IMPERIAL COLLEGE LONDON

Metals and Energy Finance (MSc)

Project Evaluation - Extractive Metallurgy

3 May 2018 10.00 – 12.00

Answer THREE out of the FOUR questions.

Each question carried 25 marks. . Do not answer more than three questions.

QUESTION 1 – Comminution

A crushing circuit consists of a primary jaw crusher, a screen and a secondary cone crusher. The primary crusher is located adjacent to the pit and is in open circuit, with the crusher product being conveyed to the plant. In the plant this material is fed into the cone crusher, which is in closed circuit with a screen which takes the product from this crusher. The product from this circuit is the feed to the mill circuit:

a) Draw the circuit, labelling streams and items. (3)
b) Why might it be advantageous to locate the primary crusher near the pit (1)

The total solid feed to the primary crusher is 1200 t/hr. In analysing the performance of this circuit, consider two size fractions, namely the particles larger than 1cm and those passing 1cm. In the feed to the primary crusher only 15% of the material is already finer than 1cm. Assume that the jaw crusher will break 30% of the material coarser than 1cm to finer than this size, while the cone crusher will break 50% of the material above 1cm to finer than this size. Assume that the screen's performance is such that the partition number for the coarser particles is 0.8, while it is 0.3 for the finer particles.

c) Carry out a mass balance over the circuit calculating the mass flowrates of each of the components in each of the streams. What is the percentage passing 1cm in the crusher circuit product/mill circuit feed? (10)

It can be hard to measure the flow rates of particles and so mass balances often need to be carried out using other measurements. A new screen is being tested and measurements of the particle size were taken. In the feed 30% passed 1cm, while in the underflow 80% passed 1cm and in the overflow 10% passed 1cm.

d) What is the mass recovery to the overflow? (4)

While some water is added into the crushing circuit, it is not enough to achieve the desired slurry viscosity for the milling, which is a strong function of the water content. In the product from the crushing circuit there is already 10% water by mass.

e) How much additional water needs to be added to achieve a solids content of 30% by volume if the solid density is 2.7? (4)

The 80% passing size for the mill circuit feed is 0.8cm and for the product it is 90 µm.

f) What will the power requirement of the milling circuit be given that the Bond work index of the material is 15kWh/t? (3)

QUESTION 2 – Flotation

A rougher bank has 4 cells in it. The feed to the bank has a solids mass flowrate of 1000 t/hr and a grade of 1.5% copper. In order to assess the performance the flowrate and grade of the concentrate streams from each of the cells was measured:

	Cell 1	Cell 2	Cell 3	Cell 4
Concentrate Grade	30	25	20	15
Solids flow rate to concentrate (t/hr)	8	6	5	4

a) Calculate the cumulative down the bank grade recovery curve. (8)

Flotation circuits typically consist of banks of cells, with a rougher, cleaner, scavenger arrangement being a commonly used one.

b) Sketch a standard rougher, cleaner, scavenger circuit labelling the banks and streams. (3)

A circuit with such a configuration achieves an 80% recovery over the rougher bank, a 50% recovery over the scavenger bank and a 60% recovery from the cleaner bank.

c) What is the overall recovery of the circuit? (6)

Assuming that we keep the operating conditions of the other banks constant, but increase the froth depth in the cleaner circuit.

