

MINING ENGINEERING 1
APPLIED MATHEMATICS

A. 1 OPEN PIT MINING PROBLEM:ok

August, 1981. A new mining project has a blocked positive reserve of 2,000,000 WMT of iron ore. Specific gravity of the ore is 4 and moisture content is 7.5%. With a waste to ore volumetric ratio of 3:1 it was decided to mine the ore by open pit. The Mining Engineer recommended to the management that before the ore extraction starts, 100,000 metric tons of ore be exposed first. The occurrence is such that no matter how the orebody is approached, the waste to ore ratio remains constant.

- a. Compute volume of overburden to be stripped to expose 100,000 dry metric tons of ore.
- b. At a waste stripping rate of 1,000 cubic meters per day, how long will it take to do the initial stripping before mining starts.
- c. After the initial stripping, it is imperative that the waste to ore ratio is maintained with the same waste stripping rate of 1,000 cubic meters per day. Compute ore extraction rate in dry metric tons per day.
- d. Allowing 10% ore dilution, what is the expected life of the mine assuming 330 working days a year.

Solution:

a. Volume of waste (overburden) / Volume of ore = 3

Volume of ore = 100,000 DMT / 0.925 = 108,108 WMT x 1 cu.m/4 = 27,027 cu.m.

Volume of overburden = 3 x volume of ore = 3 x 27,027 cu.m. = 81,081 cu.m.

b. Let : N = no. of days to do initial stripping

N = 81,081 cu.m. / 1,000 cu.m./day = 81 days

c. Waste Extraction rate / Ore Extraction rate = 3

Ore Extraction rate = (Waste Extraction rate / 3) = (1,000 cu.m./day / 3) = 333.33 cu.m./day x 4 WMT/cu.m. = 1,333.33 WMT/day x (1-Moisture) = 1,233.33 DMT/day

d. Life of Mine = Total Ore Tonnage / Ore Extraction rate = (2,000,000 WMT x 1.1) / (1,333.33 WMT/day x 330 days/year)

$$\text{Life of Mine} = \underline{\underline{5 \text{ years}}}$$

A.2 OPEN PIT MINING PROBLEM:

August, 1984. An exploration project has resulted in the delineation of a mineable ore reserve of 2,000,000 WMT of low grade gold ore. Specific gravity of both the ore and waste is 2.5 with a moisture content of 8%. Mine management decided to mine the ore by open pit with a waste to ore ratio of 3:1 and involving a pre-stripping of 120,000 m³ waste. Assuming that the waste to ore ratio will remain constant throughout the life of the mine.

- Compute the total tonnage in DMT of ore to be exposed after the preliminary stripping activities.
- At a waste stripping capacity of equipment at 2500 WMT/DP, how long will it take to do the initial stripping before mining starts?
- With the same stripping rate, what will be the ore extraction rate in dry metric tons per day?
- Allowing 10% for ore dilution, what is the projected life of the mine assuming 300 working days a year?

Solution:

$$\begin{aligned}
 \text{a.} \quad & \frac{\text{Volume of waste}}{\text{Volume of ore}} = 3 \\
 & \frac{120,000 \text{ cu.m.}}{\text{Volume of ore}} = 3 \\
 & \text{Volume of ore} = 40,000 \text{ cu.m.} \\
 & \begin{aligned} \text{Tonnage of ore} &= 40,000 \text{ cu.m.} \times 2.5 \text{ WMT/cu.m} \\ \text{Tonnage of ore} &= 100,000 \text{ WMT} \times 0.92 \text{ DMT/WMT} \\ \text{Tonnage of ore} &= \underline{\underline{92,000 \text{ DMT}}} \end{aligned} \\
 \text{b.} \quad & \text{Let ;} \\
 & N = \text{no.of days to do initial stripping} \\
 & N = \frac{\text{Weight of waste}}{\text{Waste extraction rate}} \\
 & N = \frac{120,000 \text{ cu.m} \times 2.5 \text{ WMT/cu.m}}{2,500 \text{ WMT/day}} \\
 & N = \underline{\underline{120 \text{ days}}} \\
 \text{c.} \quad & \begin{aligned} \text{Ore Extraction rate} &= \frac{\text{Waste Extraction rate}}{3} \\ \text{Ore Extraction rate} &= \frac{2,500 \text{ WMT/day}}{3} \\ \text{Ore Extraction rate} &= 833.33 \text{ WMT/day} \times 0.92 \\ \text{Ore Extraction rate} &= \underline{\underline{766.67 \text{ DMT/day}}} \end{aligned} \\
 \text{d.} \quad & \begin{aligned} \text{Life of Mine} &= \frac{\text{Total Reserve}}{\text{Ore Extraction rate}} \\ \text{Life of Mine} &= \frac{2,000,000 \text{ WMT} \times 1.1}{833.33 \text{ WMT/day} \times 300 \text{ days/year}} \\ \text{Life of Mine} &= \underline{\underline{8.8 \text{ years}}} \end{aligned}
 \end{aligned}$$

A.3 OPEN PIT MINING PROBLEM:

August, 1988. A gold mine extracts ore from several small pits to feed its mill. An orebody was determined to have a tonnage of 14,000 tons and a grade of 3.5 gms Au/ton. Cost of mining is detailed as follows:

Dozing	=	P460.00/hr
Dozing capacity	=	166.4 bcm/hr
Loading	=	P5.85/lcm (swell factor of 1.64)
Hauling	=	P13.60/lcm/km
Other services (ore only)	=	P60.00/ton

Explanation of terms:

Bcm - bank cubic meters - the volume of in-situ material

Lcm - loose cubic meter - the volume of broken material

The in-situ material when blasted becomes 1.64 lcm of muck. Assume the specific gravity of ore and waste is 2.2. The orebody is 0.90 km away from the mill and 0.80 km away from the waste dump. Cost of milling is P168.00/ton milled. Mine General overhead, depreciation, depletion, amortization, interest and other charges, and administration totaling P300.00/ton milled. Gold price is \$435/oz and the exchange rate is P21.00 per US \$1.00. Historical record shows that ore grade determined at the pit drops by 10% by the time the ore reaches the mill. Recovery of metal at the mill is 85%. Assume that gross metal value equals revenue from sale of metal.

- Determine the cost for every ton of ore extracted. (Ans=P74.74/ton)
- Determine the cost for every ton of waste mined. (Ans=P13.73/ton)
- Using X as the corresponding waste tonnage for every ton of ore mined, present a formula for total cost (all cost items included) of mining the 14,000 tons of ore and the corresponding waste.
- Determine the total metal sale considering the two-step drop of grade from ore grade to recovered grade. (Ans=P11,007,675)
- Equate item c to item d and determine the value of X
- What is the breakeven stripping ratio? (Ans=17.74:1)

Solution:

a. Cost /ton of ore extracted:

$$\text{Volume} = \frac{\text{weight} \quad 1 \text{ ton}}{\text{Density} \quad 2.2 \text{ tons/bcm}} = 0.4545 \text{ bcm}$$

$$\text{Dozing Time} = \frac{\text{Volume} \quad 0.4545 \text{ bcm}}{\text{Dozing capacity} \quad 166.4 \text{ bcm/hr}} = 0.00273 \text{ hour}$$

$$\text{a.1 Dozing} = P460/\text{hr} \times 0.00273 \text{ hr} = P 1.26$$

$$\text{a.2 Loading} = P5.85/\text{lcm} \times 0.4545 \text{ bcm} \times 1.64 \text{ lcm/bcm} = P 4.36$$

$$\text{a.3 Hauling} = P13.60/\text{lcm-km} \times 0.4545 \text{ bcm} \times 1.64 \text{ lcm/bcm} \times 0.9 \text{ km} = P 9.12$$

$$\text{a.4 Other Services} = P60.00$$

$$\text{Total Cost per ton of ore extracted} = P74.74/\text{ton}$$

ore

b. Cost / ton of waste mined:

$$\text{b.1 Dozing} = P460/\text{hr} \times 0.00273 \text{ hr} = P 1.26$$

$$\text{b.2 Loading} = P5.85/\text{lcm} \times 0.4545 \text{ bcm} \times 1.64 \text{ lcm/bcm} = P 4.36$$

$$\text{b.3 Hauling} = P13.60/\text{lcm-km} \times 0.4545 \text{ bcm} \times 1.64 \text{ lcm/bcm} \times 0.8 \text{ km} = P 8.11$$

$$\text{Total Cost per ton of waste mined} = P13.73/\text{ton}$$

waste

c. Total Cost

Let, X = tons of waste per ton of ore

Total Cost = Cost of mining ore + Cost of mining waste + Milling Cost + Other Costs

c.1 Cost of Mining ore = $P74.74/\text{ton} \times 14,000 \text{ tons}$ ----- P
1,046,360

c.2 Cost of Mining waste = $\frac{(P13.73)}{\text{tons-waste}} \times \frac{(X \text{ tons-waste})}{\text{tons-ore}} \times 14,000 \text{ tons}$ ----- P
192,220 X

c.3 Cost of Milling = $P168/\text{ton} \times 14,000 \text{ tons}$ ----- P 2,352,000

c.4 Other Costs = $P300/\text{ton} \times 14,000 \text{ tons}$ ----- P
4,200,000

c.5 Total Cost = $P7,598,360 + P192,220 X$

d. Total Metal Sale:

Mill Head = 10% less of Mine head = $3.5 \text{ gms-Au/MT} \times 0.90 = 3.15 \text{ gms-Au/MT}$

Wt. of metal in concentrate = Wt. of metal in ore $\times$ Recovery

Wt. of metal in concentrate = $14,000 \text{ tons} \times 3.15 \text{ gms-Au/ton} \times 0.85$

Wt. of metal in concentrate = $37,485 \text{ gms} \times 1 \text{ oz}/31.1035 \text{ gms}$

Wt. of metal in concentrate = 1,205 oz.

Total metal sale = $1,205 \text{ oz.} \times \$435/\text{oz} \times P21/\$$

Total metal sale = P11,007,675

e. Total Waste Tonnage

Total Cost	=	Total Metal Sale
$P7,598,360 + P192,220 X$	=	P11,007,675
P192,220 X	=	P3,409,675
		Tons-waste
X	=	$17.74 \frac{\text{tons-waste}}{\text{tons-ore}}$

f. Breakeven Stripping Ratio = $\frac{\text{volume of waste}}{\text{volume of ore}}$

= **17.74**

A.4 OPEN PIT MINING PROBLEM:

August, 1987. A gold mine extracts ore from several small pits to feed its mill. An orebody was determined to have a tonnage of 14,000 tons and a grade of 3.5 gms Au/ton. Cost of mining is detailed as follows:

Dozing	=	P460.00/hr
Dozing capacity	=	166.4 bcm/hr
Loading	=	P5.85/lcm (swell factor of 1.64)
Hauling	=	P13.60/lcm/km
Other services (ore only)	=	P60.00/ton

The pit is 0.90 km away from the mill and 0.80 km away from the waste dump. The mill head is 85% of the mine cut sample. The mill recovery is 80%. Milling cost is P168.00/ton milled. Other relevant costs are: Mine General overhead, depreciation, depletion, amortization, interest and other charges, and administration totaling P300.00/ton milled. Metal prices are as follows: \$470/oz for gold and \$8.00/oz for silver with an exchange rate of P20.50 per US \$1.00. Assume that gross metal value equals revenue from sale of metal. Further assume that ore and waste rock specific gravity is 2.2. Determine the breakeven stripping ratio of said orebody. (Ans=14.2)

Solution:

At Breakeven: Operating Costs = Revenue from sale of metal

Operating Cost = Mining Cost + Milling Cost + Other relevant cost

A. Mining Cost = Cost of Mining Ore + Cost of Mining Waste

a. Cost of Mining Ore:

Dozing cost = Dozing rate x Dozing ore period

Dozing ore period = Total ore tonnage / Dozing capacity

Dozing ore period = 14,000 tons / 166.4 bcm/hr x 2.2 tons/bcm

Dozing ore period = 38.243 hrs.

17,592 Dozing Cost = P460/hr x 38.243 hrs = P

P61,053 Loading Cost = P5.85/lcm x 14,000 tons x 1.64 lcm/bcm =

2.2 tons/bcm

P127,741 Hauling Cost = P13.60/lcm-km x 14,000 tons x 1.64 lcm/bcm x 0.9 km =

2.2 tons/bcm

Other Services = P60.00/ton x 14,000 tonnes

= P840,000

P1,046,386 Total Cost of Mining Ore =

b. Cost of Mining Waste:

Dozing Cost = Dozing rate x Dozing waste period

Dozing waste period = Total waste tonnage / Dozing capacity

$$\begin{aligned}
 & \text{Dozing waste period} = \frac{W \text{ tons}}{166.4 \text{ bcm/hr} \times 2.2 \text{ tons/bcm}} \\
 & \text{Dozing waste period} = \frac{W}{366} \text{ hr} \\
 & \text{Dozing cost} = \text{P460/hr} \times \frac{W}{366} \text{ hr} = \text{P1.257W} \\
 & \text{Loading Cost} = \text{P5.85/lcm} \times \frac{W \text{ tons}}{2.2 \text{ tons/bcm}} \times 1.64 \text{ lcm/bcm} = \text{P4.361W} \\
 & \text{Hauling Cost} = \text{P23.60/lcm-km} \times \frac{W \text{ tons}}{2.2 \text{ tons/bcm}} \times 1.64 \text{ lcm/bcm} \times 0.8 \text{ km} = \text{P8.111W} \\
 & \text{Total Cost of Mining Waste} = \text{P13.729W} \\
 & \text{Then:} \\
 & \text{Mining Cost} = \text{Total cost of mining ore} + \text{Total cost of mining waste} \\
 & \text{Mining Cost} = \text{P1,046,386} + \text{P13.729W} \text{ ----- Equation 1} \\
 & \text{B. Milling Cost} = \text{P168/ton} \times 14,000 \text{ tons} = \text{P2,352,000.00} \\
 & \text{C. Other Costs} = \text{P300/ton} \times 14,000 \text{ tons} = \text{P4,200,000.00} \\
 & \text{D. Total Operating Cost} = \text{Mining Cost} + \text{Milling Cost} + \text{Other Cost} \\
 & \text{Total Operating Cost} = (\text{P1,046,386.00} + \text{P13.729W}) + \text{P2,352,000.00} + \text{P4,200,000.00} \\
 & \text{Total Operating Cost} = \text{P7,598,386} + 13.729W \\
 & \text{Solving for the Revenue from Sale of Metal:} \\
 & \text{Revenue} = \text{Weight of metal} \times \text{Price of metal} \\
 & \text{Recovery} = \frac{\text{Wt. of metal in Concentrate (Cc)}}{\text{Wt. of metal in ore (Ff)}} \\
 & \text{Wt. of metal in concentrate} = \text{Wt. of metal in ore} \times \text{Recovery} \\
 & \text{Wt. of metal in ore} = \text{Wt. of ore} \times \text{Mill Head} \\
 & \text{Mill Head} = 0.85 \times \text{Mine cut sample} \\
 & \text{Mill Head} = 0.85 \times 3.5 \text{ gm Au/MT} \\
 & \text{Mill Head} = 2.975 \text{ gm Au/MT} \\
 & \text{Wt. of metal in ore} = 14,000 \text{ MT} \times 2.975 \text{ gm Au/MT} \\
 & \text{Wt. of metal in ore} = 41,650 \text{ gms Au} \\
 & \text{Wt. of metal in concentrate} = 41,650 \text{ gms Au} \times 0.80 \\
 & \text{Wt. of metal in concentrate} = 33,320 \text{ gms Au} \times 1\text{oz./31.1035 gms} \\
 & \text{Wt. of metal in concentrate} = 1,071.262 \text{ oz. Au} \\
 & \text{Then;} \\
 & \text{Revenue} = 1,071.262 \text{ oz} \times \$470/\text{oz} \times \text{P20.50}/\$ \\
 & \text{Revenue} = \text{P10,321,610.00} \\
 & \text{But:} \\
 & \text{Operating Cost} = \text{Revenue} \\
 & \text{P7,598,386} + \text{P13.729W} = \text{P10,321,610} \\
 & \text{P13.729W} = \text{P2,723,224} \\
 & \text{W} = 198,356 \text{ tons} \\
 & \text{Therefore:} \\
 & \text{Breakeven Stripping Ratio} = \frac{\text{Wt. of waste}}{\text{Wt. of metal}}
 \end{aligned}$$

$$\begin{aligned} \text{Breakeven Stripping Ratio} &= \frac{\text{Wt. of ore}}{\text{198,356 tons}} \\ &= \frac{\text{14,000 tons}}{\text{198,356 tons}} \\ \text{Breakeven Stripping Ratio} &= \mathbf{14.2} \end{aligned}$$

A.5 OPEN PIT MINING PROBLEM:ok

August 2001. A Mine Planner is locating the new ultimate pit boundary or limit of an open pit gold mine due to increase of gold prices in the world market and additional ore reserves. Determine the maximum allowable stripping ratio given the following:

- | | |
|---------------------------------------|--------------|
| Value of ore | = P25/MT |
| Stripping Cost for overburden | = P5/MT |
| Production Cost (excluding stripping) | = P15/MT |
| Pit Slope | = 60 degrees |
| Bench height | = 5 m |
| Berm Width | = 2.25 m |
| Road gradient | = 12% |
- a. 5 : 1 b. 3.5 : 1 c. 6 : 1 d. **2 : 1**

