24/3/97

Memorandum

DATE: March 24, 1997

TO: Peter Benjamin

FROM: Nick Clarke

RE: Copper Clay Metallurgy - Preliminary Thoughts

CC: Peter Williams, Shane Polle, Ken Morrison

1. INTRODUCTION

Some work has been completed on a sample of copper clays from an outcrop at King Lyell. The work comprises wet and dry sieve sizing and cyclosizing, assay of size fractions and inspection of some size fractions under the stereo microscope. The report by Kevin Wills has also been read, and the MLMRC flotation results reported there have been summarised. This memorandum records the results of work to date, and some resulting preliminary thoughts.

2. SUMMARY

Copper concentrations in the copper clay sample analysed are highest in the coarsest fraction, and decline with decreasing size. A considerable amount of the coarse fractions consisted of agglomerated clay balls. Presumably if these were broken up, the grade of the coarser fractions would increase.

The copper appeared to be present as minerals tentatively identified as cuprite, malachite, chrysocolla, chlacocite and possibly chalcopyrite. From comments by Ken Morrison, it is assumed the cuprite was formed by oxidation of native copper. No native copper was observed. The cuprite appeared quite crystalline.

The copper minerals broke down quite readily, to particles in the range 10 - 30 microns, and possibly finer. It seems likely from this and K. Wills theories of formation that all the copper is present as discrete copper mineral, but it is possible some is adsorbed onto the clays.

Historical and current information suggests that recoveries on ore of treatable grade and type would be in the range 60 - 70% could be achieved by either gravity or flotation separation. In practice, it is likely both head grades and recoveries would be highly variable. A process of desliming and conventional gravity separation might achieve recoveries in the 60% range, based on the sample analysed. Use of centrifugal separators could improve recovery and might avoid the need for desliming. Leaching might achieve higher recoveries, and could generate saleable copper metal on site. However, difficulties with liquid-solid separation and capital cost may militate against leaching.

The viability of an operation to recover copper from the copper clays is somewhat doubtful. The grade is very low for the available tonnage. Capital costs would be substantial, since even a simple gravity recovery plant would require proper means of tailings disposal. Possibly, a plant could be established by mining the King Lyell first, and following with the Blocks.

A small amount of metallurgical testwork is proposed. It is suggested this be followed by a desk-top scoping study, to determine if an operation could potentially be feasible, and what direction further work should take.

3. RESULTS

3.1 Assays

Table 1 records the results of the size analysis and assays on size fractions. Table 2 gives copper distributions by size.

It is apparent there is rather poor agreement between calculated and measured assays. The measured head assay was 2.70% Cu, whereas the assay calculated from the size fractions was 2.44% Cu. The measured assay of the -38 micron fraction was 1.24% Cu, whereas the calculated assay from cyclosizer fractions was 0.80% Cu. Some very coarse lumps resulting from the size analysis are not included in the assays shown. The lumps were assayed later, and ran (from memory) about 10% Cu. This would bring the calculated assay up to better agreement with measured. The poor agreement between measured and calculated assays on the -38 micron fraction could be due to loss of - Cone 5 material, although much of it was recovered. If this is the cause of the difference, then the lost material would have been very low grade.

Inspection of Table 2 shows that grades are highest in the coarsest fraction, and decline steadily with size.

3.2 Mineralogy

The +106 micron fraction was selected to examine under the stereomicroscope. All grains appeared a uniform pale brown. Weak acid was applied and it was found that most of the lumps consisted of agglomerated fines. This size fraction was rewashed on a 106 micron sieve, with gentle brushing to break down the "clay" lumps, and 84% of the weight passed through the sieve. The remaining oversize material contained particles tentatively identified as chrysocolla, malachite and a dark red brown substance likely to be cuprite. Around 10% of the mass was transparent silicates (quartz?) and 20% clay lumps remained.

The material washed through the sieve contained a lot of clay material plus ?cuprite and ?malachite in all sizes down to 20 microns and finer.

