V_m = V - V_{tip} = \frac{1}{3}\pi R^2 H_c - \frac{1}{3}\pi r^2 \Delta h \tag{4.23}$$

Volume of waste

$$V_w = V_m - \pi r^2 h \tag{4.24}$$

One of the ways of describing the geometrical efficiency of a mining operation is through the use of the term 'stripping ratio'. It refers to the amount of waste that must be removed to release a given ore quantity. The ratio is most commonly expressed as

$$\text{SR} = \frac{\text{Waste (tons)}}{\text{Ore (tons)}} \tag{4.25}$$

however a wide variety of other units are used as well. In strip coal mining operations for example the following are sometimes seen

$$\text{SR} = \frac{\text{Overburden thickness (ft)}}{\text{Coal thickness (ft)}}$$

$$\text{SR} = \frac{\text{Overburden (yd}^3)}{\text{Coal (tons)}}$$

The ratio of waste to ore is expressed in units useful for the design purpose at hand. For this example, the ratio will be defined as

$$\text{SR} = \frac{\text{Waste (volume)}}{\text{Ore (volume)}} \tag{4.26}$$

Note that if the waste and ore have the same density, then Equation (4.25) and Equation (4.26) are identical.

If the volumes (or tons) used in the SR calculation correspond to those (cumulatively) removed from the start of mining up to the moment of the present calculation then the overall stripping ratio is being calculated. For this example the overall stripping ratio at the time mining ceases is

$$\text{SR (overall)} = \frac{V_w}{V_o} = \frac{V_m - \pi r^2 h}{\pi r^2 h} \tag{4.27}$$

On the other hand a stripping ratio can also be calculated over a much shorter time span. Assume that during year 5, X_o tons of ore and X_w tons of waste were mined. The stripping ratio for year 5 is then

$$\text{SR (year 5)} = \frac{X_w}{X_o}$$

This can be referred to as the instantaneous stripping ratio where the 'instant' in this case is 1 year.

If at the end of year 4, X_{o4} tons of ore and X_{w4} tons of waste had been mined then the overall stripping ratio up to the end of year 5 is

$$\text{SR (overall to end of year 5)} = \frac{X_{w4} + X_w}{X_{o4} + X_o}$$

Obviously the 'instant' could be defined as a longer or shorter time period. If during a given day the mine moves 5000 tons of waste and 2000 tons of ore, the instantaneous stripping ratio (for that day) is

$$\text{SR (instantaneous)} = \frac{5000}{2000} = 2.5$$

The determination of final pit limits as will be described in detail in Chapter 5, involves the calculation of a pit limit stripping ratio to be applied to a narrow strip at the pit periphery.