Solution:

$$\text{Stripping ratio} = \frac{\text{Weight of waste}}{\text{Weight of ore}}$$

Assume:

$$\text{Weight of ore} = 1 \text{ MT}$$

Let:

$$W = \text{weight of waste}$$

Assume: Operating Cost = Value of ore

Operating Cost = Cost of mining ore + Cost of stripping overburden

$$\begin{aligned} (1\text{MT} \times \text{P15/MT}) + (W \times \text{P5/MT waste}) &= 1\text{MT} \times \text{P25/MT} \\ \text{P15} + (\text{P5} \times W) &= \text{P25} \end{aligned}$$

$$\text{P5} \times W = \text{P10}$$

$$W = \frac{\text{P10}}{\text{P5}}$$

$$W = 2$$

Therefore:

$$\mathbf{\text{Stripping Ratio} = 2:1}$$

A.6 OPEN PIT MINING PROBLEM:

August 2003:

Given the following:

Mining Cost	= P50.00/MT
Milling Cost	= P200.00/MT
Marketing Cost	= \$55.00/oz.
Administration Cost	= P50.00/MT
Stripping Cost	= P25.00/MT
Excise Tax	= 2% of revenue
Gold Price	= \$350.00/oz.
Recovery	= 85%
Stripping ratio	= 6:1
Forex	= P30.00/\$

- How many gm Au/MT (grade to breakeven)?
a. **1.62 gm /MT** b. 2.01 gm /MT c. 1.28 gm /MT d. 1.89 gm /MT
- How many oz/MT
a. 0.15 oz/MT b. 0.05 oz/MT c. 0.11 oz/MT d. 0.10 oz/MT
- If the mine call factor is 15%, what should be the grade in gmAu/MT that you will mine?
a. 2.12 gm /MT b. 1.56 gm /MT c. 1.83 gm /MT d. **1.91 gm /MT**
- If the mill grade is 3 gmAu/MT, what is the grade of the ROM assay?
a. 3.21 gm /MT b. 3.15 gm /MT c. **3.53 gm /MT** d. 3.65 gm /MT

Solution:

Assume :

Weight of ore = 1 MT

Then:

Weight of waste = 6 MT

Let: W = weight of metal in concentrate

Solving for Revenue:

Revenue = W (gm) x 1 oz/31.1035 gm x \$350/oz x P30/\$
Revenue = P337.58 W

Solving for Operating Cost:

Mining Cost	= P50.00/MT x 1 MT	-----	P50.00
Stripping Cost	= P25.00/MT x 6 MT	-----	P150.00
Milling Cost	= P200/MT x 1MT	-----	P200.00
Admin Cost	= P50.00/MT x 1MT	-----	P50.00
Marketing Cost	= \$55.00/oz x P30/\$ x 1oz/31.1035 gm x W(gm)---		
P53.05 W			
Excise Tax	= 0.02 x (P337.58 W)	-----	P 6.75 W
	Total Operating Cost	-----	P450 +

59.80W

To Breakeven:

Total Operating Cost	=	Revenue
450 + 59.8W	=	337.58W
277.78W	=	450
W	=	1.62 gms

Then:

$$\begin{aligned}
 1. \text{ Mill grade} &= 1.62 \text{ gms/1MT} \\
 \text{Mill grade} &= \underline{\underline{\mathbf{1.62 \text{ gms/MT}}}} \\
 2. \text{ Oz./MT} &= 1.62 \text{ gms/MT} \times 1 \text{ oz/31.1035 gms} \\
 \text{Oz./MT} &= \underline{\underline{\mathbf{0.052 \text{ oz./MT}}}}
 \end{aligned}$$

$$3. \text{ Mine Call Factor} = \frac{\text{Mine grade} - \text{Mill Grade}}{\text{Mine grade}}$$

$$\text{Mine Call Factor} = 1 - \frac{\text{Mill Grade}}{\text{Mine Grade}}$$

$$\frac{\text{Mill Grade}}{\text{Mine Grade}} = 1 - \text{Mine Call Factor}$$

$$\text{Mine Grade} = \frac{\text{Mill Grade}}{1 - \text{Mine call factor}}$$

$$\text{Mine Grade} = \frac{1.62 \text{ gms/MT}}{(1 - 0.15)}$$

$$\text{Mine Grade} = \underline{\underline{\mathbf{1.91 \text{ gms/MT}}}}$$

4. If Mill grade = 3.0 gms/MT

$$\text{Mine Grade} = \frac{3.0 \text{ gms/MT}}{(1 - 0.15)}$$

$$\text{Mine Grade} = \underline{\underline{\mathbf{3.53 \text{ gms/MT}}}}$$

A.7 OPEN PIT MINING PROBLEM:ok

August 2000. In the same gold mining operation, the management is looking for a new mining contractor. One of its prospected mining contractors submitted a bid that will make the mine spend about P18,000,000.00 in both its mining and stripping alone if it accepted the contract. If cost of stripping is about 75% of the mentioned cost, determine the following:

1. Cost/MT of mining if total ore to be handled is 90,000 MT?
 - a. P40/MT b. P60/MT c. **P50/MT** d. P70/MT
2. Cost/MT of stripping if total waste to be handled is 540,000 MT.
 - a. P30/MT b. **P25/MT** c. P75/MT d. P40/MT

Solution:

$$\begin{aligned}
 \text{Stripping cost} &= 0.75 \times \text{P18,000,000} \\
 \text{Stripping cost} &= \text{P13,500,000.00}
 \end{aligned}$$

$$\begin{aligned}
 \text{Mining Cost} &= \text{P18,000,000} - \text{P13,500,000} \\
 \text{Mining Cost} &= \text{P4,500,000.00}
 \end{aligned}$$

$$1. \text{ Mining cost/MT} = \frac{\text{P4,500,000}}{90,000 \text{ MT}}$$

$$\text{Mining Cost/MT} = \underline{\underline{\mathbf{P50.00/MT}}}$$

$$\begin{aligned}
 2. \text{ Stripping cost/MT} &= \frac{\text{P13,500,000}}{\text{540,000 MT}} \\
 \text{Stripping cost/MT} &= \underline{\text{P25.00/MT}}
 \end{aligned}$$

A.8 OPEN PIT MINING PROBLEM:ok

August 2003: You are the manager of the company contracted to do pre-stripping of a large copper deposit minable by open pit with total minable reserves of 15 million tons and an overall stripping ratio of 2 to 1 which the mine owner wants to reduce to 1 to 1 during operations. At what average daily rate will you have to pre-strip the mine if you are given only 6 months and you have 25 working days per month.

- a. 30,000 TPD b. 60,000 TPD c. **100,000 TPD** d. 120,000 TPD

Solution:

Since stripping ratio was reduced to 1:1 during operations, then,

$$\text{Total Waste} = 15,000,000 \text{ MT}$$

Then:

$$\text{Stripping rate} = \frac{15,000,000 \text{ MT}}{6 \text{ months} \times 25 \text{ days/month}}$$

$$\text{Stripping rate} = \underline{\text{100,000 TPD}}$$

A.9 OPEN PIT MINING PROBLEM:ok

August 2005: In a typical bench mining operation, 1 ft of ore face must be exposed for each 4 to 5 tons of required daily production. Supposed the required daily production is 20,000 tons, then advance stripping of the bench face must continue until:

- a. **4,000 - 5,000 ft** b. 10,000 - 12,000 ft
 b. c. 2,000 - 3,000 ft d. 6,000 - 7,000 ft

Solution:

$$\text{Rate of Mining} = 4 - 5 \text{ tons/ft}$$

For 4 tons/ft :

$$\text{Advance} = \frac{20,000 \text{ tons}}{4 \text{ tons/ft}}$$

$$\text{Advance} = \underline{\text{5,000 ft.}}$$

For 5 tons/ft:

$$\text{Advance} = \frac{20,000 \text{ tons}}{5 \text{ tons/ft.}}$$

Advance = 4,000 ft.

A.10 OPEN PIT MINING PROBLEM:

August 2004: A currently operating pit copper mine is being re-optimized for pit expansion. The final or ultimate pit limit will be defined by the following planning and economic parameters:

Operating cost per ton	= P140.00
Capital cost per ton milled	= P 35.00
Metal Prices:	
Pound Copper	= 1.20 USD
Ounce Gold	= 380 USD
Ounce Silver	= 25 USD
Excise tax for copper	= 2%
Conversion rate	= P25.00/USD
Mill Recovery	= 85%
Concentrate Grade:	
Copper	= 25%
Gold	= 2 grams/MT
Silver	= 50 grams/MT

As Chief Mining Engineer, what design cut-off grade will you apply re-optimizing the open pit?

- a. 0.20% Cu b. 0.25% Cu c. **0.27% Cu** d. 0.19% Cu e. 0.22% Cu

Solution:

Assume:

Weight of ore = 1 MT

Recovery = $\frac{\text{Wt. of Cu in concentrate}}{\text{Wt. of Cu in feed}}$

Wt. of Cu. In concentrate = 1 MT x f (% Cu) x 0.85
Wt. of Cu in concentrate = 0.85f

But:
concentrate Wt. of Cu in concentrate = Wt. of concentrate, C (MT) x %Cu in
Wt. of Cu in concentrate = C x 0.25

Then:
 $\frac{C \times 0.25}{C} = \frac{0.85f}{3.4f}$

Wt. of Au in concentrate = 3.4f x 2 gms/MT
= 6.8f

Wt. of Ag in concentrate = 3.4f x 50 gms/MT
= 170f

Solving for Revenue:

1. Copper	= 0.85f (MT) x 2,000 lbs/MT x \$1.20/lb x P25/\$	-----
P51,000f		
2. Gold	= 6.8f (gms) x 1 oz/31.1035 gms x \$380/oz. x P25/\$	-----
P2,077f		
3. Silver	= 170f (gms) x 1 oz/31.1035 gms x \$25/oz. x P25/\$	-----
P3,416f		

Total Revenue ----- P56,493f

Solving for Total Cost:

1. Operating Cost	= P140/MT x 1 MT	-----	P140
2. Capital Cost	= P35/MT x 1 MT	-----	P35
3. Excise Tax	= 0.02 x P56,493f	-----	P1,130f

Total Cost ----- P175 +

P1,130f
Therefore:

Total Revenue	=	Total Cost
P56,493f	=	P175 + P1,130f
P55,363f	=	P175
f	=	0.0029 x 100%
f	=	0.29 % Cu

A.11 OPEN PIT MINING PROBLEM:

August, 2002. North Davao Mining Corporation if it re-opens its Amacan Copper Mine in Maco, Compostela Valley, will need to re-optimize its open pit mine using present conditions. The ultimate pit will therefore be defined by the following new set of planning, economic and operating parameters:

Operating cost per ton milled	- P 280.00
Capital Cost per ton milled	- 70.00
Metal Prices:	
Pound Copper	- 1.0 USD
Ounce Gold	- 400.0 USD
Ounce Silver	- 10.0 USD
Excise tax for copper	- 2%
Currency conversion rate	- P 51.50 per USD
Mill Recovery (Cu)	- 85%
Concentrate grades:	
Copper	- 27%
Gold	- 5 grams per MT
Silver	- 50 grams per MT

As the mine's Chief Mine Planning Engineer, calculate the design cut-off grade that will be applied in re-optimizing the open pit.

Solution:

$$\begin{aligned}
 \text{Feed} &= 1 \text{ MT} \\
 \text{Mill Recovery} &= \frac{C \times 0.27}{1 \text{ MT} \times f} \\
 0.85 &= \frac{0.27C}{f} \\
 C &= 3.15 f \quad \text{----- Equation 1}
 \end{aligned}$$

Solving for Weight of Metal:

Wt. of Cu	=	Cc (Cu)
Wt. of Cu	=	3.15 f (MT) x 0.27
Wt. of Cu	=	0.8505 f (MT) x 2,205 lbs/Mt
Wt. of Cu	=	1,875.35 f (lbs)
Wt. of Au	=	Cc (Au)
Wt. of Au	=	3.15 f (MT) x 5 gms/MT x 1 oz/31gms
Wt. of Au	=	0.508 f (oz)
Wt. of Ag	=	Cc (Ag)
Wt. of Ag	=	3.15 f (MT) x 50 gms/Mt x 1 oz/31 gms
Wt. of Ag	=	5.08 f (oz)

Solving for Value of Metal:

Value of Cu	=	1,875.35 f (lbs) x 1 USD/lb x P51.50/1 USD
Value of Cu	=	96,560 f
Value of Au	=	0.508 f (oz) x 400 USD/oz x P51.50/1 USD
Value of Au	=	10,465 f
Value of Ag	=	5.08 f (oz) x 10 USD/oz x P51.50/1 USD
Value of Ag	=	2,616 f

$$\begin{aligned}\text{Total Value of Metal} &= \text{Value of Cu} + \text{Value of Au} + \text{Value of Ag} \\ \text{Total Value of Metal} &= 109,661 \text{ f}\end{aligned}$$

Then:

$$\begin{aligned}\text{Excise tax} &= 2\% \text{ of Total Value of Metal} \\ \text{Excise tax} &= 0.02 \times 109,661 \text{ f} \\ \text{Excise tax} &= 2,193 \text{ f}\end{aligned}$$

To determine the cut-off grade:

$$\begin{aligned}\text{Total Value of Metal} &= \text{Total Operating Cost} \\ \text{Total Value of Metal} &= \text{Capital Cost} + \text{Operating Cost} + \text{Excise Tax} \\ 109,661 \text{ f} &= P70 + P 280 + 2,193 \text{ f} \\ 107,468 \text{ f} &= P 350 \\ \text{f} &= 0.00325 \times 100 \\ \text{f} &= \mathbf{0.325 \% \text{ Cu}}\end{aligned}$$

A.12 OPEN PIT MINING PROBLEM:

August, 1998. A mine has a mineable ore reserve of 100 million MT blocked by 70 drill holes at a grid drilling of 100m x 100m. Ore modeling was done by Inverse Distance Square Method using the flat search radius of 100 meters. Pit optimization was done using the computerized 4D Lerch Grossman Algorithm. As the Chief Mining Engineer of the company and using the Taylor's Formula for tonnage optimization, at what capacity range will you design the mine? (Ans=11,500 - 17,000 TPD)

Solution:

Taylor's Formula

$$\begin{aligned}\text{Mine Life, yrs} &= 6.5 \times \sqrt[4]{\text{Tonnage (millions)} \times (1 \pm 0.20)} \\ &= 6.5 \times \sqrt[4]{100 \times (1 \pm 0.20)} \\ &= 6.5 \times 3.162 \times (1 \pm 0.20) \\ &= 20.553 \times (1 \pm 0.20) \\ &= 16.44 \text{ yrs. to } 24.66 \text{ yrs.}\end{aligned}$$

$$\begin{aligned}\text{Capacity (16.44 yrs), TPD} &= 100,000,000 \text{ MT}/16.44 \text{ yrs} \\ &= 6,082,725 \text{ MT/yr} \times 1 \text{ yr}/365 \text{ days} \\ &= \mathbf{16,665 \text{ TPD}}\end{aligned}$$

$$\begin{aligned}\text{Capacity (24.66 yrs), TPD} &= 100,000,000 \text{ MT}/24.66 \text{ yrs} \\ &= 4,055,150 \text{ MT/yr} \times 1 \text{ yr}/365 \text{ days} \\ &= \mathbf{11,110 \text{ TPD}}\end{aligned}$$

A.13 OPEN PIT MINING PROBLEM:

August 2004, 2006, 2008. The operating parameters of a new open pit mine in Nueva Viscaya which is scheduled for development under the MPSA system are as follows:

Ore tonnage per annum	-	10,440,000 MT
Strip ratio	-	0.77 : 1
Material Specific Gravity	-	2.75
Bench Height	-	12.50 meters
Sub-Grade	-	2.50 meters
Drill Pattern	-	10 x 10 meters
Drill penetration rate	-	110.0 meters per shift
Drilling days per year	-	348 days
Drilling shift per day	-	3 shifts
Drill utilization	-	65%
Time efficiency	-	50 min per 60 min

As the newly hired Drilling and Blasting Superintendent, how many rotary drills (excluding spares) will you need to sustain the total broken muck requirements of the open pit. Consider the theoretically computed fraction unit as one (1) unit.