The +38 micron fraction was inspected, and contained relatively little clay lump. The major species present was ?quartz, with a substantial amount of ?cuprite and rather less ?malachite. There was visibly less copper mineral present than in the +106 micron fraction.

The -38 micron dry sieve fraction was observed, and appeared to consist dominantly of ?quartz, with 20- 30% clay lump, and trace ?cuprite and rare ?malachite.

The -38 micron wet fraction was inspected. By dispersing in water, it was possible to observe minor cuprite, but the sample appeared to consist dominantly of clay lump.

No recognisable native copper was seen in any fractions. Presumably, from Ken Morrison's observations, all native copper had converted to cuprite. Since native copper can certainly be stable in air, there is presumably some reason why the native copper in the copper clays is so reactive.

Very rare grey black and very friable mineral was seen, possibly chalcocite. Rare fresh and sharp edged sulphide was also seen, and it was thought could have arisen from contamination in screening, but could also have come from the copper clays.

It seems quite likely from the assays and observation that most, possibly all, of the copper is present as discrete copper mineral and tends to be fairly coarse. However, because of the generally soft nature of the minerals, they are readily broken down to finer sizes. It is possible some copper is absorbed onto clay mineral.

3.3 Flotation

Table 3 summarises the results of flotation tests conducted by MLMRC on a number of copper clay samples. Unfortunately only one test records the reagents used, which were xanthate, a di-thiophosphate and Aero 404, which is recommended for tarnished and secondary copper minerals.

One test tried sulphidisation on a mids product without success - it is assumed from the mention of this that no other tests used sulphidisation.

Results of flotation appear to have been quite good. Three out of the four samples had about 80% of the copper in the "sand" fraction, in two cases in about 25% of the weight. Flotation recovery of the copper in this fraction was about 90%. Flotation recovery overall varied from 31% to 77%. The lowest recovery was from a sample with head grade 0.13% Cu, but one high grade sample (head grade 1.8%) also gave poor overall recovery because 54% of the copper reported to the slimes. However, nothing is known of how the sand/slime separation was done, or what the approximate split size was.

It appears from these results that a considerable variation in metallurgical response to physical separation procedures could be expected and would be driven mainly by the size distribution of the copper.

It is interesting that the recovery from Lyell Blocks ore by gravity methods was quoted at 72%.

4. PROCESS OPTIONS

A number of approaches could be envisaged for recovering copper:

a) Deslime (10 microns?) and float the oversize, potentially in the existing plant, probably more successfully with a tailored reagent regime. Maximum recovery about 70%.

b) Deslime (38 microns) and recover copper from oversize on say spirals. Maximum recovery about 60%.

c) Recover copper using a centrifugal concentrator, with or without prior desliming. Possible machines are a continuous Falcon or Knelson concentrator, Kelsey jig or Mosley Multi-gravity concentrator. The latter two are quite expensive, but cannot be ruled out at this stage. Maximum recovery unknown, but probably about 70% on the ore sample examined.

d) For the above three approaches, very coarse material might go directly to concentrate, and the concentrate could probably be mixed with existing sulphide flotation concentrate for sale and treatment.

e) Recover the copper by leaching. Acid leaching is the simplest approach, but given the presence of native copper and/or cuprite, will probably be unsuccessful. Ammonia/ammonium carbonate leaching would probably be technically successful - in fact almost ideally suited to this ore. Leaching followed by solvent extraction and electrowinning would have the benefit of producing copper on site. The biggest difficulty I think would be liquid-solid separation - desliming to eliminate clay fines would probably be necessary. Other problems would be capital cost and ammonia recovery costs. Recovery could be >90% if the clay fraction is leached, but more likely about 70% with desliming and direct discard of the clay.

I have little doubt from data available now that most or all of the above methods would be technically feasible. The highest recovery would likely be from leaching, and the lowest from desliming and spirals. Cost would probably be in the same order as recovery. The big question is whether any of these techniques could be economically viable. King Lyell would obviously be the best prospect on the basis of grade, but with very little tonnage available to justify an operation.