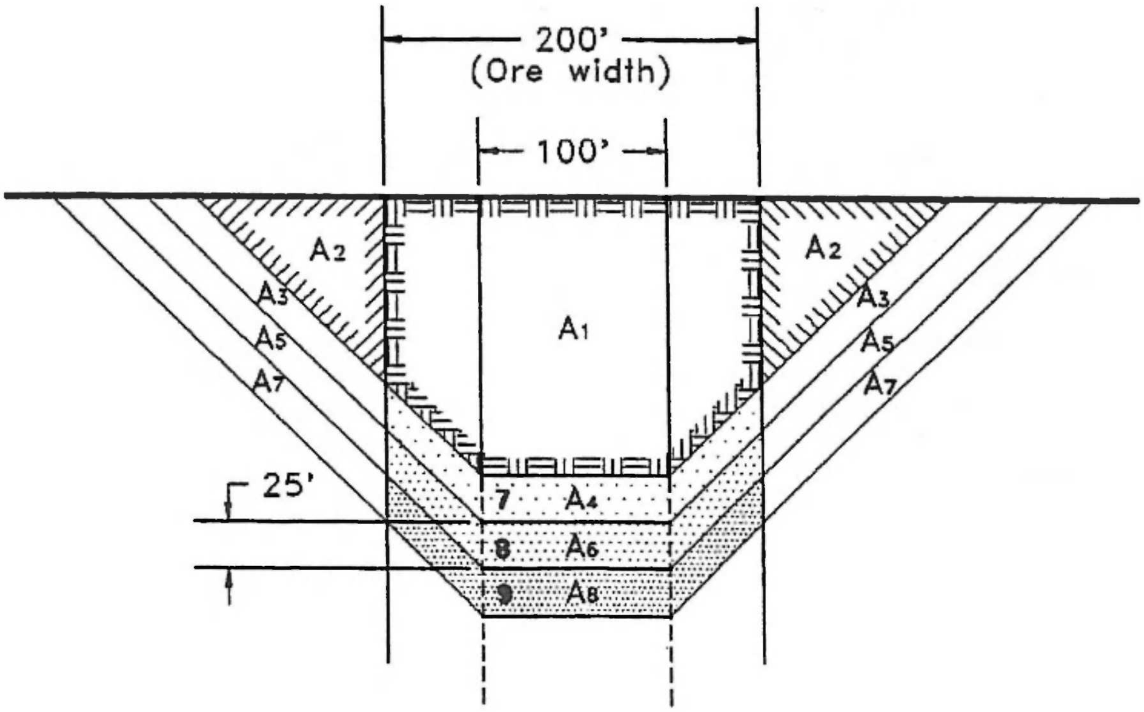


Figure 4.127. Section for stripping ratio calculations.

To illustrate this concept consider the simple cross-section shown in Figure 4.127. It will be assumed that:

- the pit is deepened in bench height increments of 25 ft;
- the minimum pit width is 100 ft;
- overall slope angle is 45°
- the density of the ore and waste is the same;
- the ore is of constant grade.

The original pit on this section (Fig. 4.127), consists of 6 benches and has a depth of 150 ft. The area of ore A_o is

$$A_o = A_1 = 200 \times 100 + 50 \times 150 = 27,500\,\text{ft}^2$$

The area of waste A_w is

$$A_w = 2A_2 = 100 \times 100 = 10,000\,\text{ft}^2$$

The overall stripping ratio SR (overall) is

$$\text{SR (overall)} = \frac{A_w}{A_o} = \frac{10,000}{27,500} = 0.36$$

Deepening of the pit by one bench (bench 7) requires the removal of $2A_3$ of waste. The amount of ore uncovered is A_4

$$A_4 = 100 \times 25 + 100 \times 25 = 5000\,\text{ft}^2$$

$$2A_3 = 125 \times 125 - 100 \times 100 = 5625\,\text{ft}^2$$

The instantaneous stripping ratio is

$$\text{SR (instantaneous)} = \frac{5625}{5000} = 1.125$$

The overall stripping ratio with bench 7 removed is

$$\text{SR (overall)} = \frac{15,625}{32,500} = 0.48$$

With mining of bench 8, another 5000 ft^2 of ore (A_5) is removed. This requires the stripping of

$$2A_5 = (150)^2 - (125)^2 = 6875\,\text{ft}^2$$

of waste. The instantaneous stripping ratio is

$$\text{SR (instantaneous)} = \frac{6875}{5000} = 1.375$$

The overall stripping ratio is

$$\text{SR (overall)} = \frac{22,500}{37,500} = 0.60$$

For bench 9:

$$A_8 = 5000\,\text{ft}^2$$

$$2A_7 = (175)^2 - (150)^2 = 8125\,\text{ft}^2$$

$$\text{SR (instantaneous)} = \frac{8125}{5000} = 1.625$$

$$\text{SR (overall)} = \frac{30,625}{42,500} = 0.72$$

As can be seen in this simple example, with each cut, the same amount of ore 5000 ft^2 must pay for an increasing amount of waste. The overall stripping ratio is less than the instantaneous value. There becomes a point where the value of the ore uncovered is just equal to the associated costs with the slice. This would yield the maximum pit on this section. Assume that in this case the breakeven stripping ratio is 1.625. Then the final pit would stop with the mining of bench 9. Through pit deepening, the walls of the pit are moved away or 'pushed back' from their original positions. The term 'push-back' is used to describe the process by which the pit is deepened by one bench.

4.11 GEOMETRIC SEQUENCING

There are several ways in which the volume of Figure 4.126 can be mined. As before, the first step in the process is to divide the volume into a series of benches (see Fig. 4.128). If a single bench is mined per year then the ore production would remain constant while both the total production and the stripping ratio would decrease. This would lead to a particular cash flow and net present value.

For most mining projects, a large amount of waste mining in the early years of a project is not of interest.