- A. 8 drills B. 4 drills C. 5 drills **D. 2 drills** E. 6 drills

Solution:

Ore Production = 10,440,000 MT/year

Total Waste = 0.77 x 10,440,000
Total Waste = 8,038,800 MT/year

Total Material = Ore Production + Total Waste
Total Material = 10,440,000 MT + 8,038,800 MT
Total Material = 18,478,800 MT/year

Depth of hole/bench = Bench Height + Sub-Grade
Depth of hole/bench = 12.5m + 2.5m
Depth of hole/bench = 15m

Drill penetration/day = 110 m / shift x 3 shifts/day
Drill penetration/day = 330 m / day

No. of holes/day = $\frac{330 \text{ m/day}}{15\text{m/hole}}$

No. of holes/day = 22 holes

Drill Production/day = 22 holes/day x Bench Height x 10m x 10m
Drill Production/day = 22 holes/day x (12.5m x 10m x 10m)
Drill Production/day = 22 holes/day x 1,250 cu.m/hole
Drill Production/day = 27,500 cu.m./day x 2.75 MT/cu.m.
Drill Production/day = 75,625 MT

Annual Production = 75,625 MT/day x 348 days/year x 0.65 x 50/60
Annual Production = 14,255,312 MT

No. of Drills needed = $\frac{\text{Total Material}}{\text{Annual Production/drill}}$

No. of Drills needed = $\frac{18,478,800 \text{ MT}}{14,255,312 \text{ MT/unit}}$

No. of Drills needed = 1.29 units

No. of Drills needed = **2 units**

A.14 OPEN PIT MINING PROBLEM:

August 2003, 2005. The recently discovered dessiminated porphyry copper deposit in Nueva Viscaya will be developed by open pit mining method. The operating parameters of the open pit mine are as follows:

Annual Mine Ore Production	-	10,440,000 Mt
Strip Ratio	-	0.77:1
Power Shovel Brand/Size	-	P&H 1900 AL
Shovel Bucket Size	-	9.19 cu.m.
Shovel Utilization	-	80%
Bucket Fill Factor	-	90%
Rated truck size	-	77.1 MT
Truck Fill Factor	-	95%
Total Truck spot time	-	0.33 min
Load cycle time	-	0.50 min
Time efficiency	-	50 min per 60 min

As the Mine Superintendent, how many power shovels will you use (excluding spares) to sustain ore delivery to the copper mills and at the same time maintain a workable pit geometry by promptly stripping the waste?

- A. 1 unit B. 6 units C. 3 units D. 8 units E. 5 units

Solution:

Ore Production	=	10,440,000 MT/year
Total Waste	=	0.77 x 10,440,000
Total Waste	=	8,038,800 MT/year
Total Material	=	18,478,800 MT/year
Shovel Capacity	=	Bucket size x Shovel Fill Factor x Utilization
Shovel Capacity	=	9.19 cu.m. x 0.90 x 0.80
Shovel Capacity	=	6.6168 cu.m.
Cycle time	=	Truck spot time + Load cycle time
Cycle time	=	0.33 min + 0.50 min
Cycle time	=	0.83 min
Effective time/year	=	365 days/year x 24 hrs/day x 50 min/hr
Effective time/year	=	438,000 min
Tonnage/shovel	=	$\frac{\text{Shovel Capacity}}{\text{Cycle Time}}$
Tonnage/shovel	=	$\frac{6.6168 \text{ cu.m.}}{0.83 \text{ min}} \times 438,000 \text{ min/year}$
Tonnage/shovel	=	3,491,757 cu.m. x 2.5 MT/cu.m. = 8,729,392 MT
No. of Shovel needed =	=	$\frac{\text{Total Material}}{\text{Tonnage/shovel}}$
No. of Shovel needed =	=	$\frac{18,478,800 \text{ MT}}{8,729,392 \text{ MT/shovel}}$
No. of Shovel needed =	=	2.12 units
No. of Shovel needed =	=	3 units

A.15 OPEN PIT MINING PROBLEM:

August, 1998. A new open pit mine will be developed under the FTAA of the New Mining Act of 1995. The operating parameters of this mine are defined as follows:

Annual mill capacity	= 10,440,000 MT
Stripping ratio	= 0.77 : 1
Shovel bucket size	= 9.19 cubic meters
Shovel utilization	= 80%
Bucket fill factor	= 90%
Truck spot time	= 0.33 minute
Load cycle time	= 0.50 minute
Time efficiency	= 50 mins per 60 mins

As Mine Engineer of the open pit, how many power shovels will you order for purchase (excluding spare units) in order to sustain the ore requirement of the mill and at the same time maintaining a workable and safe pit geometry by promptly stripping the waste?

- a. 2 units b. 1 unit c. 4 units d. **3 units** e. 5 units

Solution:

Stripping ratio=
$$\frac{\text{Waste Tonnage}}{\text{Ore Tonnage}}$$

0.77 =
$$\frac{\text{Waste Tonnage}}{10,440,000 \text{ MT}}$$

Waste Tonnage = 8,038,800 MT

Total Tonnage = Ore tonnage + Waste tonnage
Total Tonnage = 10,440,000 MT + 8,038,800 MT
Total Tonnage = 18,478,800 MT

Shovel Capacity =
$$\frac{\text{Load Volume}}{\text{Cycle time}}$$

Shovel Capacity =
$$\frac{\text{Shovel Bucket size} \times \text{Bucket Fill factor}}{\text{Truck spot time} + \text{Load cycle time}}$$

Shovel Capacity =
$$\frac{9.19 \text{ m}^3 \times 0.90}{0.33 \text{ min} + 0.50 \text{ min}}$$

Shovel Capacity = 9.965 m³/min

With 80% utilization:
Shovel Capacity = 9.965 m³/min x 0.80
Shovel Capacity = 7.97 m³/min

Assume density = 2.5 tons/m³

Shovel capacity = 7.97 m³/min x 2.5 tons/m³ x 60min/hr x 24 hrs/day x 365 days/yr. x 50 min/60 min.
Shovel Capacity = 8,727,150 MT/yr-unit

No. of unit =
$$\frac{\text{Total Tonnage}}{\text{Shovel capacity}}$$

No. of units =
$$\frac{18,478,800 \text{ MT/yr.}}{8,727,150 \text{ MT/yr-unit}}$$

No. of units = 2.12 units

No. of units = **3 units**

A.16 OPEN PIT MINING PROBLEM:

August, 1984. Coal is produced from two small pits. Production from pit A, a better quality of coal, is sold at a gain of 30% based on production costs while production from pit B which is the same quantity but of poor quality is sold at a loss of 30%. If the selling price is P800/DMT, compute for:

- Production costs at pit A. (Ans=P615/DMT)
- Production costs at pit B. (Ans=P1,143/DMT)
- Loss or gain of mine per DMT coal sold. (Ans=P158/DMT loss)
- Selling price of coal per DMT so that the mine will make a 50% profit. (Ans=P1,318/DMT)

Solution:

$$\begin{aligned}
 \text{a. } \% \text{ Gain} &= \frac{\text{Selling price} - \text{Production cost of Pit A (PC}_A\text{)}}{\text{Production cost of Pit A (PC}_A\text{)}} \\
 0.30 &= \frac{P800 - PC_A}{PC_A} \\
 0.30 PC_A &= P800 - PC_A \\
 PC_A &= \underline{\underline{\mathbf{P615/DMT}}}
 \end{aligned}$$

$$\begin{aligned}
 \text{b. } \% \text{ Loss} &= \frac{\text{Production Cost of B (PC}_B\text{)} - \text{Selling Price}}{\text{Production Cost of B (PC}_B\text{)}} \\
 0.30 &= \frac{PC_B - P800}{PC_B} \\
 0.30 PC_B &= PC_B - P800 \\
 PC_B &= \underline{\underline{\mathbf{P1,143/DMT}}}
 \end{aligned}$$

$$\begin{aligned}
 \text{c. Gain of Pit A} &= \text{Selling Price} - \text{Production Cost of Pit A} \\
 &= P800 - P615 \\
 &= P185/DMT
 \end{aligned}$$

$$\begin{aligned}
 \text{Loss of Pit B} &= \text{Production Cost of Pit B} - \text{Selling Price} \\
 &= P1,143 - P800 \\
 &= P343/DMT
 \end{aligned}$$

$$\begin{aligned}
 \text{Loss of Mine} &= P343 - P185 \\
 \text{Loss of Mine} &= \underline{\underline{\mathbf{P158/DMT}}}
 \end{aligned}$$

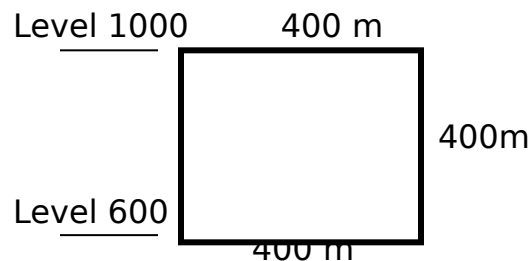
$$\begin{aligned}
 \text{d. } \% \text{ Profit} &= \frac{\text{Profit @ A} + \text{Profit at B}}{\text{Production Cost @ A} + \text{Production Cost @ B}} \\
 0.50 &= \frac{(SP - PC_A) + (SP - PC_B)}{PC_A + PC_B} \\
 0.50 &= \frac{(SP - P615) + SP - P1,143}{P615 + P1,143} \\
 2 SP &= P2,637 \\
 SP &= \underline{\underline{\mathbf{P1,318.50/DMT}}}
 \end{aligned}$$

B.1 MATERIAL DENSITY PROBLEM:ok

August, 2003. A steeply dipping and almost vertical gold vein 3.6 meters thick was followed by a drift at level 1000 meters above sea level to a strike length of 400 meters until it was intercepted by a younger intrusive formation. The same gold vein was intercepted by a crosscut at elevation 600 meters above sea level and at about the same coordinates as at level 1000 meters and followed it to a length of 400 meters until it was cut by the same intrusive. Assuming a consistent thickness of the vein, how much ore reserve is there if the material specific gravity is 2.5?

- A. 1,220,00 MT B. 1,550,000 MT **C. 1,440,000 MT**
D. 1,660,000 MT E. 1,330,000 MT

Solution:



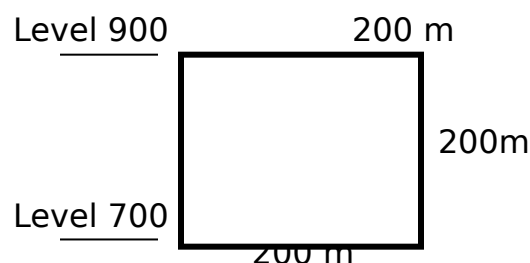
$$\begin{aligned}\text{Ore Reserve} &= 400\text{m} \times 400\text{m} \times 3.6\text{m} \\ \text{Ore Reserve} &= 576,000 \text{ cu.m.} \times 2.5 \text{ MT/cu.m} \\ \text{Ore Reserve} &= \mathbf{1,440,000 \text{ MT}}\end{aligned}$$

B.2 MATERIAL DENSITY PROBLEM:ok

August 2002, 2008. A steeply dipping sub-bituminous coal seam 1.8 meters thick was followed by a drift at level 900 to a strike length of 200 meters until it was lost by an intrusive formation. The same coal seam was intercepted by a crosscut at level 700 meters and followed it to a length of 200 meters until the same intrusive was reached. Assuming a consistent thickness of coal seam, how much coal reserve is there if the material specific gravity is 1.6?

- A. 157,632 MT B. 162,940 MT C. 207,455 MT
D. 101,894 MT E. 314,675 MT

Solution:



$$\begin{aligned}\text{Coal reserve} &= 200\text{m} \times 200\text{m} \times 1.8\text{m} \\ \text{Coal Reserve} &= 72,000 \text{ cu.m.} \times 1.6 \text{ MT/cu.m} \\ \text{Coal Reserve} &= \mathbf{115,200 \text{ MT}}\end{aligned}$$

B.3 MATERIAL DENSITY PROBLEM:ok

August 2003, 2006, 2008. A copper bearing host rock is blasted and found to exhibit a swell factor of 30%. What is the resulting material density in tons per loose cubic meter (LCM) if the specific gravity is 2.5?

- A. 1.92** B. 2.8 C. 42.53 D. 0.75 E. 8.33

Solution:

Volume of broken ore = Volume of ore in place (1 + swell factor)

Volume of broken ore = $\frac{\text{Weight of ore in place}}{\text{Density of ore in place}} \times 1.3$

Volume of Broken ore = $\frac{1 \text{ MT}}{2.5 \text{ MT/m}^3} \times 1.3$

Volume of broken ore = 0.52 m³ (LCM)

Then:

Density of Broken ore = $\frac{\text{Weight of Broken ore (MT)}}{\text{Volume of broken ore (m}^3\text{)}}$

Density of broken ore = $\frac{1 \text{ MT}}{0.52 \text{ m}^3}$

Density of broken ore = **1.92 MT/LCM**

Alternate Solution:

Density of broken ore = $\frac{\text{Density of ore in place}}{(1 + \text{swell factor})}$

Density of broken ore = $\frac{2.5 \text{ MT/BCM}}{1.3 \text{ LCM/BCM}}$

Density of broken ore = **1.92 MT/LCM**

B.4 MATERIAL DENSITY PROBLEM:ok

August, 2002. Dacite, andesite and diorite are common host rocks for disseminated porphyry copper mineralization. Their specific gravity is established at 2.75. Calculate their resulting material density in tons per loose cu.m. (LCM) if they are known to exhibit a swell factor of 33% when blasted.

Solution:

$$\begin{aligned} \text{Density of broken ore} &= \frac{\text{Density of ore in place}}{(1 + \text{swell factor})} \\ \text{Density of broken ore} &= \frac{2.75 \text{ MT/BCM}}{1.33 \text{ LCM/BCM}} \\ \text{Density of broken ore} &= \underline{\underline{\mathbf{2.07 \text{ MT/LCM}}}} \end{aligned}$$

B.5 MATERIAL DENSITY PROBLEM:ok

August, 2001. An ore block has been estimated to contain 45,000 MT @ 55% Fe. If the swell factor is 30%, find the volume that will be occupied by the broken ore. Tonnage factor is 2.8 Mt per m³ for the ore in place.

Solution:

$$\begin{aligned} \text{Volume of Broken Ore} &= \text{Volume of ore in place} + (\text{Volume of ore in place} \times \text{swell factor}) \\ \text{Volume of Broken Ore} &= \text{Volume of ore in place} \times (1 + \text{swell factor}) \\ \text{Volume of Broken Ore} &= \frac{\text{Weight of ore (MT)}}{\text{Tonnage Factor (MT/m}^3)} \times (1 + \text{Swell Factor}) \\ \text{Volume of Broken Ore} &= \frac{45,000 \text{ MT}}{2.8 \text{ MT/m}^3} \times (1 + 0.30) \\ \text{Volume of Broken Ore} &= \underline{\underline{\mathbf{20,893 \text{ cubic meters}}}}} \end{aligned}$$

C.1 GRADE CONTROL PROBLEM:ok

August, 2003. Heritage Resources and Mining Corporation, the country's premier producer of chemical grade chromite concentrates has the following chromite stockpiles ready for export to China:

Cambayas stockpile	-	38% Cr ₂ O ₃ , 1.1% SiO ₂
Rio Chico stockpile	-	48% Cr ₂ O ₃ , 2.0% SiO ₂

How much ore will you get from each stockpile to produce a blended export shipment of 3,000 MT assaying 46% Cr₂O₃ and 1.82 SiO₂?

- A. Cambayas - 800 MT; Rio Chico - 2,200 MT
- B. Cambayas - 700 MT; Rio Chico - 2,300 MT
- C. Cambayas - 600 MT; Rio Chico - 2,400 MT**
- D. Cambayas - 500 MT; Rio Chico - 2,500 MT
- E. Cambayas - 400 MT; Rio Chico - 2,600 MT

Solution:

Let: C = tonnage at Cambayas stockpile
R = tonnage at Rio Chico stockpile

By material balance:

$$\begin{aligned} C + R &= 3000 \\ R &= 3000 - C \text{ ----- Eq. 1} \end{aligned}$$

By mass balance:

$$\begin{aligned} (C \times 38) + (R \times 48) &= 3000 \times 46 \\ 38 C + 48 R &= 138,000 \text{ ----- Eq. 2} \end{aligned}$$

Substitute Eq. 2 in Eq. 1:

$$\begin{aligned} 38 C + ((48 \times (3000 - C)) &= 138,000 \\ 38 C + 144,000 - 48 C &= 138,000 \\ 10 C &= 6,000 \end{aligned}$$

$$C = 600 \text{ MT (Cambayas)}$$

$$R = 2,400 \text{ MT (Rio Chico)}$$

C.2 GRADE CONTROL PROBLEM:ok

August, 1986. As a grade control engineer of a gold mine, you are required to maintain a daily mill feed of 3,000 metric tones of gold ore assaying 5 grams Au per MT. If 1,200

metric tonnes of ore with a grade of 3.5 grams Au per MT are already available, how many more tonnes of ore will you need for blending from stopes A and B with corresponding grades of 4.5 gms Au/MT and 6.5 gms Au/MT respectively.

Solution:

	Tonnage	Grade
A	-	4.5
B	-	6.5
C	1,200	3.5
	-----	-----
Total	3,000	5.0

By Material Balance:

$$\begin{aligned}
 A + B + 1,200 &= 3,000 \\
 A + B &= 1,800 \\
 A &= 1,800 - B \text{ (Equation 1)}
 \end{aligned}$$

But:

$$\begin{aligned}
 4.5A + 6.5B + 1,200 (3.5) &= 3,000 (5.0) \\
 4.5A + 6.5B + 4,200 &= 15,000 \\
 4.5A + 6.5B &= 10,800 \text{ (Equation 2)}
 \end{aligned}$$

Substitute Eq. 1 in Eq. 2:

$$\begin{aligned}
 4.5 (1,800-B) + 6.5B &= 10,800 \\
 8,100 -4.5B + 6.5B &= 10,800 \\
 2B &= 2,700 \\
 B &= \textbf{1,350 MT}
 \end{aligned}$$

And:

$$\begin{aligned}
 A &= 1,800 - 1,350 \\
 A &= \textbf{450 MT}
 \end{aligned}$$

C.3 GRADE CONTROL PROBLEM:review

AUGUST 2001. Heritage Resources and Mining Corporation, the country’s premier producer of chemical grade chromite concentrates has the following production data from its mining operations in Eastern Samar:

<u>Mine</u>	<u>Ore Grade</u>	<u>Cr:Fe ratio</u>
Llorente Mine	38% Cr ₂ O ₃	1.1 : 1
Homonhon Mine	48% Cr ₂ O ₃	2.0 : 1

Being the Grade Control Engineer, how much ore will you program for extraction from each mine to produce a blended export shipment of 3,000 MT assaying 46% Cr₂O₃ and 1.82 : 1 Cr : Fe ratio.