5. VIABILITY

The combination of low grade and low tonnage for the copper clay resource means that it is likely to be difficult to mine and treat economically. Capital and operating costs for a small plant would be high, and the best chance would probably be if enough tonnes could be proved up for say an operation treating 1 million tpa over 5 years.

On the downside, capital cost would be significant, even for a simple deslime and spiral or float type operation. The simplest flowsheet would involve perhaps a log washer or agitator to disperse the clays in water, a screen to remove coarse material and trash, spiral concentrators for recovery, possibly a thickener and either a tailings dam or a pipeline to the existing tailings dam. Addition of flocculant and possibly a clarification agent would likely be necessary to produce a clear overflow for discharge to the environment or re-use in the plant.

Taking account of the cost of tailings disposal, the cost of $1.50/t for copper recovery seems too low, for a small scale operation. For a resource of say 600,000 t, and a throughput of say 200,000 tpa, 1 shift operator represents a cost of $1/t ore.

The cost of copper transport and recovery will depend very much on concentrate grade if the copper is sold as a concentrate. However, it is important to note that the cost of realisation depends on the amount of copper in the ore, so low grade ore has low realisation costs.

On the upside, perhaps mining costs could be reduced below those quoted by Kevin Wills. Would it be possible to recover clay using a floating dredge/pump and so perhaps reduce the stripping ratio?

There is some possibility that grades in drill holes could be underestimates, given the relatively coarse size of the copper. The sludge assays quoted by Kevin suggest that lost material might be higher in grade. If coarse and dense copper occurs in a clay matrix, is the copper more or less likely to be recovered in drilling than the clay?

However, it seems unlikely that actual grades would be much higher than those estimated from drill holes, since the Lyell Blocks operation was presumably trying to follow high grade material, and head grades declined from 4.3% at the start of operations to 1.5% by the end. Still, 1.5% is substantially higher than 0.5% Cu.

For the Blocks, there also seems to be a reasonable chance of some gold and silver which would be recovered in a gravity or flotation process - the amount seems highly variable. If we assume 0.2 g/t Au and 3 g/t silver, that would add about $4/t to the ore value.

6. RECOMMENDATIONS

6.1 Metallurgical Testwork

I suggest the following approach:

1) Re-screen both surface exposure and drill samples by wet sieving with gentle brushing to break up clay lump. Brushing should be continued until the water runs clear. To avoid damaging the sieve, this could be done on say a 75 micron sieve, then the undersize poured through a 38 micron sieve. The +38 micron fraction should be dry screened on 600, 300, 150, 75 and 38 micron screens. Assay a portion of the -38 micron combined dry and wet fraction.

The combined -38 dry + wet fraction of undersize should be carefully dispersed in water in a 1 litre beaker, with say 100 mm of water depth. Disperse avoiding swirling, then allow to settle for 120 minutes (+- 10 minutes OK). Decant off the suspended solids, but be careful to avoid disturbing the settled solids. Leave about 1 cm of water in the beaker. Make up to 10 mm water depth, and repeat. Repeat again. This should eliminate most material finer than around 4 microns. Cyclosize, and collect the first dustbin full of cyclosizer - cone 5 product, then discard the remainder of the - cone 5. Cones 1-3 and 4 & 5 may be combined. Assay all sieve fractions, cone 1-3, cone 4-5, -cone 5 cyclosizer and beaker decantation fines separately, for copper only. Also make up samples of the combined sieve size fractions, cones 1-5 and -5 combined, and beaker decant solids and assay for gold and silver.

2) Prepare a sample of surface exposure and drill "core" material for gravity separation , by the same procedure as used for preparing the cyclosizer feed sample. Send settled and decanted solids to Central Mineralogical Services for Superpanning and microscopic examination.