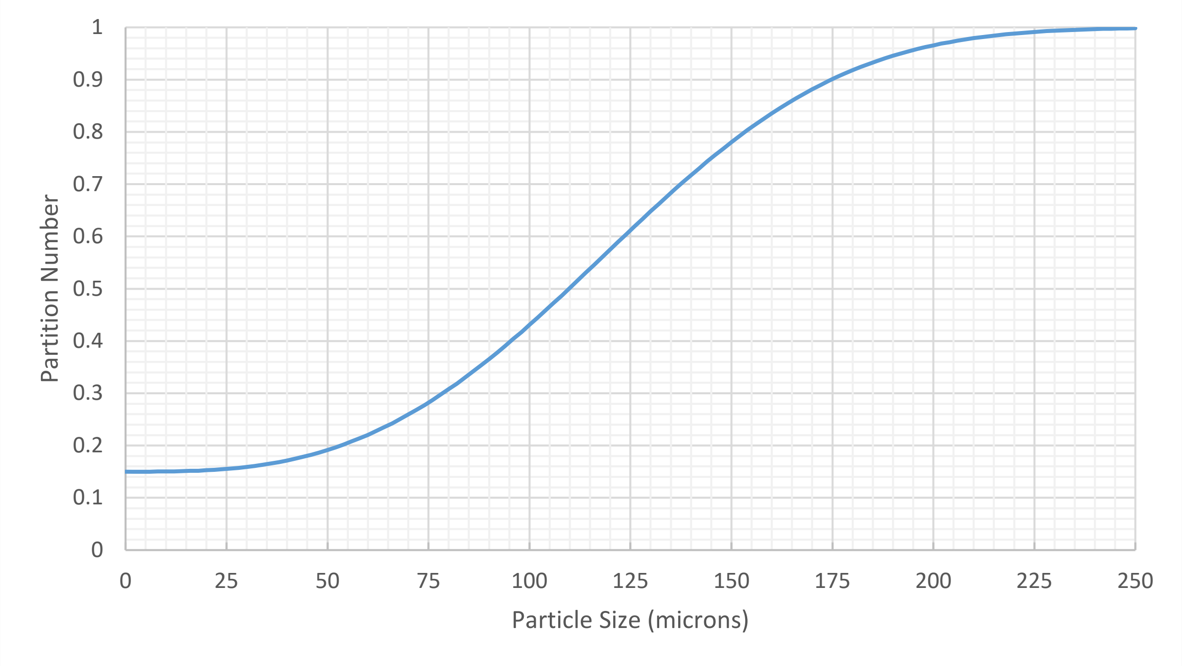
d) What would happen to the circuit recovery and concentrate grade? (1)
e) What would happen to the feed rates and grades for each of the banks and why? (3)

Typically, for a given feed rate and grade, the circuit recovery can only be increased at the expense of grade and vice versa.

f) Assume that you have a typical smelter contract in which you are paid for a portion of the metal value and charged based on the mass of concentrate to be treated, and that the circuit is operating at the optimum trade-off between grade and recovery at the current metal price. How would you adjust your circuit's operation in response to an increase in the metal price and why? (4)

QUESTION 3 – Classification and Performance

A cyclone manufacturer provided you with the following partition curve for your desired feed rate.



a) What is the d_{50} and bypass ratio for this cyclone? (2)

The feed to our cyclone will have the following cumulative particle size distribution.

	Feed
Sieve size	Cumulative Passing
microns	%
212	100
150	95
105	70
74	20
53	15

b) Calculate the expected cumulative size distributions for both the overflow and the underflow. Show all working. (12)

c) What is the mass recovery to the underflow? (2)

d) As the cyclone wears, what would you expect to happen to the cut-size and the mass recovery to the underflow and why? (3)

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In minerals processing different measurements often need to be combined in order to carry out a mass balance. For a cyclone we know that the total mass flow rate (water and solids) in the feed is 100 tph and that it is 30% solids by volume. The underflow is 40% solids by volume and the overflow is 20% by volume.

e) If the solids has an SG of 2.5, what is the mass flow rate of the solids and the water in the underflow? (6)

QUESTION 4 – Metal Extraction

A porphyry copper deposit has a weathered, predominantly copper oxide, cap, with a higher grade copper sulphide region below this cap. The copper grade decreases with distance from the centre of the deposit. It is a large deposit and it has been decided to use both heap leaching as well as a concentrator in which the ore is milled followed by flotation.

a) Which material would you send to heap leaching and which to the concentrator? (3)

The mine will have 4 multi-lift leach pads that will also act as the final disposal location for the material. New material will be being added to one of the heaps while the other 3 are being irrigated, with the cycle time for each heap being 8 months (2 months of lift construction and 6 months of leaching). Material for heap leaching is being mined at the rate of 5 000 tpd and each lift is 10 m.

b) What is the average surface area of each of the heaps if the voidage is 20% and the solids have an SG of 2.6? (4)

The ore being placed on the heap has an average copper grade of 1.0% and the lixiviant addition rate is 900 m^3/hr. 5% of the liquid is lost through evaporation and other loses and the pregnant solution contains an average of 2.1 kg Cu/m^3. The solvent extraction stage recovers 90% of this copper, with the barren solution having additional acid and water added to make up for the lost volume and consumed material before being added back onto the heap.

c) What is the average recovery of the copper from the heap leaching assuming no losses in the SX/EW? (4)

The organic solvent enters the solvent extraction stage at a concentration of 2 kg Cu/m^3 and leaves at a concentration of 6 kg Cu/m^3.

d) What is the flowrate of the organic solvent? (2)

On the attached sheet I have given the equilibrium curve between the copper in the aqueous phase and that in the organic solvent.