- Llorente - 400 MT, Homonhon - 2,600 MT
- Llorente - 800 MT, Homonhon - 2,200 MT
- Llorente - 700 MT, Homonhon - 2,300 MT
- Llorente - 600 MT, Homonhon - 2,400 MT**
- Llorente - 500 MT, Homonhon - 2,500 MT

Solution:

$$L + H = 3000 \quad (\text{Equation 1})$$

$$\begin{aligned} 0.38L + 0.48H &= 0.46 \times 3000 \\ 0.38L + 0.48H &= 1380 \quad (\text{Equation 2}) \end{aligned}$$

$$\begin{aligned} 1.1L + 2.0H &= 1.82 \times 3000 \\ 1.1L + 2.0H &= 5460 \end{aligned}$$

$$H = \frac{5460 - 1.1L}{2}$$

Substitute H in Equation 2:

$$\begin{aligned} 0.38L + 0.48 \times \frac{5460 - 1.1L}{2} &= 1380 \\ 0.38L + 1310.4 - 0.264L &= 1380 \\ 0.116L &= 69.6 \\ \mathbf{L} &= \mathbf{600 \text{ MT}} \end{aligned}$$

And,

$$\begin{aligned} H &= 3000 - 600 \\ \mathbf{H} &= \mathbf{2,400 \text{ MT}} \end{aligned}$$

C.4 GRADE CONTROL PROBLEM:

AUGUST 2002. A manganese mining operator has stockpiled 1,000 MT @ 52% Mn ore. He operates mine A from which he obtains 45% Mn ore and mine B from which he get 35% Mn. He intends to mine 2 MT from mine B for every MT from mine A. If shipping requirement is 43% Mn, how many MT can he ship using all the stockpile?

Solution:

$$\text{Stockpile} + \text{Mine A} + \text{Mine B} = \text{Total shipment}$$

$$(1,000 \text{ MT} \times 0.52) + (\text{Mine A} \times 0.45) + (\text{Mine B} \times 0.35) = (1,000 \text{ MT} + \text{Mine A} + \text{Mine B}) \times 0.43$$

$$520 + 0.45 \text{ Mine A} + 0.35 \text{ Mine B} = 430 + 0.43 \text{ Mine A} + 0.43 \text{ Mine B}$$

$$520 + 0.45 \text{ Mine A} + 0.35 (2 \times \text{Mine A}) = 430 + 0.43 \text{ Mine A} + 0.43 (2 \times \text{Mine A})$$

$$520 + 0.45 \text{ Mine A} + 0.70 \text{ Mine A} = 430 + 0.43 \text{ Mine A} + 0.86 \text{ Mine A}$$

$$520 + 1.15 \text{ Mine A} = 430 + 1.29 \text{ Mine A}$$

$$0.14 \text{ Mine A} = 90$$

$$\text{Mine A} = 642.86 \text{ MT}$$

Then:

$$\text{Mine B} = 2 \times 642.86$$

$$\text{Mine B} = 1,285.72 \text{ MT}$$

Therefore:

$$\text{Total Shipment} = 1,000 \text{ MT} + 642.86 \text{ MT} + 1,285.72 \text{ MT}$$

$$\text{Total Shipment} = \underline{\underline{2,928.58 \text{ MT}}}$$

C.5 GRADE CONTROL PROBLEM:review

AUGUST 2000. A coal mine is mining from two coal seams A and seam B. Typical characteristics of coal product are shown below. A sales note received requiring a 'blend' of seam A and seam B:

Coal	% Ash	% Sulphur	Btu/lb	Cost, US\$/MT
Seam A	10	0.5	13,000	40
Seam B	15	1.0	11,000	30
Blend	12 max	1.5	12,000	Minimum

What proportion of seam A and seam B coal will meet the specifications for the “blend” coal at minimum cost?

Solution:

Using % Ash:

Seam B)

(% Ash in A x Seam A) + (% Ash in B x Seam B) = % Ash in Blend x (Seam A + Seam B)

$$10A + 15B = 12(A + B)$$

$$10A + 15B = 12A + 12B$$

$$2A = 3B$$

$$\frac{A}{B} = \frac{3}{2}$$

$$\frac{A}{B} = \frac{1.5}{1}, \text{ the proportion is } A=1.5 \text{ and } B=1$$

Using BTU/lb:

$$13,000A + 11,000B = 12,000(A + B)$$

$$13,000A + 11,000B = 12,000A + 12,000B$$

$$1,000A = 1,000B$$

$$\frac{A}{B} = \frac{1}{1}, \text{ the proportion is } A=1 \text{ and } B=1$$

Using proportion A=1.5 and B=1:

% Ash in Blend

$$= \frac{10A + 15B}{A + B}$$

$$= \frac{(10 \times 1.5) + (15 \times 1)}{2.5}$$

$$= 12 \text{ \% (passed the blend specs)}$$

BTU/lb in Blend

$$= \frac{13,000A + 11,000B}{A + B}$$

$$= \frac{(13,000 \times 1.5) + (11,000 \times 1)}{2.5}$$

$$= 12,200 \text{ BTU (passed the blend specs)}$$

Using proportion A=1 and B=1:

% Ash in Blend

$$= \frac{10A + 15B}{A + B}$$

$$= \frac{(10 \times 1) + (15 \times 1)}{2}$$

$$= 12.5 \% \text{ (failed the blend specs)}$$

$$\text{BTU/lb in Blend} = \frac{13,000A + 11,000B}{A + B}$$

$$= \frac{(13,000 \times 1) + (11,000 \times 1)}{2}$$

$$= 12,000 \text{ BTU (passed the blend specs)}$$

Therefore, the required proportion is:

$$\begin{aligned} \text{Seam A} &= 1.5 \\ \text{Seam B} &= 1.0 \end{aligned}$$

C.6 GRADE CONTROL PROBLEM:review

AUGUST 2003. Assuming that a 1000 megawatt power plant needs 10,000 dry short tons (DST) of coal with a heating value of 10,000 BTU/lb on dry basis and your mine can only produce 5,000 DST of coal at a heating value o 8,000 BTU/lb on dry basis, how much imported coal will you have to blend with your own coal if imported coal has a heating value of 12,000 BTU/lb on dry basis?
 a. 4000 DST b. **5000 DST** c. 6000 DST d. 3000 DST

Solution:

Brand	Tonnage	Grade
Local Coal	5,000 DST	8,000 BTU/lb
Imported Coal	M	12,000 BTU/lb
	-----	-----
Total	10,000 DST	10,000 BTU/lb

$$\begin{aligned} (5,000 \times 8,000) + (M \times 12,000) &= 10,000 \times 10,000 \\ 40,000,000 + 12,000 M &= 100,000,000 \\ 12,000 M &= 60,000,000 \\ M &= \mathbf{5,000 \text{ DST}} \end{aligned}$$

C.7 GRADE CONTROL PROBLEM:

August, 1987. A tailings dump from mining operations is suspected to be rich in gold. It is divided in a 100-meter grid. If the dump measures 300m x 300m, the values are shown in the figures. Determine the average grade of gold content. (Ans=3.274 gm Au/MT)

	6.0/2.0	5.8/5.9	5.3/3.1	6.1/2.8
5.5/3.6	4.8/3.1	5.4/2.7	5.9/3.1	
4.8/3.2	3.8/2.8	4.6/2.2	5.7/3.6	
5.3/4.5	4.7/4.5	5.1/3.9	5.6/2.5	

DAC 2012 Mining Engineering 1 Review

26

100m

100m

100m

100m 100m 100m

300m

Tonnage Factor: 1 cubic meter = 2.0 tons
6.0/2.0 = Depth in meter/grams per ton Au

Solution:

Test Pit No.	Depth, m	Value, gm/MT	Area, m2	Volume, m3	Tonnage, MT	Tonnage x Value (MT x gm/MT)
1	6.0	2.0	50 x 50	15,000	30,000	60,000
2	5.8	5.9	50 x 100	29,000	58,000	342,200
3	5.3	3.1	50 x 100	26,500	53,000	164,300
4	6.1	2.8	50 x 50	15,250	30,500	85,400
5	5.5	3.6	100 x 50	27,500	55,000	198,000
6	4.8	3.1	100 x 100	48,000	96,000	297,600
7	5.4	2.7	100 x 100	54,000	108,000	291,600
8	5.9	3.1	100 x 50	29,500	59,000	182,900
9	4.8	3.2	100 x 50	24,000	48,000	153,600
10	3.8	2.8	100 x 100	38,000	76,000	212,800
11	4.6	2.2	100 x 100	46,000	92,000	202,400
12	5.7	3.6	100 x 50	28,500	57,000	205,200
13	5.3	4.5	50 x 50	13,250	26,500	119,200
14	4.7	4.5	50 x 100	23,500	47,000	211,500
15	5.1	3.9	50 x 100	25,500	51,000	198,900
16	5.6	2.5	50 x 50	14,000	28,000	70,000
TOTAL					915,000	2,995,650

Therefore;

Average Value=

Tonnage x Value

Tonnage

Average Value=

2,995,650 MT x gm/MT

915,000 MT

Average Value=

3.274 gm/MT

C.8 GRADE CONTROL PROBLEM:ok

August, 2004, 2008. A high grade gold vein was sampled by channel sampling and showed the following assays:

Sample No.	Width (cm)	Assay (gmt)
1	15	3.0

2	12	3.5
3	10	5.1
4	12	5.5
5	15	6.0
6	15	6.3
7	18	7.3
8	17	8.8
9	19	8.5
10	18	9.0

Assuming that the samples were taken at regular intervals, what is the average grade of the gold vein?

A. 7.58 gmt B. 5.58 gmt C. 8.58 gmt **D. 6.58 gmt** E. 4.58 gmt

Solution:

Sample No.	Width (cm)	Assay (gmt)	Width x Assay
1	15	3.0	45.0
2	12	3.5	42.0
3	10	5.1	51.0
4	12	5.5	66.0
5	15	6.0	90.0
6	15	6.3	94.5
7	18	7.3	131.4
8	17	8.8	149.6
9	19	8.5	161.5
10	18	9.0	162.0
	-----		-----
Total	151		993

$$\text{Average Grade} = \frac{\text{Total (Width x Assay)}}{\text{Total Width}}$$

$$\text{Average Grade} = \frac{993}{151}$$

$$\text{Average Grade} = \underline{\underline{\mathbf{6.58\ gmt}}}$$

C.9 GRADE CONTROLK PROBLEM:review

AUGUST 2000. In an open pit gold mining operation, a grade control engineer established 3 ore blocks that can be mined the following day. Ore block A has 1,500 MT ore @ 1.03 gm Au/MT, ore block B has 4,000 MT ore @ 2.16 gm Au/MT and ore block C has 3,200 MT ore @ 1.75 gm Au/MT.

- What is the total tonnage of the ore blocks?
 - 5,800 MT
 - 8,700 MT**
 - 6,600 MT
 - 7,700 MT
- What is the weighted average grade?
 - 1.81 gm /MT**
 - 1.65 gm/MT
 - 1.67 gm/MT
 - 2.01 gm/MT

- b. What is the diluted grade if dilution is 10%?
 a. 3.01 gm/MT b. 2.71 gm/MT c. 0.75 gm/MT d. **1.63 gm/MT**
- c. How many oz./MT if recovery is 80%?
 a. 456 oz. b. **365 oz.** c. 286 oz. d. 430 oz.

Solution:

	<u>Ore Block</u>	<u>Tonnage</u>	<u>Grade</u>	<u>Tonnage x Grade</u>
	A	1,500 Mt	1.03 gm Au/MT	1,545 gm Au
	B	4,000 Mt	2.16 gm Au/MT	8,640 gm Au
	C	3,200 MT	1.75 gm Au/MT	5,600 gm Au
		-----	-----	-----
	Total	8,700 MT	1.81 gm Au/MT	15,785 gm Au
1. Total Tonnage of ore blocks	= 8,700 MT			
2. Weighted Average Grade =	$\frac{\text{Total (Tonnage x Grade)}}{\text{Total Tonnage}}$			
	$\frac{15,785 \text{ gm Au}}{8,700 \text{ MT}}$			
Weighted Average Grade =	1.81 gm Au/MT			
3. Diluted Grade =	1.81 gm Au/Mt x 0.90			
Diluted Grade =	1.63 gm Au/MT			
4. Recovery =	$\frac{\text{Wt. of metal in concentrate}}{\text{Wt. of metal in ore}}$			
	$\frac{\text{Wt. of metal in concentrate}}{\text{Wt. of metal in concentrate}}$			
gms x 0.80	= 8700 MT x 1.63 gm Au/MT x 1 oz./31.1035			
	= 365 oz.			

D.1 UNDERGROUND MINING PROBLEM:

August 2001:

If you are a Draw Control Engineer in a Block Caving operation and you are required to have at least a subsidence of 1 meter of the caved ore from each draw raise, compute for the amount of draw you should get from each finger raise (each with a diameter of 3 meters) considering that the in-situ density of the ore is 2 MT/cu.m., with a swell factor of 17% upon caving.

- b. 14.10 MT/m b. 19.47 MT/m c. **12.10 MT/m** d.
15.22 MT/m

Solution:

$$\text{Volume of ore} = \pi/4 \times (3\text{m})^2 \times 1\text{m}$$

$$\text{Volume of ore} = 7.07 \text{ lcm}$$

$$\text{Swell Factor} = \frac{\text{LCM} - \text{BCM}}{\text{BCM}}$$

$$\text{LCM} - \text{BCM} = \text{BCM} \times \text{swell factor}$$

$$\text{LCM} = \text{BCM} + (\text{BCM} \times \text{swell factor})$$

$$\text{LCM} = \text{BCM} \times (1 + \text{swell Factor})$$

$$7.07 = \text{BCM} \times 1.17$$

$$\text{BCM} = 6.043 \text{ cu.m.}$$

Therefore:

$$\text{Amount of draw} = 6.043 \text{ cu.m.} \times 2.0 \text{ MT/cu.m.}$$

$$\text{Amount of draw} = \mathbf{\underline{12.10 \text{ MT/m}}}$$

D.2 UNDERGROUND MINING PROBLEM:ok

August 1998, 2002, 2003, 2004, 2008. A hardwood timber is used as post to support an average load of 20,000 pounds, 5 feet high. If the timber has a compressive stress of 900 psi, what would be the typical size of timber support most economical to be used for the purpose?

- A. 10 x 10 inches B. 6 x 6 inches C. 12 x 14 inches
D. 8 x 6 inches **E. 4 x 6 inches**

Solution:

$$\begin{aligned} \text{Compressive Strength} &= \frac{\text{Load}}{\text{Surface Area}} \\ \text{Surface Area} &= \frac{20,000 \text{ lbs}}{900 \text{ lbs / in.}^2} \\ \text{Surface Area} &= 22.22 \text{ in.}^2 \end{aligned}$$

Therefore, use 4" x 6" timber

D.3 UNDERGROUND MINING PROBLEM:ok

August, 2005. As a Mine Foreman, you have been assigned to support an underground heading that is experiencing heavy loose grounds. You were supplied with hardwood timber of various sizes to support an average load of 20,000 pounds, 1.52 meters high. If the timber has a compressive strength of about 61 bars, what approximate size of timber would you use as posts for the purpose?

- A. 30 x 36 cm B. 25 x 25 cm C. 15 x 15 cm D. 15 x 20 cm
E. 10 x 15 cm

Solution:

$$\begin{aligned} \text{Compressive Strength} &= \frac{\text{Load}}{\text{Surface Area}} \\ \text{Surface Area} &= \frac{20,000 \text{ lbs}}{61 \text{ bars} \times 14.7 \text{ lbs / in.}^2/\text{bar}} \\ \text{Surface Area} &= 22.304 \text{ in.}^2 \times (2.54 \text{ cm/in})^2 \\ \text{Surface Area} &= 144 \text{ cm}^2 \end{aligned}$$

Therefore, use 10 x 15 cm timber

D.4 UNDERGROUND MINING PROBLEM:

August, 1982. An 8' x 8' x 30' haulage way in an underground mine requires an artificial support system either by conventional timber support or rockbolting. Given the following data, which method would you recommend as the cheapest. Support your answer with computations.

- Data for conventional support:
- Distance between sets = 5 ft
 - Size of post and caps = 8" x 8" x 8'
 - Size of side laggings(4 pcs/set) = 2" x 6" x 4'
 - Size of top laggings (2 pcs/set) = 2" x 6" x 4'
 - Size of braces (4 pcs/set) = 2" x 6" x 8'
 - Cost per bd. Ft. of timber = P2.30
 - Two miners at P25.00 each per day can install the timber supports for 6 days. Miscellaneous cost is P100.00
- Data for rockbolting:

Rockbolts (expansion type) = 40 pcs @ P50.00 each
 Two miners at P25.00 each per day can install these bolts in 2 days.
 Two drillers at P25.00 each per day can drill 10 holes per day.