3) Leach a sample of unsized surface ore in sulphuric acid. Record the acid consumption and measure the copper extraction.

4) Leach a sample of unsized surface ore in acid drainage liquor (say from sample point H3 & H4). If necessary, supplement the AD with sulphuric acid.

5) Leach a sample of unsized surface ore in ammonia/ammonium carbonate under atmospheric conditions.

6) Deslime about 2 kg of surface exposure ore by decanting from a bucket (allow 10 minutes settling time per cm of water depth, and repeat the decantation operation 3 times). Assay and discard the suspended solids. Carry out a flotation test on the settled solids. It may be necessary to obtain a suitable reagent from Cytec, depending on what we have available.

The above testwork will indicate what methods of concentration could be viable, and what recovery might be expected.

6.2 Scoping Study

Before proceeding further, I suggest a desktop scoping study should be carried out, to see whether a project could potentially be feasible.

At this stage I would suggest that for the project to have a chance, and to have a worthwhile impact on cash flow, it would be necessary for the copper clays in the Blocks at least to be economic to treat, so that a reasonable plant capacity could be established based on mining King Lyell first, then the Blocks.

6.3 Drill Core Loss

Would it be possible to establish the effect of core loss on drill grade by drilling a short hole in accessible copper clays, then confirming grade from a bulk sample?

Nick Clarke

TABLE 1: Size Analysis

Size Analysis on:	Cu Clay Sample 1 - King Lyell Outcrop
Sampled:	

Cyclo Cone	Sieve Size Microns	Wgt g	Cu %	Zn %	Pb %	Wgt Held %	Cum Wgt Pass %
	850					0	100.00
	600	5.09	5.80	0.50	0.017	2.56	97.44
	425	9.27	5.80	0.50	0.017	4.67	92.77
	300	10.17	5.80	0.50	0.017	5.12	87.65
	212	5.79	5.80	0.50	0.017	2.91	84.74
	150	0.00	0.00	0.00	0.000	0.00	84.74
	106	8.58	6.35	0.48	0.015	4.32	80.42
	75	0.00	0.00	0.00	0.000	0.00	80.42
	53	8.58	6.85	0.43	0.011	4.32	76.10
	38	3.63	3.65	0.39	0.011	1.83	74.27
	-38	147.52	1.24	0.36	0.016	74.27	-
Cone 1	45.2	-	1.23	0.26	0.010	0.44	73.83
Cone 2	33.1	-	1.23	0.26	0.010	1.01	72.82
Cone 3	23.7	-	1.23	0.26	0.010	4.31	68.51
Cone 4	17.2	-	0.67	0.22	0.007	6.24	62.27
Cone 5	12.3	-	0.67	0.22	0.007	6.20	56.08
- Cone 5	-12.3	-	0.79	0.42	0.019	56.08	-
Calc -38			0.80	0.37			
Total Calculated		198.63	2.44	0.39	0.016	100.00	-
Total Measured		200	2.70	0.40	0.015		

Wet Sieve Weights	
Head g	200
+38W g	55.27
-38W g	143.56
-38D g	3.96
Checks:	
-38 Dry	3.96
Total	
Dry Calc	55.07
Dry Meas	55.27
+38 Wet	55.27
-38 Wet	143.56
Total	
Wet Calc	198.83
Wet Meas	200.00

Cyclosizer Data	Wgt g	Wgt %
Cone 1	0.31	0.590
Cone 2	0.72	1.358
Cone 3	3.08	5.807
Cone 4	4.45	8.400
Cone 5	4.42	8.342
-Cone 5	-	75.504
Head	52.98	100.000

Particle S.G.	2.7
Elutriation Temperature	11
Elutriation Flowrate	180
Elutriation Time, mins	30