An alternative mining geometry is shown in Figure 4.129 in which a number of levels are mined at the same time. The overall geometry looks much like that of an onion.

An initial 'starter-pit' is first mined. In this example, the pit bottom extends to the edge of the orebody and the slope angle of the starter pit is the same as that of the final pit (Θ).

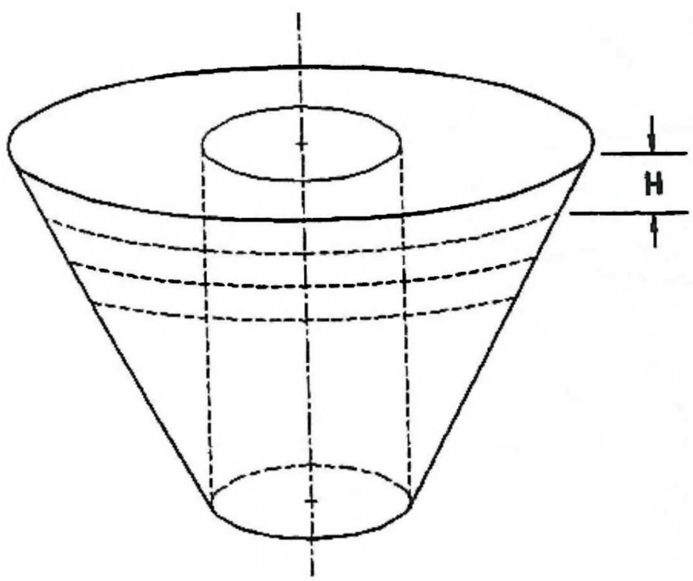


Figure 4.128. Sequential geometry 1 (Fourie, 1992).

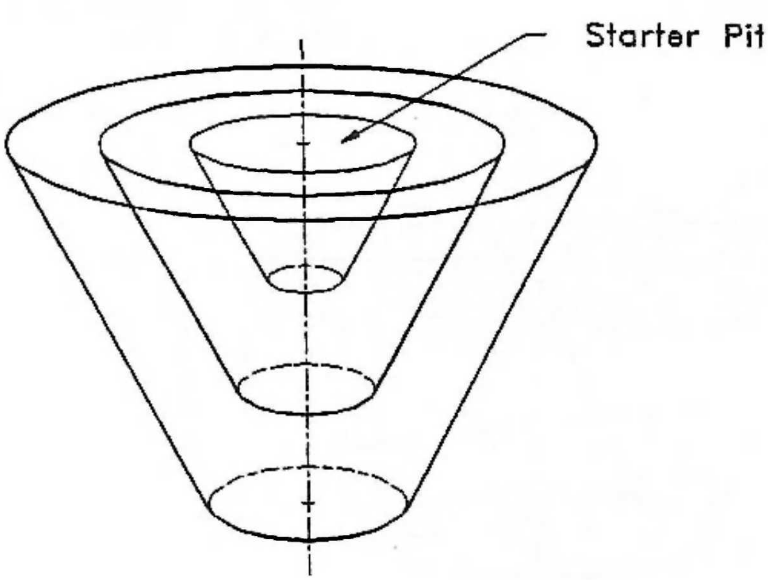


Figure 4.129. Sequential geometry 2 (Fourie, 1992).

In theory one could then slowly 'eat-away' at the sides and bottom of this starter pit until the final pit geometry is achieved. There are practical limits however on the minimum size 'bites' which can be considered both for planning and execution. The 'bites' in surface mining terms are called *push-backs* or phases. For modern large pits the minimum push-back distance (thickness of the bite) is of the order of 200 to 300 ft. For smaller pits it can be of the order of 100 to 200 ft. In this particular example the push-backs result in the pit being extracted in a series of concentric shells. The amount of material (ore and waste) contained in each shell is different. Hence for a constant production rate there might be x years of ore production in shell 1, y years of production in shell 2, etc. Eventually there will be a transition in which mining is conducted in more than one shell at a given time.