Solution:

A. For Timber Supports:

	No. of sets	=	$\frac{30\text{ ft}}{5\text{ft/set}}$		
		=	6 sets + 1 set		
		=	7 sets		
	Solving for total board foot:				
bd.ft.	Post	=	2pcs/set x 7 sets x (8" x 8" x 8')/12 bd.ft./pc	-----	597.33
	Cap	=	1pc/set x 7 sets x (8" x 8" x 8')/12 bd.ft./pc	-----	
298.67 bd.ft.	Side laggings	=	4 pcs/set x 7 sets x (2" x 6" x 4')/12 bd.ft./pc	-----	112.00
bd.ft.	Top laggings	=	2 pcs/set x 7 sets x (2" x 6" x 4')/12 bd.ft./pc	-----	56.00
bd.ft.	Braces	=	4 pcs/set x 7 sets x (2" x 6" x 8')/12 bd.ft./pc	-----	224.00
bd.ft.					
			TOTAL	-----	1,288 bd.ft.

	Solving for Total Cost:				
	Timber Cost	=	1,288 bd.ft. x P2.30/bd.ft.		
			P2,962.40		
300.00	Labor Cost	=	2 miners x P25.00/miner-day x 6 days	-----	P
	Miscellaneous	=		-----	P 100.00
			Total Cost for Timber Supports	-----	
P3,362.40					

B. For Rockbolting:

	Rockbolt Cost	=	40 pcs x P50.00/pc.	-----	
P2,000.00					
100.00	Labor Cost	=	2 miners x P25.00/miner-day x 2 days	-----	P
200.00	Drilling Cost	=	2 drillers x P25.00/driller-day x 40 holes x 1 day/10 holes--		P
			Total Cost for Rockbolting	-----	P2,300.00

Therefore, Rockbolting is recommended.

D.5 UNDERGROUND MINING PROBLEM:ok

AUGUST 2000. Determine the daily and yearly production rate in a room and pillar coal mining using conventional equipment under the following conditions:

Working place	= 6 x 18 ft
Working Time	= 7 hrs/shift, 2 shift/day, 250
days/year	
Working sections	= 14
Advance per cut	= 10 ft
Cuts per shift	= 12
Tonnage factor (tf)	= 24 ft ³ /ton

Solution:

	Production	=	$\frac{\text{Volume, ft}^3}{\text{Tonnage Factor, ft}^3/\text{ton}}$
cut	Volume	=	Working place area x advance per
	Volume	=	6 ft x 18 ft x 10 ft
shift/day	Volume	=	1,080 ft ³ /cut x 12 cuts/shift x 2
	Volume	=	25,920 ft ³ /day
	Production	=	$\frac{25,920 \text{ ft}^3/\text{day}}{24 \text{ ft}^3/\text{ton}}$
	Production	=	1,080 tons/day x 14 sections
	Production/day	=	15,120 tons/day
	Production/yr.	=	15,120 tons/day x 250 days/yr.
	Production/yr.	=	3,780,000 tons/yr.

MINING ENGINEERING 1 PUMPING PROBLEM

E.1 PUMPING PROBLEM

August 2002, 2004, 2008. A mine shaft which originally contains 400,000 cu.m. of water receives a daily influx of 5,000 cu.m. The shaft must be dewatered in 60 days by pumping 20 hours per day. Being the Mine Superintendent, how many pumps will you install if the capacity of each pump is 1,000 gallons per minute and running at an efficiency of 60%? Allow additional stand-by pumps of two (2) units. Assume that there are 264 gallons per cu.m.

A. 8 units **B. 7 units** C. 3 units D. 6 units E. 9 units

Solution:

$$\text{Influx} = 5,000 \text{ cu.m./day} \times 60 \text{ days} = 300,000 \text{ cu.m.}$$

$$\text{Total water} = 400,000 \text{ cu.m.} + 300,000 \text{ cu.m.} = 700,000 \text{ cu.m.}$$

$$\text{Rated Pump Capacity} = 1,000 \text{ gallons / min.} \times 1 \text{ cu.m./264 gal} \times 60 \text{ min/hr}$$

$$= 227.27 \text{ cu.m. /hr}$$

$$\text{Actual Pump Capacity} = 227.27 \text{ cu.m. / hr} \times 0.60$$

$$= 136.36 \text{ cu. M. /hr}$$

$$\text{No. of Pumps needed} = \frac{\text{Total Water}}{\text{Actual Pump Capacity}}$$

$$= \frac{700,000 \text{ cu.m}}{136.36 \text{ cu. M. /hr/pump} \times 20 \text{ hrs/day} \times 60 \text{ days}}$$

$$= 4.28 \text{ pumps}$$

$$= 5 \text{ pumps}$$

Therefore:

$$\text{The total number of pumps needed standby} = 5 \text{ pumps} + 2 \text{ pumps}$$

$$= 7 \text{ units}$$

E.2 PUMPING PROBLEM:

AUGUST 2001. An amount of 10,000 gallons of water has to be pumped for 2 hours. Total head is 50 feet and pump efficiency is 80%. What should be the horsepower rating of the pump? If the motor has efficiency of 70%, what is the motor horsepower?

Solution:

$$\text{Pump Hp} = \frac{wQH}{550 \times e_p}$$

where:

w

= specific weight of liquid = 62.4 lb/ft³ for water

Q

= capacity in ft³/sec

= 10,000 gallons/2hrs x 1hr/60min

= 83.33 gallons per minute x 1min/60sec x 1 ft³/7.481 gal.

H

= head in ft. = 50 ft

Ep

= pump efficiency = 80%

550

= constant, ft-lb/sec-hp

$$\text{Pump Hp} = \frac{62.4 \text{ lb/ft}^3 \times 0.1856 \text{ ft}^3/\text{sec} \times 50 \text{ ft}}{550 \text{ ft-lb/sec-hp} \times 0.80}$$

$$\text{Pump Hp} = 1.32 \text{ hp or } 1.5 \text{ hp}$$

$$\text{Motor Hp} = \frac{\text{Pump Hp}}{\text{Motor efficiency}}$$

$$\text{Motor Hp} = \frac{1.32 \text{ hp}}{0.70}$$

$$\text{Motor Hp} = 1.88 \text{ hp or } 2.0 \text{ hp}$$

E.3 PUMPING PROBLEM:

August, 1983. It is necessary to dewater a mine whose water inflow is 4000 liters per minute and the head is 100 meters. If pump and motor efficiency are 80% and 90% respectively, What is the required horsepower of the motor? (Ans=125 hp)

Solution:

$$\text{Motor Hp} = \frac{wQH}{550 e_p e_m}$$

$$\text{Motor Hp} = \frac{62.4 \text{ lb/ft}^3 \times 4000 \text{ li/min} \times 1 \text{ ft}^3/28.35 \text{ li} \times 100\text{m} \times 3.28 \text{ ft/m}}{550 \text{ ft-lb/sec-hp} \times 60 \text{ sec/min} \times 0.80 \times 0.90}$$

$$\text{Motor Hp} = 121.54 \text{ hp or } \underline{125 \text{ hp}}$$

E.4 PUMPING PROBLEM:

AUGUST 2003. A sump is to be drained by 3 pumps. A, B and C. Pump A alone can drain the sump in 12 hours. Pump B in 9 hours and Pump C in 7 hrs. If the pumps are operated at the same time, how long will it take to drain the sump?

Solution:

Capacity, Q

=

Volume, V

Time, T

Q_A

=

V

12

, capacity of Pump A

Q_B

=

V

9

, capacity of Pump B

Q_C

=

V

7

, capacity of Pump C

Q_T

=

V

T

, capacity of combined Pump A, B, C

Q_T

=

Q_A + Q_B + Q_C

V

t

=

V

12

+

V

9

+

V

7

V

t

=

63V + 84V + 108V

756

V

t

=

255V

756

t

=

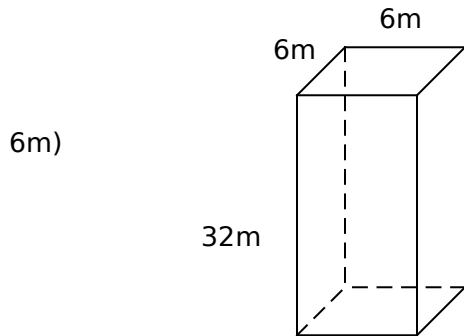
2.96 hrs.

E.5 PUMPING PROBLEM:

August, 1980. A pumping system is to be designed for a shaft 32 meters deep and with a cross section of 6 meters by 6 meters. Water enters the shaft by ground seepage with a maximum rate of 20 gallons per hour per square foot of the shaft’s surface and also by rainfall at the maximum rate of 6 inches per hour. What maximum capacity must the system have in gallons per minute.

Answer: 2,907.48 gpm

Solution:



$$\text{Shaft Surface Area} = 4 (6\text{m} \times 32\text{m}) + 1 (6\text{m} \times 6\text{m})$$

$$\text{Shaft Surface Area} = 804 \text{ sq. m.} \times (3.28 \text{ ft/m})^2$$

$$\text{Shaft Surface Area} = 8,650 \text{ sq. ft.}$$

Solving for Maximum capacity:

By Ground Seepage:

$$Q = 20 \text{ gal/hr-ft}^2 \times 8,650 \text{ ft}^2 \times 1 \text{ hr/60 min.}$$

$$Q = 2,883.33 \text{ gpm}$$

By Rainfall:

$$Q = 6 \text{ in./hr} \times 1\text{m}/39.36 \text{ in.} \times (6\text{m} \times 6\text{m}) \times 1 \text{ hr/60min} \times 264 \text{ gal/m}^3$$

$$Q = 24.15 \text{ gpm}$$

Then;

$$QT = 2883.33 \text{ gpm} + 24.15 \text{ gpm}$$

$$QT = \mathbf{2,907.48 \text{ gpm}}$$

E.6 PUMPING PROBLEM:

August, 1987. The bottom of an open pit excavation is well below the water table. The total inflow is allowed to accumulate at the bottom of the pit where a sump with a capacity of 57,600 cubic meters is located. The inflow is estimated at 2.5 cubic meters per second. There are approximately 220 gallons in one cubic meter. If the water is to be pumped for 18 hours each day, what will be the pump capacity in gallons per minute so that the pump will just be at full capacity at the start of each pumping per day? Assume 90% pump efficiency.

Solution:

Since pumping operation is 18 hours each day, then the amount of water accumulated in the sump without pumping operation for 6 hours is:

$$V = Qt$$

$$V = 2.5 \text{ m}^3/\text{sec} \times 6 \text{ hrs} \times 1\text{hr}/3600 \text{ sec} \times 220 \text{ gal/m}^3$$

$$V = 11,880,000 \text{ gallons}$$

For 18 hours of pumping operation:

Vol.to be pumped-out for 18 hours = Vol. of water w/o pumping + Volume of water for 18 hrs.

$$Q_p (\text{gal/min}) \times 18 \text{ hrs} \times 60 \text{ min/hr} \times 0.90 = (2.5 \text{ m}^3/\text{sec} \times 18 \text{ hrs} \times 1 \text{ hr}/3600 \text{ sec} \times 220 \text{ gal/m}^3)$$

$$\begin{aligned} 972 Q_p &= 47,520,000 \\ Q_p &= \mathbf{48,888.88 \text{ gpm}} \end{aligned}$$

E.7 PUMPING PROBLEM:

August, 1982. The service station of a mining operation is equipped with a diesel driven pump which can fill the tank of an off-highway truck in 20 minutes. An auxiliary electric driven pump is also available which can fill the tank in 30 minutes. Cost for running the

diesel driven pump is P45.00/hr. Cost for running the electric driven pump is P25.00/hr. If you have to service 100 trucks and labor costs are P120.00/day and P100.00 to electrify the diesel driven pump, which of the following would you recommend?

- c. Run the diesel driven pump
- d. Run the electric driven pump**
- e. Run both diesel and electric driven pump at the same time

Solution:

A. Run the Diesel Driven Pump:

1. Running Cost	=	P45/hr x 20min/truck x 1hr/60min x 100 trucks	-----
P1,500.00			
2. Labor Cost	=	P120/day x 20min/truck x 1hr/60min x 1day/24hrs x 100 trucks	--- P
166.67			
3. Electrification			----- P 100.00

		Total Diesel Driven Pump Cost	-----
P1,766.67			

B. Run the Electric Driven Pump:

1. Running Cost	=	P25/hr x 30min/truck x 1hr/60min x 100 trucks	-----
P1,250.00			
2. Labor Cost	=	P120/day x 30min/truck x 1hr/60min x 1day/24hrs x 100 trucks	-- P
250.00			

		Total Electric Driven Pump Cost	-----
P1,500.00			

C. Run both the Diesel and Electric Driven Pump:

Let: X = running hours
V = tank volume

$$\begin{array}{rcl} \frac{V}{20} + \frac{V}{30} & = & \frac{V}{X} \\ 3V + 2V & = & V \\ \hline 60 & & X \\ 5VX & = & 60V \\ X & = & 12 \text{ min.} \end{array}$$

1. Running Cost	=	P70/hr x 12min/truck x 1hr/60min x 100 trucks	-----
P1,400.00			
2. Labor Cost	=	P240/day x 12min/truck x 1hr/60min x 1day/24hrs x 100 trucks	-- P
200.00			
3. Electrification			----- P
100.00			

		Total Cost of running both pumps	-----
P1,700.00			

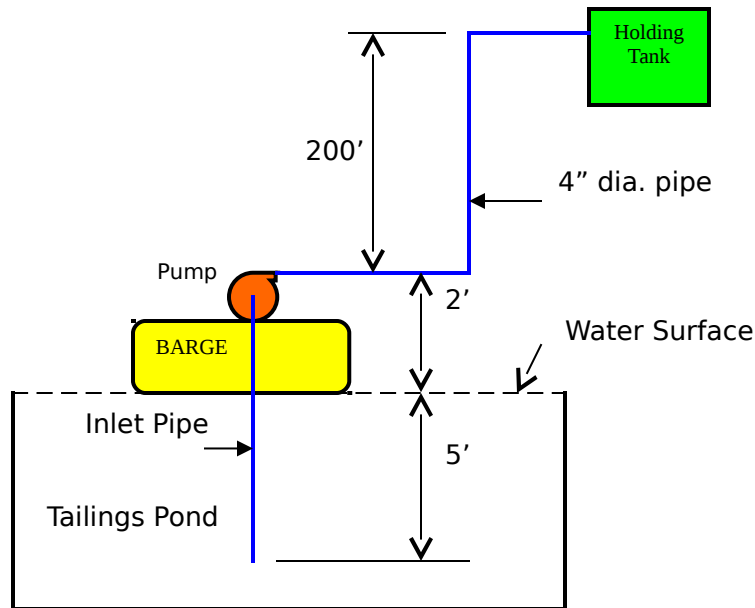
Therefore:
I will recommend to use the electric driven pump.

E.8 PUMPING PROBLEM:

August, 1982. A copper concentrator is reclaiming water from a tailings pond. The water has to be pumped through a 4" pipe to a holding tank which level is kept 200 ft. above the pump discharge. The pump is located on a barge and the inlet is 2 ft. above the water surface. The intake of the system is 5 ft. below the water surface. Friction loss is 150 ft./lb. of water through the 5000 ft of pipe which includes the total equivalent of all piping and fitting from the tailings pond to the holding tank.

1. What would be the net horsepower requirement for the water pump and its capacity if the volume rate of flow desired is at 0.25 ft³/sec.
(Ans=30 hp)
2. With a pump efficiency of 85%, what is the hourly pumping cost, if cost of electricity is P0.80/kw-hr. (Ans=P20.90/hr)

Solution:



$$\begin{aligned} \text{Total Head} &= \overset{0}{\text{Pressure head}} + \overset{0}{\text{Velocity Head}} + \text{Elevation head} + \text{Friction head} \\ \text{Total Head} &= \text{Elevation Head} + \text{Friction Head} \\ \text{Total Head} &= 207 \text{ ft} + (150 \text{ ft/lb} \times 62.4 \text{ lb/ft}^3 \times ((\pi/4 \times (4/12)^2 \text{ ft}^2 \times 5000 \text{ ft}))/5000 \end{aligned}$$

ft

$$\text{Total Head} = 1,024 \text{ ft.}$$

a.

$$\begin{aligned} H_p &= \frac{wQH}{550} \\ H_p &= \frac{62.4 \text{ lb/ft}^3 \times 0.25 \text{ ft}^3/\text{sec} \times 1,024 \text{ ft}}{550 \text{ ft-lb/sec-hp}} \\ H_p &= 29.04 \text{ or } \mathbf{30 \text{ hp}} \end{aligned}$$

b. with 85% pump efficiency

$$\begin{aligned} H_p &= 30/0.85 \\ H_p &= 35 \text{ hp} \end{aligned}$$

Then;

$$\begin{aligned} \text{Hourly Pumping Cost} &= 35 \text{ hp} \times 1 \text{ hr} \times P0.80/\text{kw-hr} \times 0.746 \text{ kw/hp} \\ \text{Hourly pumping Cost} &= \mathbf{P 20.90} \end{aligned}$$

E.9 PUMPING PROBLEM:

August, 1985. A watery section of an underground mine is being dewatered by two units of electrically operated pumps, each having a pumping capacity of 60 m³/hr operating 20 hours per day. Total water inflow to the watery section was measured at 100 m³/hr. A power interruption stopped pumping operation for 60 hours. As normal operation must be resumed not later than 72 hours, it was decided that a third pump will be hired to supplement the two units in use. Each of the three pumps will be operating 22 hrs/day.