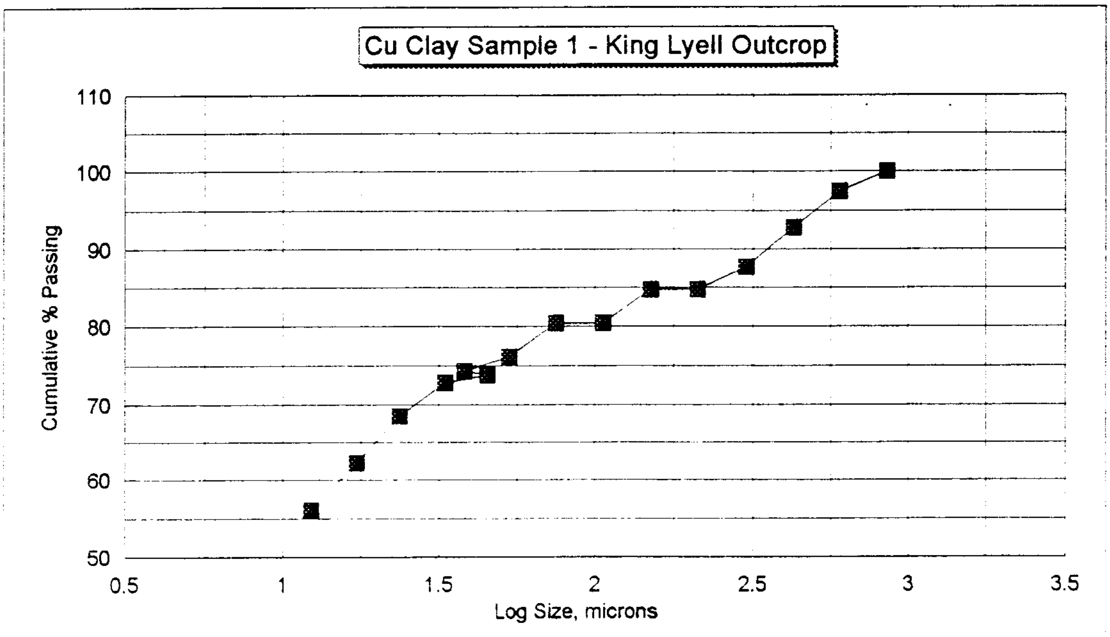

TABLE 2: Copper Clay King Lyell Outcrop Sample

Sieve Size Microns	Weight % Held	Cu %	Cu Units	Cu Dist %	Corr Cu Dist %	Cum Cu Dist %
212	15.26	5.80	88.51	36.23	36.23	36.23
106	4.32	6.35	27.43	11.23	11.23	47.46
53	4.32	6.85	29.59	12.11	12.11	59.57
38	1.83	3.65	6.68	2.73	2.73	62.30
-38 Measd	74.27	1.24	92.09	37.70	37.70	100.00
-38 Calcd	74.28	0.80	59.72			
23.7	5.76	1.23	7.08	2.90	4.47	66.78
12.3	12.44	0.67	8.33	3.41	5.26	72.04
-12.3	56.08	0.79	44.30	18.13	27.96	100.00
Total Calc	100	2.44	244.31	100.00		
Head Assay	100	2.70				

TABLE 3: Summary of MLMRC Flotation Results on Copper Clays

Sample	Head % Cu	Sands						Slime				
		% Cu	% Wgt	Cu Dist %	Cu Rec Sand %	Cu Rec Total %	Conc Cu %	% Cu	% Wgt	Cu Dist %	Cu Rec %	Conc Cu %
Native Copper Clays	2.3	6.67	26.53	77.60	91.49	71.00	26.81	0.70	73.47	22.40		
Cu Blocks Float II	0.1	0.13	50.30	57.30	55.15	31.60	0.86	0.09	49.75	43.01		
Consols 3	0.9	1.78	22.70	86.72	89.46	41.60	17.30	0.61	77.30	53.50		
Cu Blocks V	0.3	0.34	64.80	84.80	91.27	77.40	10.48	0.11	35.20	15.20		
King Lyell Outcrop (1996)	1.4	5.91	25.70	62.30				1.24	74.30	37.70		