Sequencing within a pit shell and between shells becomes important. To this point simple concentric shells have been considered. The next level of complication is to split the overall pit into a number of sectors such as shown in Figure 4.130. Each sector (I → V) can be

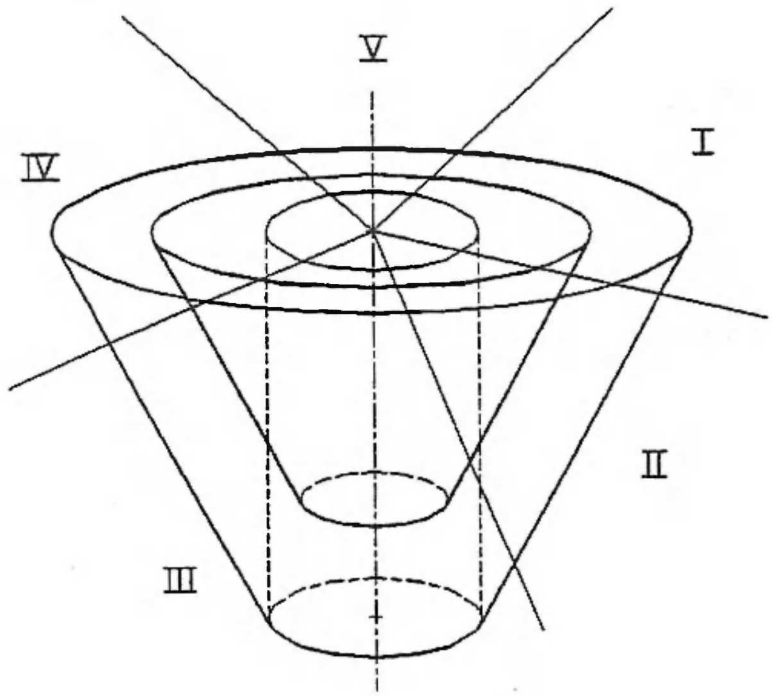


Figure 4.130. Sequential geometry 3 (Fourie, 1992).

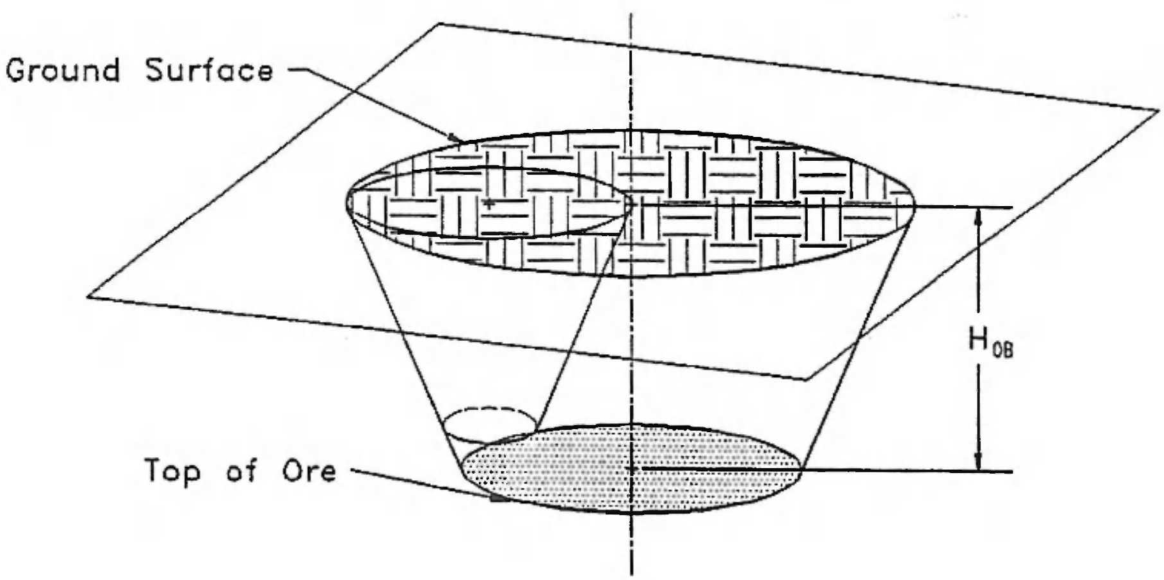


Figure 4.131. Sequential geometry 4 (Fourie, 1992).

considered as a separate production or planning unit. A natural basis for dividing the pit this way is due to slope stability/design considerations.

It has been assumed that the orebody outcrops (is exposed) at the surface. If this is not the case, such as is shown in Figure 4.131, then a preproduction or stripping phase must be first considered.