- What should be the effective pumping capacity of the third pump in m³/min.? (Ans=1.33 m³/min.)
- What would be the HP rating of the motor if it will be working against a total head of 300 meters and an efficiency of 80%? (Ans=110 hp)
- What will be the pumping cost if cost of electricity is P1.33/kw-hr.? (Ans=P17,942.00)

Solution:

- Capacity of third pump in m³/min.

For 60 hours power interruption:

$$\begin{aligned}\text{Volume of water in the sump} &= \text{Inflow} \times 60 \text{ hours} \\ \text{Volume of water in the sump} &= 100 \text{ m}^3/\text{hr} \times 60 \text{ hrs} \\ \text{Volume of water in the sump} &= 6,000 \text{ m}^3\end{aligned}$$

For 72 hours of pumping:

$$\begin{aligned}\text{Volume of water pumped-out} &= \text{total volume of water in the sump} \\ V_1 + V_2 + V_3 &= \text{inflow during interruption} + \text{inflow during}\end{aligned}$$

pumping

$$\text{But: } V_1 = V_2$$

$$\begin{aligned}2(Q_1 t_1) + Q_3 t_3 &= 6,000 \text{ m}^3 + (100 \text{ m}^3/\text{hr} \times 72 \text{ hrs}) \\ 2(60 \text{ m}^3/\text{hr} \times 72 \text{ hrs} \times (22/24)) + Q_3 \times 72 \text{ hrs} \times (22/24) &= 13,200 \text{ m}^3 \\ 7,920 \text{ m}^3 + (66 \text{ hrs} \times Q_3) &= 13,200 \text{ m}^3 \\ 66 \text{ hrs} \times Q_3 &= 5,280 \text{ m}^3 \\ Q_3 &= 80 \text{ m}^3/\text{hr} \times 1 \text{ hr}/60 \text{ min.} \\ Q_3 &= \mathbf{1.333 \text{ m}^3/\text{min.}}\end{aligned}$$

- Motor Horsepower:

$$H_p = \frac{wQH}{550 e_p}$$

$$H_p = \frac{62.4 \text{ lb}/\text{ft}^3 \times (1.333 \text{ m}^3/\text{min} \times 35.29 \text{ ft}^3/\text{m}^3) \times (300 \text{ m} \times 3.28 \text{ ft}/\text{m})}{(550 \text{ ft-lb}/\text{sec-hp} \times 60 \text{ sec}/\text{min}) \times 0.80}$$

$$H_p = 109.41 \text{ hp or } \mathbf{110 \text{ hp}}$$

- Pumping Cost:

Solving for the Motor Horsepower of Pump 1 and 2:

$$H_p = \frac{wQH}{550 e_p}$$

$$\begin{aligned}\text{ft/m)} \quad H_p &= \frac{62.4 \text{ lb}/\text{ft}^3 \times (60 \text{ m}^3/\text{hr} \times 35.29 \text{ ft}^3/\text{m}^3 \times 1 \text{ hr}/3600 \text{ sec}) \times (300 \text{ m} \times 3.28}{(550 \text{ ft-lb}/\text{sec-hp}) \times 0.80}\end{aligned}$$

$$H_p = \mathbf{82 \text{ hp}}$$

$$\begin{aligned}\text{Pumping Cost} &= \text{P1.33}/\text{kw-hr} \times (H_{p1} + H_{p2} + H_{p3}) \times 0.746 \text{ kw}/\text{hp} \times (72 \text{ hrs} \times 22/24) \\ \text{Pumping Cost} &= \text{P1.33}/\text{kw-hr} \times (82 \text{ hp} + 82 \text{ hp} + 110 \text{ hp}) \times 0.746 \text{ kw}/\text{hp} \times 66 \text{ hrs} \\ \text{Pumping Cost} &= \text{P1.33}/\text{kw-hr} \times 274 \text{ hp} \times 0.746 \text{ kw}/\text{hp} \times 66 \text{ hrs} \\ \text{Pumping Cost} &= \mathbf{P17,942}\end{aligned}$$

E.10 PUMPING PROBLEM:

AUGUST 1986. A 1,000-gallon tank is to be filled with water using 3 pumps. Pump A can fill the tank in 4 hours, Pump B in 7 hours and Pump C in 10 hours. If pump A is operated in 1 hour, pump B in 2.5 hours, how long should pump C be operated to fill the tank into capacity?

Solution:

Solving for capacity of each pump:

$$\begin{array}{lll} Q_a & = 1,000 \text{ gal./4 hrs} & = 250 \text{ gal/hr} \\ Q_b & = 1,000 \text{ gal./7 hrs} & = 143 \text{ gal/hr} \\ Q_c & = 1,000 \text{ gal./10 hrs} & = 100 \text{ gal/hr} \end{array}$$

Let: t = operating time of pump C

$$250 \text{ gal/hr} \times 1 \text{ hr} + 143 \text{ gal/hr} \times 2.5 \text{ hrs} + 100 \text{ gal/hr} \times t = 1,000 \text{ gal.}$$

$$250 + 357 + 100t = 1,000$$

$$100t = 393$$

$$\underline{t = 3.93 \text{ hrs}}$$

E.11 PUMPING PROBLEM:

AUGUST 1984. A small gold sluicing operation gets its water requirement from a reservoir 72 ft x 2.5 ft x 12 ft. It is desired that water be changed at 100 feet per minute using a 6-inch pump with 80% efficiency. How long will it take for the water to last?

Solution:

$$\text{Volume of reservoir} = 72 \text{ ft} \times 2.5 \text{ ft} \times 12 \text{ ft} = 2,160 \text{ cu. Ft.}$$

$$\text{Capacity, } Q = \text{Area} \times \text{Velocity}$$

$$Q = \pi/4 \times (6 \text{ in} \times 1 \text{ ft}/12 \text{ in})^2 \times 100 \text{ ft/min} \times \text{Pump efficiency}$$

$$Q = 19.635 \text{ ft}^3/\text{min} \times 0.80$$

$$Q = 15.71 \text{ ft}^3/\text{min}$$

Let: t = time to consume the water

$$t = \frac{2,160 \text{ ft}^3}{15.71 \text{ ft}^3/\text{min}}$$

$$t = \underline{137 \text{ min.}}$$

E.12 PUMPING PROBLEM:

AUGUST 1990. A shaft, which originally contains 400,000 m³ of water, is experiencing a daily influx of 5,000 m³ of water. It is desired to dewater the shaft for 60 days at 20 hours pumping per day. How many units of 1,000-gpm pumps with efficiencies of 60% should be used granting that additional stand-by units are required?

Solution:

$$\begin{aligned}\text{Water influx for 60 days} &= 5,000 \text{ m}^3/\text{day} \times 60 \text{ days} &= 300,000 \text{ m}^3 \\ \text{Total Volume for 60 days} &= 400,000 \text{ m}^3 + 300,000 \text{ m}^3 &= 700,000 \text{ m}^3\end{aligned}$$

Solving for volume dewatered per pump:

$$\begin{aligned}\text{Volume / pump} &= 1000 \text{ gal/min} \times 1 \text{ m}^3/264 \text{ gal} \times 60 \text{ min/hr} \times 20 \text{ hrs/day} \times 60 \text{ days} \\ \text{Volume / pump} &= 272,727 \text{ m}^3\end{aligned}$$

With 60% pump efficiency:

$$\begin{aligned}\text{Volume / pump} &= 272,727 \text{ m}^3 \times 0.60 \\ \text{Volume / pump} &= 163,636 \text{ m}^3\end{aligned}$$

Solving for no. of units required:

$$N = \frac{700,000 \text{ m}^3}{163,636 \text{ m}^3/\text{unit}}$$

$$N = 4.28 \text{ units}$$

$$N = 5 \text{ units}$$

E.13 PUMPING PROBLEM:

AUGUST 1992. At the mill, pump discharged mill tailings to 12 inch x 12 inch launder situated 80 feet above the level pump. The velocity of the stream of the mill tailings at head of the launder is 10 ft per second. Assume that the launder is 75% full. What is the rating of the pump being used? The specific gravity of the tailings is 1.2.

Solution:

$$\begin{aligned}Q &= AV \\ Q &= \{(12 \times 12) \text{ in.}^2 \times 1 \text{ ft}^2/144 \text{ in}^2 \times 0.75\} \times 10 \text{ ft/sec} \\ Q &= 7.5 \text{ ft}^3/\text{sec}.\end{aligned}$$

$$\text{Sp. Gr. Of tailings} = \frac{\text{Density of tailings}}{\text{Density of water}}$$

$$1.2 = \frac{\text{Density of tailings}}{62.4 \text{ lbs/ft}^3}$$

$$\text{Density of tailings} = 74.88 \text{ lbs/ft}^3$$

Then:

$$\text{Pump Hp} = \frac{wQH}{550}$$

$$\text{Pump Hp} = \frac{74.88 \text{ lbs/ft}^3 \times 7.5 \text{ ft}^3/\text{sec} \times 80 \text{ ft}}{550 \text{ ft-lb/sec-hp}}$$

$$\text{Pump Hp} = 81.7 \text{ hp}$$

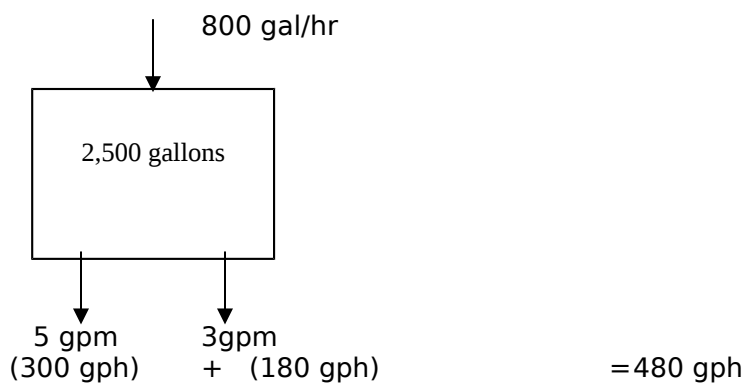
$$\text{Pump Hp} = \mathbf{85 \text{ hp}}$$

E.14 PUMPING PROBLEM:

AUGUST 1996. A 2,500-gallon water tank delivers water to 2 workings: 5 gpm to working # 1 and 3 gpm to working # 2. How long will it take for a pump of 800 gallon per

hour capacity to fill the tank if it operates 1.5 hours before water is allowed out of the tank?
 The tank is empty before the pump was operated.

Solution:



Volume of water for 1.5 hrs pumping:

$$V = 800 \text{ gal/hr} \times 1.5 \text{ hrs} = 1,200 \text{ gallons}$$

Let; Vr = remaining volume to fill the tank

$$Vr = 2,500 \text{ gal.} - 1,200 \text{ gal.} = 1,300 \text{ gal.}$$

Let; R = rate of water rising

$$\begin{aligned} R &= Q \text{ in} - Q \text{ out} \\ R &= 800 \text{ gal/hr} - 480 \text{ gal/hr} \\ R &= 320 \text{ gal/hr} \end{aligned}$$

Let: t = time to fill the tank

$$\begin{aligned} t &= \frac{Vr}{R} \\ t &= \frac{1,300 \text{ gal}}{320 \text{ gal/hr}} \\ t &= 4.06 \text{ hrs} \end{aligned}$$

Therefore:

$$\begin{aligned} \text{Total time to fill the tank} &= 1.5 \text{ hrs} + 4.06 \text{ hrs} \\ \text{Total time to fill the tank} &= \mathbf{5.56 \text{ hrs}} \end{aligned}$$

E.15 PUMPING PROBLEM:

AUGUST 1997. Tails certain mill must be discharged a velocity of 8 ft per second. If 5-hp, 8 inch x 6 inch pumps with 60% efficiency are to be used, how many units have to be installed? The total head is 50 ft and the specific gravity of the tailings is 1.75.

Solution:

Solving for total discharge:

$$Q = AV$$

$$Q = (8 \times 6) \text{ in}^2 \times 1 \text{ ft}^2/144 \text{ in}^2 \times 8 \text{ ft./sec}$$

$$Q = 2.67 \text{ ft}^3/\text{sec}$$

Solving for density of tailings:

$$\text{Specific Gravity of tailings} = \frac{\text{Density of tailings}}{\text{Density of water}}$$

$$1.75 = \frac{\text{Density of tailings}}{62.4 \text{ lbs/ft}^3}$$

$$\text{Density of tailings} = 109.20 \text{ lbs/ft}^3$$

Solving for discharge per pump:

$$\text{Pump Hp} = \frac{wQH}{550 \times e_p}$$

$$5 = \frac{109.2 \text{ lbs/ft}^3 \times Q \times 50 \text{ ft}}{550 \text{ ft-lb/sec-hp} \times 0.60}$$

$$5,460 Q = 1,650$$

$$Q = 0.30 \text{ ft}^3/\text{sec}$$

Then:

$$\text{No. of Pumps required} = \frac{2.67 \text{ ft}^3/\text{sec}}{0.30 \text{ ft}^3/\text{sec-unit}}$$

$$\text{No. of pumps required} = \mathbf{9 \text{ units}}$$

E.16 PUMPING PROBLEM:

AUGUST 1998. A sand fill plant consist of a slurry pump and a cyclone. The specific gravity of the slurry at the discharge plunge of the pump is 2.5; head is 10 ft. The cyclone underflow and the overflow discharge at 30 gpm and 50 gpm, respectively, what is the rating of the slurry pump if the efficiency is 60%. If 3 floors of a cut-and-fill stope measuring 50 feet long and 5 ft wide 6 ft high, how long will it take to finish the job.

Solution:

$$\begin{aligned}\text{Total Volume of stope} &= (50 \text{ ft} \times 5 \text{ ft} \times 6 \text{ ft}) \times 3 \\ \text{Total Volume of stope} &= 4,500 \text{ ft}^3\end{aligned}$$

$$\text{Specific Gravity of slurry} = \frac{\text{Density of slurry}}{\text{Density of water}}$$

$$2.5 = \frac{\text{Density of slurry}}{62.4 \text{ lbs/ft}^3}$$

$$\text{Density of slurry, } w = 156 \text{ lbs/ft}^3$$

$$\begin{aligned}Q &= 30 \text{ gpm} + 50 \text{ gpm} \\ Q &= 80 \text{ gal/min} \times 1 \text{ ft}^3/7.481 \text{ gal} \times 1 \text{ min}/60 \text{ sec} \\ Q &= 0.178 \text{ ft}^3/\text{sec}\end{aligned}$$

Then:

$$\text{Pump Hp} = \frac{wQH}{550 \times e_p}$$

$$\text{Pump Hp} = \frac{156 \text{ lbs/ft}^3 \times 0.178 \text{ ft}^3/\text{sec} \times 10 \text{ ft}}{550 \text{ ft-lb/sec-hp} \times 0.60}$$

$$\text{Pump Hp} = 0.85 \text{ hp}$$

$$\text{Pump Hp} = 1 \text{ hp}$$

Let N = time to finish the job

$$N = \frac{\text{Volume of stope}}{\text{Pump discharge, } Q}$$

$$N = \frac{4,500 \text{ ft}^3}{0.178 \text{ ft}^3/\text{sec}}$$

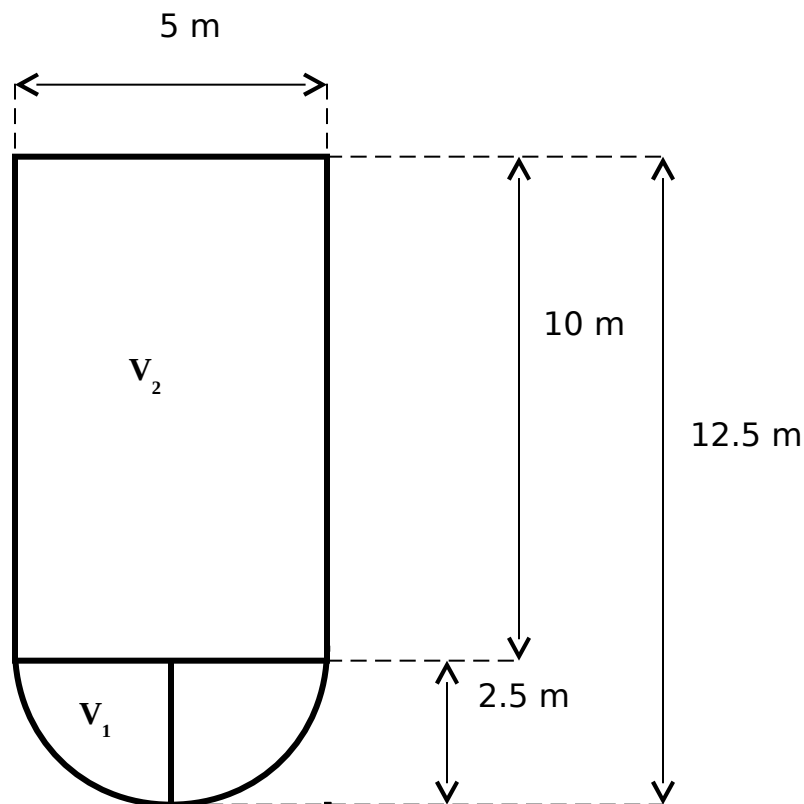
$$N = 25,281 \text{ sec.} \times 1 \text{ hr}/3,600 \text{ sec}$$

$$N = \mathbf{7.02 \text{ hrs}}$$

E.17 PUMPING PROBLEM

August, 2002. A cylindrical tank with hemi-spherical bottom was constructed to store water for underground wet drilling operations. The diameter of the tank is 5 meters and the total height is 12.5 meters from the bottom. Calculate the volume of water that can be stored in the tank if it is installed at an elevation 3 meters from the ground.