e) How many equilibrium stages are required to achieve the desired performance? Show all working and hand in the sheet. (6)

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The main alternative to the hydrometallurgical route (and the most widely used method for producing copper) is to produce a concentrate and then to pyrometallurgically treat the concentrate.

f) What are the main stages in this route for going from a copper concentrate to a saleable copper metal and what does each stage achieve? (6)

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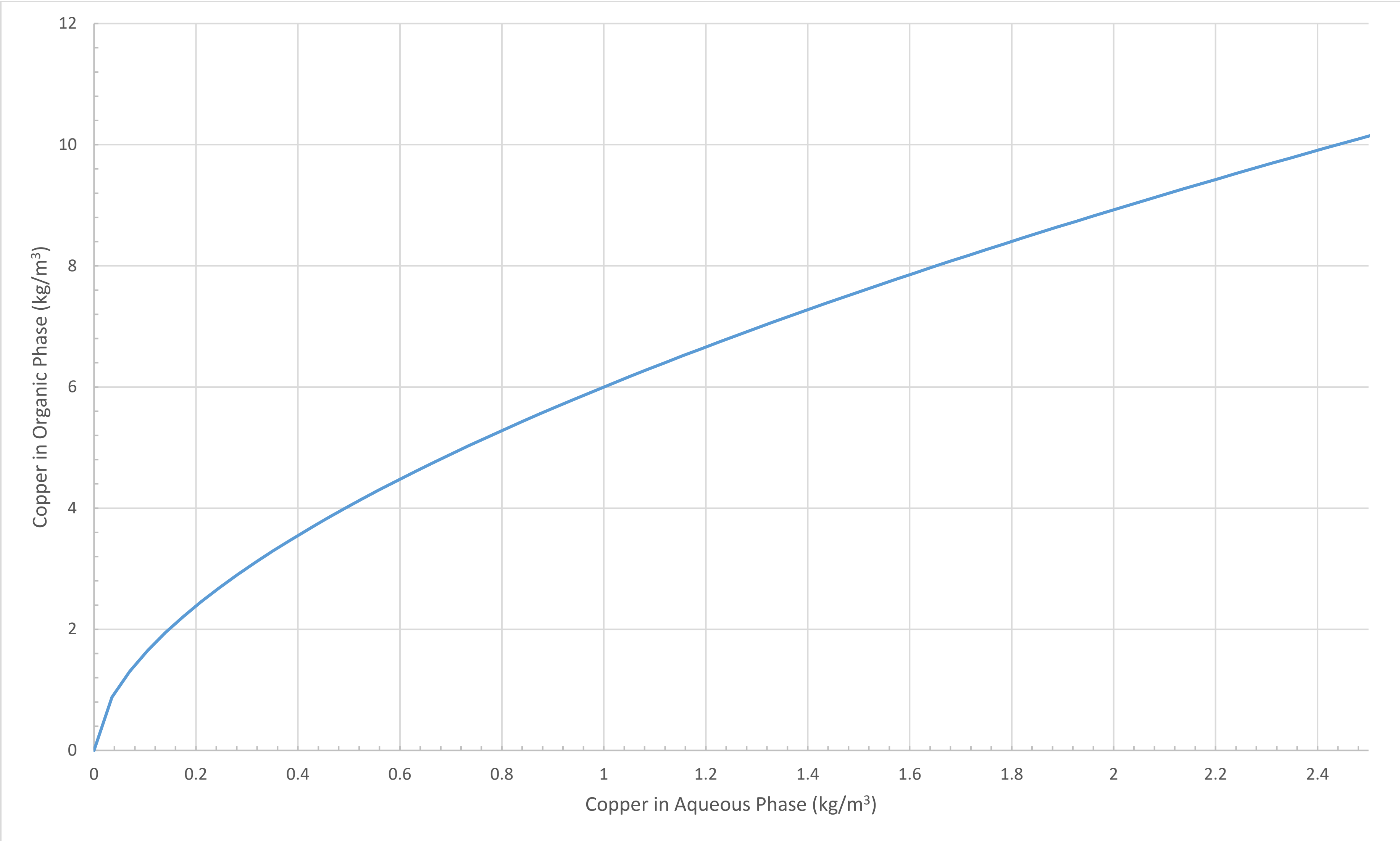

12
10
8
6
4
2
0
Copper in Organic Phase (kg/m³)
0
0.2
0.4
0.6
0.8
1
1.2
1.4
1.6
1.8
2
2.2
2.4
Copper in Aqueous Phase (kg/m³)