Due to cash flow considerations a variety of aspects enter:
- desire to reach the ore as quickly as possible,
- requirement to expose enough ore to maintain the desired plant production,
- combination of higher grade ore at greater depth versus lower grade at shallower depth.

The geometry-sequencing decisions then become even more complex.

It has been assumed for the sake of simplicity that the 'ore' is of one quality. Generally the values will vary in X–Y–Z space. The same is true for rock quality. Hence new dimensions are added to an already complex overall mine geometry-sequencing problem. Furthermore, the 'simple' addition of a haulage road to provide additional access can have a major effect on mine geometry and economics.

4.12 SUMMARY

In summary, pit geometry at any given time is influenced by many factors. Obviously the overlying material must be removed prior to removing that underlying. A certain operating space is needed by the equipment for efficiently removing the rock. The slope materials largely dictate the slope angles which can be safely used. In addition the sequencing of these geometries is extremely important so that the desired economic result (revenue and costs) is realized. Production rates, ore reserves and mine life are often highly price dependent. Hence mining geometry is a dynamic rather than static concept. To evaluate the many individual possibilities and combinations of possibilities, the computer has become invaluable.

The planning engineer must fully understand the basic geometric components which are combined to yield the overall pit geometry at any time in the life of the mine.

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* For more information see the end of the references.

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REVIEW QUESTIONS AND EXERCISES

1. Summarize the steps in the development of an open pit mine.
2. What "geometries" are involved in pit development?
3. Define or describe the following terms:
   - bench height
   - crest
   - toe
   - bench face angle
   - back break
   - bench floor
   - bench width
   - working bench
   - cut
   - safety bench/catch bench
   - double benches
   - berms
   - angle of repose
4. What are the purposes of safety benches?
5. What is the width of a safety bench?
6. What is the function of a safety berm?
7. What are some typical guidelines for a safety berm?
8. Discuss some of the aspects that enter into bench height selection?
9. Draw a sequence of three benches. Label the crest, toe, bench face angle, bench width and bank width.
10. If the bench face angle is 69°, the bench width is 30 ft and the bench height is 45 ft, determine the overall slope angle.
11. What happens if due to poor excavation practices the actual bench face angle is 66° instead? There are two possibilities to be considered.
12. Discuss the significance of Figure 4.3.
13. What are the purposes of safety benches?
14. Discuss the pro's and con's of double benching. Discuss the practical actions required to create double benches.
15. In actual surface mining operations, what happens to the "catch" benches created during the general operations? How does this affect their function? What actions might need to be taken?
16. Call has suggested the catch bench geometries shown in Figure 4.7 and in Table 4.1. Draw the geometry suggested for a 30m (double bench) height. What would be the corresponding final slope?
17. Discuss the pro's and con's of higher versus lower bench heights.
18. The dimensions for a Bucyrus Erie (BE)9 yd3 shovel are given in Figure 4.9. Some comparable dimensions for larger BE shovels are given below:

Dipper capacity (yd^3)	Dimension						
	A	B	D	E	G	H	I
12	22′–3″	40′–9″	37′–6″	50′–0″	34′–0″	7′–1″	39′–3″
20	30′–0″	55′–0″	50′–6″	65′–5″	44′–9″	9′–3″	50′–1″
27	30′–6″	57′–0″	48′–6″	67′–6″	44′–3″	6′–3″	52′–0″

Based upon the shovel geometries, what would be an appropriate maximum bench height for each?