Solution:



$$\begin{aligned}
 \text{Volume of Hemisphere, } V_1 &= \frac{2}{3} \pi r^3 \\
 \text{Volume of Hemisphere, } V_1 &= \frac{2}{3} \times 3.1416 \times (2.5)^3 \\
 \text{Volume of Hemisphere, } V_1 &= 32.725 \text{ cu.m.}
 \end{aligned}$$

$$\begin{aligned}
 \text{Volume of Cylinder, } V_2 &= \pi r^2 H \\
 \text{Volume of Cylinder, } V_2 &= 3.1416 \times (2.5)^2 \times 10 \\
 \text{Volume of Cylinder, } V_2 &= 196.35 \text{ cu.m.}
 \end{aligned}$$

$$\text{Total Volume} = 229 \text{ cu.m.}$$

MINING ENGINEERING 1 HAULING PROBLEM

F.1 HAULING PROBLEM:ok

August, 1996. In a truck and shovel operation, the time for a truck to position itself at the shovel and be loaded are 0.50 minute and 3 minutes respectively. It then takes the truck 10 minutes to haul the ore, dump it at the coarse ore bin, and return to the shovel. How many trucks do you need to keep the shovel busy with a little waiting time for the trucks?

a. 4 trucks b. 3 trucks c. 5 trucks d. 6 trucks

Solution:

Loading time = positioning + loading
Loading time = 0.50 min. + 3 min.
Loading time = 3.5 min.

Hauling time = hauling + dumping + return
Hauling time = 10 min.

Cycle time = Loading time + Hauling time
Cycle time = 3.5 min + 10 min.
Cycle time = 13.5 min

To keep the shovel busy:

No. of trucks needed = $\frac{\text{Cycle time}}{\text{Loading time/truck}}$

No. of trucks needed = $\frac{13.5 \text{ min.}}{3.5 \text{ min/truck}}$

No. of trucks needed = **4 trucks**

F.2 HAULING PROBLEM:

August, 2003. A newly discovered porphyry copper deposit in Nueva Viscaya will be developed under the new MPSA system. The best applicable mining method is by open pit. The haulage trucks to be used are WABCO 85 tonners with payload of 77 MT per truck. Truck mechanical availability is assumed at 70%. Ore and waste haul cycles are established at 16 and 10 minutes respectively. The mine is planned to deliver 25,000 MT of ore per day to the crushing plant at the strip ratio of 1.5 : 1. The pit will operate 3 shifts per day of 8 hours each with time efficiency of 50 min per 60 min. As the Mine Planning Engineer, calculate how many haul trucks will you need to sustain ore deliveries and at the same time maintain workable pit geometry at any one time. Allocate one (1) haul truck spare unit for every five (5) trucks running.

A. 14 units B. 10 units C. 11 units D. 12 units E. 13 units

Solution:

Waste Tonnage	=	1.5 x 25,000 MT/day
Waste tonnage	=	37,500 MT/day
Total Tonnage to Haul=		62,500 MT/day

Hauling Ore:

Actual Truck Load	=	77 MT x 0.70	=	53.90 MT
Effective time/day	=	8 hrs/shift x 3 shifts/day x 50 min/hr		
Effective time/day	=	1,200 min/day		
Tonnage/truck/day	=	53.90 MT/truck ----- 16 min/trip	x	1,200 min/day
Tonnage/truck/day	=	4,042.5 MT		
No. of Trucks needed (Ore)	=	25,000 MT/day ----- 4,042.5 MT/truck/day		
No. of Trucks needed (Ore)	=	6.1843 units		

Hauling Waste:

Tonnage/truck/day	=	53.90 MT/truck ----- 10 min/trip	x	1,200 min/day
	=	6,468 MT		
No. of Trucks needed (Waste)	=	37,500 MT/day ----- 6,468 MT/truck/day		
No. of Trucks needed (Waste)	=	5.7978 units		
Total Trucks needed	=	11.9821 units		
Total Trucks needed	=	12 units		
Allowing 1 truck spare for every 5 trucks running, then the total spare will be 2 units.				
Therefore:				
Total trucks needed	=	12 units + 2 spare		
Total trucks needed	=	<u>14 units</u>		

F.3 HAULING PROBLEM:

AUGUST 1998. In a underground mine, hauling is done in 6 hours per shift with locomotives pulling 10 cars. Tonnage hauled per trip of 1 locomotive is 30 MT at a hauling distance of 800 meters with an average speed of 4 Km per hour. It takes 0.75 min to load mine car and another 0.75 minute to dump its contents. If the tonnage to be milled is 1,300 MT per shift, how many units of locomotives are required?

Solution:

No. of Locomotives	=	Total Tonnage hauled ----- Tonnage/locomotive
Tonnage/Locomotive	=	Tonnage/trip x No. of trips/locomotive
No. of trips/locomotive	=	Hauling time/shift -----

			Cycle time/trip
Time	Cycle Time/trip	=	Hauling time + Loading time + Dumping
	Hauling Time	=	0.80 km x 1hr/4 km x 60 min/hr x2
	Hauling Time	=	24 mins.
	Loading time	=	0.75 min/mine car x 10 mine cars
	Loading Time	=	7.5 mins.
	Dumping Time	=	0.75 min/mine car x 10 mine cars
	Dumping Time	=	7.5 mins.
Then,	Cycle Time/trip	=	24 mins + 7.5 mins. + 7.5 mins.
	Cycle Time/trip	=	39 mins/trip
Then,			
	No. of trips/locomotive	=	$\frac{6 \text{ hrs/shift} \times 60 \text{ min/hr}}{39 \text{ mins/trip}}$
	No. of trips/locomotive	=	9.23 trips/locomotive
	No. of trips/locomotive	=	9 trips/locomotive
Then,			
	Tonnage/shift-locomotive	=	9 trips/locomotive x 30 MT/trip
	Tonnage/shift-locomotive	=	270 MT/locomotive
Therefore,			
	No. of locomotives required	=	$\frac{1,300 \text{ MT}}{270 \text{ MT/locomotive}}$
	No. of locomotives required	=	4.815 locomotives
	No. of locomotives required	=	<u>5 locomotives</u>

F.4 HAULING PROBLEM:ok

August, 1983. In a limestone quarry, a bench measuring 3 meters x 6 meters x 6 meters was blasted. Allowing 10% overbreak, calculate the number of 3 cubic meters truck loads that will be required to move the loose rocks to the mill bins. Assume 30% voids for the broken limestone. (Ans=52 truckloads)

Solution:

$$\text{Blast Volume} = 3\text{m} \times 6\text{m} \times 6\text{m} = 108 \text{ m}^3$$

With 10% overbreak:

$$\text{Blast Volume} = 108 \text{ m}^3 \times 1.1 = 118.80 \text{ m}^3 \text{ (bcm)}$$

With 30% voids:

$$\text{Blast Volume} = 118.80 \text{ m}^3 \times 1.3 = 154.44 \text{ m}^3 \text{ (lcm)}$$

Then:

$$\begin{aligned} \text{No. of trucks} &= \frac{154.44 \text{ m}^3}{3 \text{ m}^3/\text{truck}} \\ \text{No. of trucks} &= \mathbf{52 \text{ trucks}} \end{aligned}$$

F.5 HAULING PROBLEM:ok

August, 1986. Given the following data, determine the number of trucks that would be required per shift to haul copper ores from the open pit to the milling plant.

Hauling time per shift	=	6 hours
Truck Capacity	=	35 MT rear dump truck
Production rate per shift	=	18,900 MT
Total cycle time	=	5.53 min.
Assume efficiency of 50 productive minute per hour.		

Solution:

$$\begin{aligned} \text{TPH} &= \frac{\text{Truck capacity} \times \% \text{ utilization}}{\text{Cycle time}} \\ \text{TPH} &= \frac{35 \text{ MT} \times (50/60)}{5.53 \text{ min} \times 1 \text{ hr}/60 \text{ min}} \\ \text{TPH} &= 316.46 \text{ MT/hr} \\ \text{Tonnage/truck-shift} &= 316.46 \text{ MT/hr} \times 6 \text{ hrs/shift} \\ \text{Tonnage/truck-shift} &= 1,898.76 \text{ MT} \\ \text{No. of trucks} &= \frac{18,900 \text{ MT}}{1,898.76 \text{ MT/truck}} \\ \text{No. of trucks} &= \mathbf{10 \text{ trucks}} \end{aligned}$$

F.6 HAULING PROBLEM:ok

AUGUST 2005.

Determine the sustained production rate in surface mine, given the following conditions:

6 trucks	=	at 30 tons capacity
working time	=	55 min./hr, 7 hr./shift
haulage cycle	=	14 min.
efficiency	=	90%

- a. 3,500 tpd b. **4,500 tpd** c. 5,000 tpd d. 2,000 tpd

Solution:

$$\text{Truck capacity} = \frac{\text{Load}}{\text{Cycle time}}$$

$$\begin{aligned} \text{min Truck capacity} &= \frac{30 \text{ tons}}{14 \text{ min} \times 1 \text{ hr}/60 \text{ min} \times 1 \text{ shift}/7 \text{ hrs} \times 1 \text{ day}/1 \text{ shift}} \times 55 \text{ min}/60 \\ \text{Truck capacity} &= 825 \text{ TPD} \end{aligned}$$

With 90 % efficiency:

$$\begin{aligned} \text{Truck capacity} &= 825 \text{ TPD} \times 0.90 \\ \text{Truck capacity} &= 742.5 \text{ TPD/truck} \end{aligned}$$

Then:

$$\begin{aligned} \text{Sustained Production rate} &= 742.5 \text{ TPD/truck} \times 6 \text{ trucks} \\ \text{Sustained Production rate} &= 4,455 \text{ TPD} \\ \text{Sustained Production rate} &= \mathbf{4,500 \text{ TPD}} \end{aligned}$$

F.7 HAULING PROBLEM:ok

AUGUST 2006. A truck working in an open pit mine for 480 hours during the month was under repair for 60 hours and on standby for 180 hours. What was the effective utilization of the truck?

- a. 89% b. 92% c. **67%** d. 73%

Solution:

$$\begin{aligned} \text{Working hours} &= 480 \text{ hrs} \\ \text{Repair} &= 60 \text{ hrs} \\ \text{Standby} &= 180 \text{ hrs} \\ \hline \text{Total} &= 720 \text{ hrs} \end{aligned}$$

$$\% \text{ Utilization} = \frac{\text{Working hours}}{\text{Total hours}} \times 100$$

$$\% \text{ Utilization} = \frac{480 \text{ hrs}}{720} \times 100$$

$$\begin{aligned} &720 \text{ hrs} \\ \% \text{ Utilization} &= \mathbf{67\%} \end{aligned}$$

F.8 HAULING PROBLEM:ok

August, 1997. A shovel worked 500 hours a month, was under repair for 100 hours and on stand-by for 50 hours. What was the shovel physical availability?
 a. **84.6%** b. 83.3% c. 90.9% d. 76.9%

Solution:

Working hours	=	500 hrs
Repair	=	100 hrs
Stand-by	=	50 hrs

Total	=	650 hrs

Physical Availability	=	Working hrs + Stand-by ----- Total hours	x 100
Physical Availability	=	500 hrs + 50 hrs ----- 650 hours	x 100
Physical Availability	=	<u>84.6 %</u>	

F.9 HAULING PROBLEM:ok

August, 1998. Being the Chief Mining Engineer of a big copper/gold open pit mine, you are involved in equipment selection for appropriate off-highway dump trucks for the mine. You are assigned to review the technical specifications and features of 120-tonner WABCO trucks. If the truck engine has a rated horsepower rating of 250 hp and the truck accessories are likewise rated as follows: main oil pump – 5 hp; alternator – 3 hp; aircon – 2 hp; all other electrical and support accessories – 5 hp, what is therefore the trucks flywheel or brake horsepower rating?
 a. 265 hp b. 290 hp c. **235 hp** d. 200 hp e. 150 hp

Solution:

Rated Hp rating of engine accessories	=	Brake Hp rating + Truck
250 Hp	=	Brake Hp + 15 Hp
Brake Hp	=	<u>235 Hp</u>

F.10 HAULING PROBLEM:ok

August, 2003. A fleet of 15 units off-highway dump trucks are fitted with 275 Hp diesel engines each. What is the resulting flywheel or brake horsepower rating (BHp) of the trucks if they have the following engine accessories:

Motor accessories	-	6.0 Hp
Electrical motors	-	2.0 Hp
Cooling System	-	1.5 Hp
Other essentials	-	4.0 Hp

A. 261.5 Hp B. 265.5 Hp C. 288.5 Hp D. 269.0 Hp E. 273.5 Hp

Solution:

$$\begin{aligned} \text{Brake Horsepower Consumers} &= \text{Engine Rated Horsepower} - \text{Power} \\ \text{Brake Horsepower} &= 275 \text{ Hp} - 6 \text{ Hp} - 2 \text{ Hp} - 1.5 \text{ Hp} - 4 \text{ Hp} \\ \text{Brake Horsepower} &= 261.5 \text{ Hp} \end{aligned}$$

F.11 HAULING PROBLEM:ok

August, 2005. A newly purchased Caterpillar Model 3306 Diesel Engine has a rated output of 225 Hp. If the truck accessories connected to the engine have a total rating of 15 Hp, what is the brake horsepower rating of the engine?

A. 250 Hp B. 200 Hp C. 255 Hp **D. 210 Hp** E. 240 Hp

Solution:

$$\begin{aligned} \text{Brake Horsepower} &= \text{Engine Rated Hp} - \text{Accessories Hp} \\ \text{Brake Horsepower} &= 225 \text{ Hp} - 15 \text{ Hp} \\ \text{Brake Horsepower} &= 210 \text{ Hp} \end{aligned}$$

**MINING ENGINEERING 1
SHIPPING PROBLEM**

G.1 SHIPPING PROBLEM:

August, 1983. A mining company has at its pier stockpile 2000 wet metric tons of copper concentrate. A boat is scheduled to load 7500 wet metric tons of concentrate. Haulage of concentrate from the mill to the pier is by 10-ton trucks with intervals of 2 trucks per hour.

- a. How many days will it take to fill-up the commitment if the trucks operate on a 20-hour per day basis. (Ans=14 days)
- b. If the moisture content of the concentrate is 7% and the average grade is 28% Cu, how many kilograms of metallic copper are there in the whole shipment? (Ans=1,953,000 kgs)

Solution:

$$\begin{aligned} \text{Concentrate to be hauled} &= 7,500 \text{ WMT} - 2,000 \text{ WMT} \\ \text{Concentrate to be hauled} &= 5,500 \text{ WMT} \end{aligned}$$

a. No, of days to fill-up the commitment, N:

$$\begin{aligned} N &= \frac{5,500 \text{ WMT}}{20 \text{ hrs/day} \times 2 \text{ trucks/hr} \times 10 \text{ MT/truck}} \\ N &= 13.75 \text{ days} \\ N &= 14 \text{ days} \end{aligned}$$

b. Weight of metallic copper, W:

$$\begin{aligned} W &= 7,500 \text{ WMT} \times 0.93 \text{ DMT/WMT} \times 0.28 \times 1,000 \text{ kg/MT} \\ W &= \mathbf{1,953,000 \text{ kgs.}} \end{aligned}$$

G.2 SHIPPING PROBLEM:

AUGUST 1982. A chromite mining company, which is programmed to ship 7,500 WMT of lump ore, has 1,000 WMT of chromite ore stockpiled at the pier. To cover the deficit, a tramline haulage system from the pier will work 20 hours per day in transporting the ore. If each bucket has a capacity of 0.75 WMT and are spaced 50-second interval, how many days will take to satisfy the tonnage?

Solution:

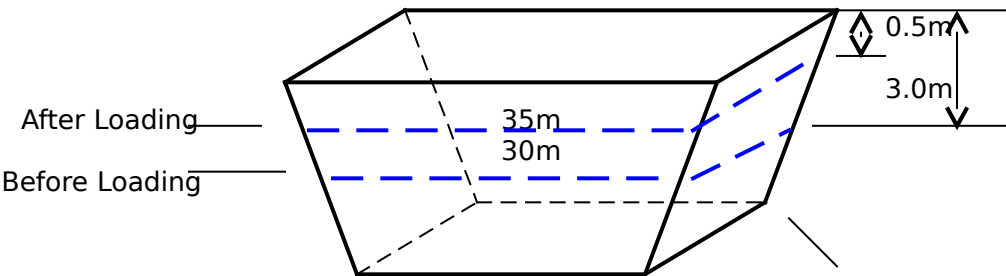
No. of days to transport	=	$\frac{\text{Total tonnage to transport}}{\text{Rate of transport}}$
Total tonnage to transport	=	Ore shipment requirement - Ore stockpiled at pier
Total tonnage to transport	=	7,500 WMT - 1,000 WMT
Total tonnage to transport	=	6,500 WMT
Rate of transport	=	0.75 WMT/50sec x 3,600 sec/hr x 20 hrs/day
Rate of transport	=	1,080 WMT/day
No. of days to transport	=	$\frac{6,500 \text{ WMT}}{1,080 \text{ WMT/day}}$
No. of days to transport	=	6 days

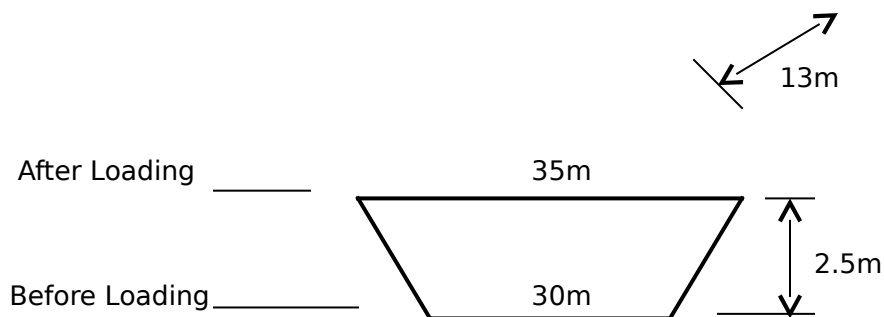
G.3 SHIPPING PROBLEM:

August, 1980. A stockpile of lumpy chromite was loaded into the barge. Before loading, the observed waterline along each side of the barge measured 30 meters long while at each side of the barge the waterline was 13 meters long. It was observed also that the surface of the water was 3 meters below the top rim of the barge. After loading, the waterline along each side became 35 meters long while along the ends of the barge, the length of the waterline did not change but the top rim of the barge was only 0.5 meter above the water's surface.

Compute:

- a. Metric tons of ore loaded.
- b. The volume of the ore loaded if void is 30% and the specific gravity of the ore is 4.2.





From Law of Buoyancy:

Then: Weight of material immersed = weight of material displaced

water a. Weight of ore = Volume of barge immersed x density of sea

$$\text{Volume of barge immersed} = \frac{35\text{m} + 30\text{m}}{2} \times 13\text{m}$$

$$\text{Volume of barge immersed} = 1,056.25 \text{ m}^3$$

$$\begin{aligned} \text{Weight of ore} &= 1,056.25 \text{ m}^3 \times 64 \text{ lbs/ft}^3 \times 35.29 \text{ ft}^3/\text{m}^3 \times 1 \\ \text{Weight of ore} &= \underline{\underline{1,082 \text{ MT}}} \end{aligned}$$

$$\text{b. Volume of ore} = \frac{\text{Weight of ore}}{\text{density of ore}}$$

$$\text{Volume of ore} = \frac{1,082 \text{ MT}}{4.2 \text{ MT/m}^3}$$

$$\text{Volume of ore} = 257.62 \text{ m}^3$$

With 30% voids:

$$\begin{aligned} \text{Volume of ore} &= 257.62 \text{ m}^3/1.3 \\ \text{Volume of ore} &= \underline{\underline{198 \text{ m}^3}} \end{aligned}$$

MINING ENGINEERING 1 VENTILATION PROBLEM

H.1 VENTILATION

August 2004, 2006, 2008. The air recirculation system of an underground mine has the following daily compressed air data for its pneumatic equipment.

Tank 1	12,000 cu.m.	60 psi
Tank 2	10,000 cu.m.	50 psi
Tank 3	15,000 cu.m.	70 psi
Tank 4	15,000 cu.m.	77 psi
Tank 5	10,000 cu.m.	75 psi

What average pneumatic pressure is available assuming there are no leaks along the full length of the underground pipelines? Express your answer in the nearest rounded bar pressure.

- A. 7 bars **B. 5 bars** C. 6 bars D. 8 bars E. 4 bars

Solution:

	Volume	Pressure	Volume x Pressure
Tank 1	12,000 cu.m.	60 psi	720,000
Tank 2	10,000 cu.m.	50 psi	500,000
Tank 3	15,000 cu.m.	70 psi	1,050,000
Tank 4	15,000 cu.m.	77 psi	1,155,000
Tank 5	10,000 cu.m.	75 psi	750,000
	-----		-----
	62,000 cu.m.		4,175,000 cu.m.

$$\text{Average Pressure} = \frac{4,175,000 \text{ cu.m. (psi)}}{62,000 \text{ cu.m.}}$$

$$\text{Average Pressure} = 67.34 \text{ psi} \times 1 \text{ bar} / 14.7 \text{ psi}$$

$$\text{Average Pressure} = 4.58 \text{ bars}$$

$$\text{Average Pressure} = 5 \text{ bars}$$

H.2 VENTILATION:

AUGUST 1984. What size of regulator will be needed to pass 20,000 cfm of air at a pressure drop of 2 inches water gauge?

Solution:

$$\text{a. } A = \frac{40Q}{\sqrt{H}}$$

Where;

A = area of regulator, ft²

Q = airflow quantity in 100,000 ft³/min

H = pressure drop in inches water gauge

$$A = \frac{40 \times (20,000 \text{ cfm}/100,000)}{\sqrt{2}}$$

$$A = 5.66 \text{ ft}^2$$

H.3 VENTILATION:

August, 1983. The mine air ventilating a drift is found to be flowing at the rate of 80 cubic feet per minute with an air pressure of 10 lbs per cubic foot. If the flow rate is increased to 160,000 cubic feet per minute, determine the expected pressure that will be registered?

Solution:

$$P = RQ^2$$

Where:

P = air pressure, lbs/ft²

R = resistance to airflow, Atkinson (lbs/ft²-kilocusec)

Q = airflow, kilocusec (1 kilocusec = 1,000 ft³/sec)

For 80 cfm:

$$R = \frac{P}{Q^2}$$

$$R = \frac{10 \text{ lbs/ft}^2}{80 \text{ ft}^3/\text{min} \times 1 \text{ kilocusec}/1000 \text{ ft}^3/\text{sec} \times 60 \text{ sec/min}}$$

$$R = \underline{\underline{2.08 \text{ Atkinson}}}$$

For 160,000 cfm:

$$P = RQ^2$$

$$P = (2.08 \text{ lbs/ft}^2\text{-kilocusec}) \times (160,000 \text{ ft}^3/\text{min} \times 1 \text{ kilocusec}/1000 \text{ ft}^3/\text{sec} \times 60 \text{ sec/min})^2$$

$$P = \underline{\underline{14.79 \text{ lbs/ft}^2}}$$

H.4 VENTILATION:

AUGUST 1987. A fan circulates 240,000 cfm and the water gauge is 3 inches. What is the resistance of the mine in Atkinson and the horsepower in the air?

Solution:

$$a. \quad P = RQ^2$$

$$R = \frac{P}{Q^2}$$

Where:

R = Air Resistance in Atkinson (lb/ft²-kilocusec)

P = Air Pressure in lb/ft³

Q = Air flow in kilocusec

$$\frac{3 \text{ in. wg} \times 5.2 \text{ lbs/ft}^2}{1 \text{ in. wg}}$$

$$R = \frac{(240,000 \text{ ft}^3/\text{min} \times 1 \text{ min}/60 \text{ sec} \times \frac{1 \text{ kilocusec}}{1000 \text{ ft}^3/\text{sec}})^2}{1000 \text{ ft}^3/\text{sec}}$$

$$R = \frac{15.6 \text{ lbs/ft}^2}{16 \text{ kilocusec}}$$

$$R = \underline{\mathbf{0.975 \text{ Atkinson}}}$$

$$\text{b. Air Hp} = \frac{wQH}{550 \times e_f}$$

where;

w = unit weight of gas, lb/ft³

Q = air flow, cfs

H = head in ft of gas

H = $P_h/12 \times 62.4/w$

Ph = pressure head, in. wg

550 = constant in ft-lb/sec-hp

e_f = fan efficiency

$$H = \frac{3 \text{ in. water} \times 1 \text{ ft}/12 \text{ in} \times 62.4 \text{ lb/ft}^3}{15.6 \text{ lbs/ft}^2 \times 1}$$

$$H = \frac{15.6}{w \text{ (lb/ft}^3\text{-gas)}}$$

$$H = \frac{15.6}{w} \text{ (ft. of gas)}$$

Substitute H in the equation:

$$\text{Air Hp} = \frac{wQ \times (15.6/w)}{550 \times e_p}$$

$$\text{Air Hp} = \frac{15.6 \times Q}{550 \times e_p}$$

$$\text{Air Hp} = \frac{15.6 \text{ lb/ft}^2 \times 240,000 \text{ ft}^3/\text{min} \times 1 \text{ min}/60 \text{ sec}}{550 \text{ ft-lb/sec-hp}}$$

$$\text{Air Hp} = \underline{\mathbf{115 \text{ hp}}}$$

H.5 VENTILATION:

AUGUST 1986. A coal mine section has a methane emission rate of 200 cfm and a 0.2% CH₄ concentration in the intake air. What is the required ventilation airflow. The threshold limit value of CH₄ is 1.0%.

Solution:

$$Q_a = \frac{Q_g}{(L - L_o)} - Q_g$$

Where:

Q_a = ventilation airflow

Q_g = gas emission rate

L = threshold limit value

L_o = gas concentration

$$Q_a = \frac{200 \text{ cfm}}{(1 - 0.2)} - 200 \text{ cfm}$$

$$Q_a = 50 \text{ cfm}$$

MINING ENGINEERING 1 MINE SAFETY PROBLEM

I.1 MINE SAFETY PROBLEM:

AUGUST 2005. In a mine with 2,000 employees where 100 were on sick leave and another 100 were on vacation leave for a particular month, there were 18 lost time accidents. Assuming the employees worked for 25 days (for 8 hrs) and no one worked overtime, what was the accident frequency rate per million manhours?

- e. **50** b. 45 c. 100 d. none of these

Solution:

$$\begin{aligned} \text{Frequency Rate, FR} &= \frac{(\text{Non-Fatal} + \text{Fatal}) \times (1,000,000)}{\text{Total Manhours worked}} \\ \text{Total employees on duty} &= 2,000 - 100 - 100 \\ \text{Total employees on duty} &= 1,800 \end{aligned}$$

$$\text{hrs/day} \quad \text{Total manhours worked} = 1,800 \text{ man} \times 25 \text{ days} \times 8$$

$$\text{Total manhours worked} = 360,000 \text{ man-hours}$$

$$\text{FR} = \frac{18 \text{ manhours} \times 1,000,000}{360,000 \text{ manhours}}$$

$$\text{FR} = 50$$

I.2 MINE SAFETY PROBLEM

August, 2004. A mine with 1,500 miners working at 80% attendance rate on regular hours, 365 days a year, incurred a total of four (4) accidents with 23 days lost during the fiscal year just concluded. What is the company's severity rate?

A. 42.51 B. 72.51 C. 52.51 D. 62.51 E. 32.51

Solution:

$$\text{Severity Rate} = \frac{\text{No. of Days Lost} \times 1,000,000}{\text{Total Manhours Worked}}$$

$$\text{Manhours Worked} = 1,500 \text{ man} \times 0.80 \times 8 \text{ hours/day} \times 365$$

days/year

$$\text{Manhours Worked} = 3,504,000 \text{ manhours}$$

$$\text{Severity Rate, SR} = \frac{23 \text{ days lost} \times 1,000,000}{3,504,000 \text{ manhours}}$$

$$\text{Severity Rate, SR} = 6.56$$

$$\text{Frequency Rate, FR} = \frac{(\text{Non-Fatal} + \text{Fatal}) \times 1,000,000}{\text{Total Manhours Worked}}$$

		Total Manhours Worked
Frequency Rate, FR	=	$\frac{4 \times 1,000,000}{3,504,000}$
Frequency Rate, FR	=	1.14
Incidence Rate, IR	=	$\frac{(\text{NLTA} + \text{NF} + \text{Fatal}) \times 1,000,000}{\text{Total Manhours Worked}}$
Incidence Rate (IR)	=	$\frac{4 \times 1,000,000}{3,504,000}$
Incidence Rate (IR)	=	1.14
CFSR	=	$\frac{\text{FR} \times \text{SR}}{2}$
CFSR	=	$\frac{1.14 \times 6.56}{2}$
CFSR	=	3.74

J.1 CONVEYING PROBLEM

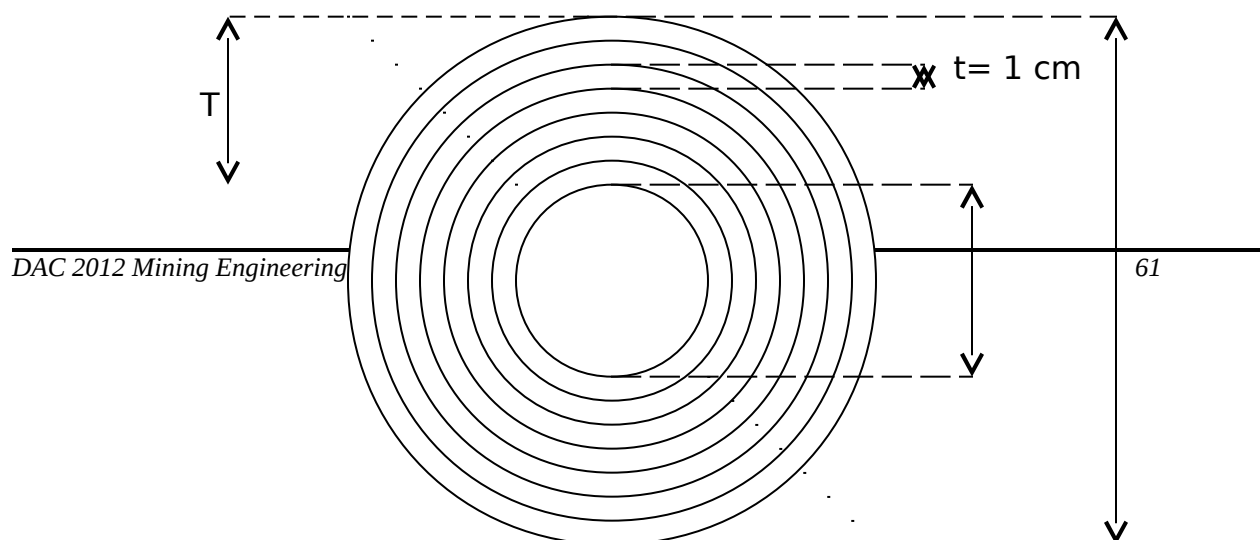
August 2004, 2005. Philex Mining Corporation ordered new set of conveyor belt to replace its cable belt at its Padcal Copper Mine. The conveyor belt that arrived in the mine are logged as follows:

Quantity	-	5 spolls
Spool Dimension:		
Core diameter	-	18 inches
Outside diameter	-	5 feet
Conveyor Width	-	36 inches
Conveyor thickness	-	1 cm.

What is the total length of the conveyor belts? Express the length in the nearest rounded meter.

- A. 929 M B. 1,029 M C. 729 M D. 629 M **E. 829 M**

Solution:



18 in. 5 ft

Let : T = total thickness of conveyor belt in the spool

$$\begin{aligned} 2T + (18 \text{ in} \times 2.54 \text{ cm/in.}) &= 5 \text{ ft.} \times 12 \text{ in./ft} \times 2.54 \text{ cm/in} \\ 2T &= 152.4 \text{ cm} - 45.72 \text{ cm} \\ T &= 53.34 \text{ cm} \end{aligned}$$

Let : N = no. of turns of conveyor belt in the spool

$$\begin{aligned} N &= \frac{53.34 \text{ cm}}{1 \text{ cm / turn}} \\ N &= 53.34 \text{ turns} \end{aligned}$$

Solving for Total Length of Conveyor in the spool:

$$\begin{aligned} \text{Total Length} &= L1 + L2 + L3 + L4 + L5 \dots\dots\dots + \\ L53.34 \end{aligned}$$

But:

$$\begin{aligned} L1 &= ii \text{ D} \\ L2 &= ii (D - 2t) \\ L3 &= ii (D - 4t) \\ L4 &= ii (D - 6t) \\ L5 &= ii (D - 8t) \end{aligned}$$

$$\begin{aligned} \text{Total Length} &= ii \text{ D} + ii (D-2t) + ii (D - 4t) + ii (D - 6t) + ii (D - 8t) \\ \text{Total Length} &= ii \times (D + D - 2t + D - 4t + D - 6t + D - 8t) \\ \text{Total Length} &= ii \times (5D - 20t) \\ \text{Total Length} &= ii \times (ND - ((N \times (N-1)t) \\ \text{Total Length} &= ii \times N (D - (N-1) t) \\ \text{Total Length} &= ii N \times (D - (N-1) t) \end{aligned}$$

$$\begin{aligned} \text{Total Length} &= (3.1416 \times 53.34) \times ((5 \text{ ft} \times 30.48 \text{ cm/ft})-(53.34 -1) \\ &\text{(1 cm)}) \\ \text{Total Length} &= 167.57 \times 100.06 \\ \text{Total Length} &= 16,767 \text{ cm} \times 1 \text{ m} / 100 \text{ cm} \\ \text{Total Length} &= 167.67 \text{ m} / \text{spool} \end{aligned}$$

For 5 spools:

$$\begin{aligned} \text{Total Length} &= 167.67 \text{ m} \times 5 \\ \text{Total Length} &= 838 \text{ m} \end{aligned}$$