19. In the largest open pit operations today, the BE 495 or P&H 4100 shovels are being used. Obtain dimensions similar to those given in the table in problem 18 for them.
20. Using Figure 4.10, what would be the reach height for a shovel with a 56 yd^3 dipper capacity?
21. The dipper capacities which are provided for a particular shovel model, normally are based upon material with a density of 3000 lbs/yd^3. What is done if the particular shovel model is used to dig coal? To dig magnetite?
22. Summarize the steps which would be followed in considering the appropriate bench geometry.
23. Compare the digging profile for the shovel shown in Figure 4.9 with a bench face drawn at a 65° angle. What is your conclusion?
24. Summarize the discussion of ore access as presented in section 4.3.
25. A drop cut example has been presented based upon the 9 yd^3 shovel. Rework the example assuming that the 27 yd^3 capacity is used instead. Find the minimum and maximum cut widths. Select an appropriate Caterpillar truck to be used with this shovel (use the information on their website).
26. What would be the volume of the ramp/drop cut created in problem 25?
27. Discuss the different aspects which must be considered when selecting the ore access location.
28. Figure 4.23 shows the situation where the ramp construction is largely in ore. Assume that the diameter of the orebody is 600 ft, the ramp width is 100 ft, the road grade is 10% and the bench height is 40 ft. Determine the approximate amount of ore removed in Figure 4.24.
29. Determine the amount of waste that would be removed if the ramp in Figure 4.24 was entirely constructed in waste.
30. Summarize the factors associated with the ramp location decision.
31. Figures 4.21 through 4.25 show the addition of a ramp to a pit. In this new case the orebody is assumed to be 600 ft in diameter, the road is 100 ft wide and the grade is 10%. Using AutoCad redo the example. What is the final ramp length? As shown, a flat portion 200 ft in length has been left between certain ramp segments.
32. Once the access to the newpit bottom has been established, discuss the three approaches used to widen the cut.
33. Summarize the steps used to determine the minimum required operating room when making parallel cuts.
34. What is the difference between the single and double spotting of trucks? Advantages? Disadvantages?
35. Redo the cut sequencing example described in section 4.4.6 assuming a cut width of 150 ft. If the shovel production rate is 50,000 tpd, how long would it take to exhaust the pushback?
36. A pit is enlarged using a series of pushbacks/laybacks/expansions. What would be the minimum and maximum cut widths using the 27 yd^3 shovel and Caterpillar model 789 trucks, assuming single pass mining? Work the problem assuming both single and double spotting of trucks.
37. Discuss the advantages/disadvantages of double spotting.
38. Redo the example described in section 4.4.5 assuming Caterpillar 993 trucks and the P&H 4100 shovel. The bench height is 50 ft and the bench face angle is 70°.
39. Assume that five 50 ft high benches are being worked as a group (Figure 4.51). Using the data from Problem 38, what would the working slope angles be when the working bench is at level 2?

40. For the slopes identified as "critical" in Table 4.2, what types of actions should be taken?
41. How might the design slope angles change during the life of a mine?
42. How do the expected stress conditions in the walls and floor of the pit change as the pit is deepened. Make sketches to illustrate your ideas.
43. What might happen at the pit bottom for the situation shown in Figure 4.61?
44. What are the four most common types of slope failure? Provide a sketch of each.
45. Assume that a 50 ft high bench is as shown in Figure 4.63. The layering goes through the toe. Assume that the following apply:
    $\phi = 32°$
    $c = 100$ kPa
    $\rho = 2.45$ g/cm^3
    Bench face angle = 60°
    Bedding angle = 20°
    What is the safety factor?
46. In section 4.6.3 it was determined that for the given conditions the safety factor was 1 ($F = 1$) for a slope of height 140 ft and a slope angle of 70°. Determine the minimum slope angle if the slope height is 200 ft instead.
47. Redo problem 46 if the required safety factor is 1.2. (See Figure 4.65).
48. Redo problem 46 assuming the presence of a tension crack of length 20 ft. You should consider the two extreme cases: (a) Crack dry; (b) Crack filled with water.
49. What is the safety factor for a slope assuming that the following apply:
    $H = 200$ ft
    Density = 165 lb/ft^3
    $c = 825$ lb/ft^2
    $Z_o = 50$ ft
    $H_w = 100$ ft
    $i = 40°$
    $\beta = 30°$
    $\phi = 30°$
50. A waste dump 500 ft high is planned. It is expected that the face angle will be 50°, the density is assumed to be 1.4 g/cm^3, the cohesion = 0 and the friction angle is 30°. Will it be stable?
51. What techniques might be used to obtain values for ϕ and c appropriate for the materials making up slopes?
52. Discuss the effect of slope wall curvature (in plan) on stability.
53. A discussion of the Afton copper/gold mine has been presented. Check the literature/Internet to see what eventually happened to the slopes.
54. In Figure 4.72, if the entire pit consisted of Structural Region B, what would happen to the North and East walls?
55. Assume that the conical pit shown in Figure 4.74 has a bottom radius of 100 ft. Redraw the figure showing the volume-slope dependence.
56. Discuss the pros' and con's of the different ways of representing bench positions on a plan map.
57. Redo the example in Figure 4.88 using AutoCad.
58. List some of the important questions that must be answered when siting a road.
59. The steps in the design of a spiral road inside the wall have been presented in section 4.8.2. Redo the example assuming a grade of 8%, a bench height of 40 ft, a bench face angle of 65° and a crest to crest dimension of 80 ft. The road width is 100 ft.
60. Redo problem 58 using AutoCad.

61. Redo the example in section 4.8.3 using AutoCad.
62. Redo the example in section 4.8.4 using AutoCad.
63. Why is it generally desirable to avoid switchbacks in a pit? Under what conditions might it become of interest?
64. If the conditions are such that a switchback becomes interesting, what should the planner do?
65. Assume that a conical pit has a depth of 500 m and the overall slope angle is 38°. Consideration is being given to adding a second access. The road would have a width of 40 m. How much material would have to be mined? Follow the approach described in section 4.8.5.
66. Using the data for the Caterpillar 797 haulage truck, answer the following questions:
    a. Load distribution of the front and rear tires.
    b. Contact area for an inflation pressure of 80 psi.
    c. What wheel loading should be used in the road design?
67. In the design and construction of a mine haulage road, what layers are involved? Describe each one starting at the lowest layer.
68. Describe the considerations in determining layer thickness.
69. How do you include the effect of dual-wheel loading?
70. What is meant by the California Bearing Ratio? How is it determined?
71. Suggest a haulage road width for two-way traffic involving Cat 797 trucks.
72. The following road cross-section dimensions apply at a particular mining operation which uses Komatsu 930 trucks:
    - safety berm width = 3.5 m
    - truck width = 7.3 m
    - space between trucks = 5.0 m
    - width of drainage ditch = 2.0 m
    - overall road width = 25 m
    - bench face angle = 75°

    Draw the section. How well does this design correspond to the "rules"?
73. Why are well-designed roads important?
74. How is a runaway ramp constructed?
75. What would happen to the road section shown in Figure 4.117 with the passage of a fully-loaded Cat 793 haulage truck? Be as specific as possible.
76. Assume that the sub-grade is a compact sand-clay soil and that you have the following construction materials: well-graded crushed rock, sand, well-graded gravel. What thicknesses would you recommend when using the Cat 793 trucks? Assume all of the materials have the same cost.
77. What is meant by poorly graded material?Well graded material?
78. In the design of a straight haulage road segment, what major factors should be considered?
79. Would the road design shown in Figure 4.118 apply for the Cat 793 truck? Why or why not?
80. What is the design rule for roadway width assuming two-way traffic?
81. List the steps to be followed in selecting a road design width.
82. What is meant by cross-slope? What are the rules involved? Why is it used?
83. If you were a haulage truck driver, what effect would cross-slope have on you?
84. What is meant by centrifugal force? How does it apply to road design?
85. What special design procedures must be applied to curves? What is meant by super-elevation?
86. What is the effect of rain and snow on super-elevated road segments?

87. Assume a curve radius of 50 ft (inner edge of pavement) and two-lane traffic. The truck is a Cat 793. What should be the minimum width? What should be the super-elevation assuming a curve speed of 15 mph?
88. What are transition zones?
89. Summarize the rules regarding the need for berms/guardrails.
90. What are the two principal berm designs in common use today?
91. On an elevated road involving Cat 793 haulage trucks, suggest a parallel berm design.
92. Summarize the rules presented by Couzens regarding haulage road gradients.
93. Summarize the practical road building and maintenance tips offered by Winkle.
94. Define what is meant by:
    a. Stripping ratio
    b. Overall stripping ratio
    c. Instantaneous stripping ratio
95. Indicate some of the different units used to express stripping ratio for different mined materials.
96. Redo the example shown in Figure 4.127 assuming that:
    - ore width = 400 ft
    - bench height = 40 ft
    - minimum pit width = 200 ft
    - original pit depth = 280 ft (7 benches)

    Calculate the appropriate values for SR(instantaneous) and SR(overall) for the mining of benches 8, 9, and 10.
97. What is meant by the term push-back?
98. Discuss the concepts of geometric sequencing. What is meant by a "phase"?
99. Discuss the different possibilities involved in pit sequencing.