The Marvel Loch Crusher Product Sampling System

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ABSTRACT

The Marvel Loch Operations (MLO) plant was treating ores from up to six MLO sources and in addition needed to blend externally sourced ores with its own ores. This presented a metallurgical accounting dilemma, which was subsequently addressed by the design, implementation and testing of an automatic sampling system.

The MLO automatic sampler was well designed and has been the subject of previous internal and external reviews. The sampling action is fully automated and the unit is essentially self cleaning particularly when sampling primary ores. The purpose of this paper is to document the rationale behind the design, the specific details of the sampling station, its use for ore purchase purposes and an assessment of the sampling system to meet the needs of the operator.

INTRODUCTION

The Marvel Loch Operations were owned and operated by Sons of Gwalia until the operations were purchased by St Barbara Limited (SBM) from the SOG administrators in March 2005. The MLO plant processed ore for SOG from a variety of mines in the Marvel Loch/Southern Cross area. The Cornishman Joint Venture (CJV) entered into an agreement with SOG for processing of CJV ore on an 'ore purchase basis'. The first batch of CJV ore was processed in 2003; the following information is based on ore processing during 2003.

The Cornishman Joint Venture Ore (CJV) produced small tonnages that needed to be blended at MLO because of the high throughput of the MLO mill. Metallurgical accounting of the CJV ore in the blend therefore required sampling prior to blending to accurately determine the head grade, metallurgical recovery and tonnage. This required the engineering, design, procurement and construction of an automatic sampling system and separate fine ore bin to account for the ore processed.

CJV ORE was batch crushed to Fine Ore Bin 3. The automatic sampling system was used to collect composite samples of CJV ore which were prepared on site by MLO personnel. The samples were then tested at an off-site laboratory to assign a head grade and a plant recovery at a nominated grind size.

CALCULATED AND ASSAY GRADE

Mill calculated grade is based on the fine gold poured plus the change in circuit inventory plus gold to tailings. The mill calculated grade is a 'real number' and should agree within plus or minus three per cent of the assay grade for acceptable standards of accounting to be achieved.

If there are differences outside this range, it follows that sampling, assaying or the procedures, need improvement. At sites where the gold is fine, this is achieved very easily, whereas where there is a significant amount of coarse gold it is more difficult and special procedures may be required.

The mill assay grade was based on two-hourly grab samples of the cyclone overflow which were individually assayed and the daily gravity gold added back in to provide a feed sample assay. During each shift the cyclone overflow assay is available but a 'gravity factor' is used to add back in the gravity gold which makes the daily assay subject to some error. The gravity factor can be variable because of the nature of the concentrates.

It is not a simple task to take reliable mill feed assay samples because of the sample size required and the influence of the 'nugget effect'. Thus, this is the reason, as a matter of course, that the leach feed sample is used and the actual gravity gold recovered is added back to provide a mill feed assay.

The grade control assay is based on blasthole sampling. Due to the limited time between blasthole drilling and delivery of ore to the plant, a crude kriging method is applied and ore block model grades assigned. Site geological and metallurgical personnel have indicated that the crude nature of this process may result in overestimation of the grade.

The tailings assay grade was based on two-hourly grab samples of the tailings stream from the final adsorption tank which were also individually assayed to provide an assessment of unrecovered gold for calculation of the 'mill calculated grade'.

RECOVERY CALCULATION

The CJV period composite was processed in duplicate with a gravity leach at an offsite Perth laboratory. The procedure involved taking a CJV composite and a mill feed composite and subjecting them to the same gravity leach procedure. This allowed for an adjustment to be made to the CJV ore recovery

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if the actual plant recovery was higher or lower than the plant test recovery (Figure 1). If there was significant variance in the duplicate tests, they were repeated.

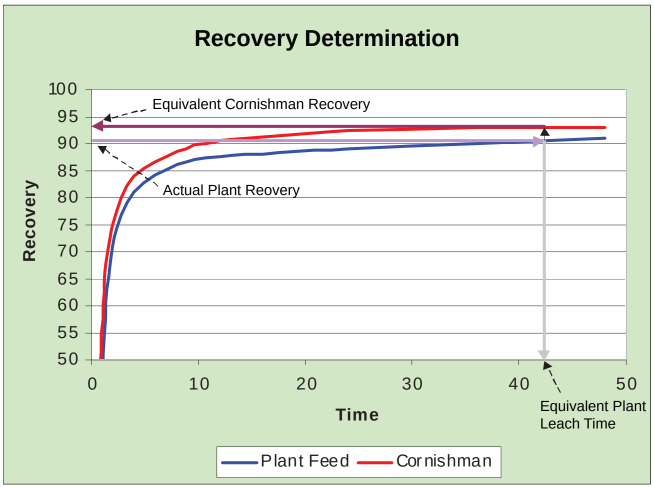

FIG 1 - Gravity leach recovery results.

The recovery was factored to:

$$R_{jv} = R_{jvtest} . R_{plant} / R_{plant\ test}$$

ie if the CJV test recovery is 96 per cent and the mill feed test recovery is 93 per cent and the actual plant recovery is 92 per cent, then the recovery allocated to the CJV is:

$$R_{jv} = 0.96 \times 0.92/0.93 = 95\%$$

This was mindful that the plant recovery could be higher or lower than the laboratory test recovery.

These tests were carried out at a fixed residence time (36 hours), fixed grind size and fixed reagent conditions.

MILL THROUGHPUT

The feed tonnage allocated to the CJV was calculated as the percentage of ore sourced from the CJV fine ore bin (FOB 3), as a ratio calculated from the shift weightometer readings from FOB 1, 2, and 3, multiplied by the mill feed conveyor shift tonnage plus any CJV ore reclaimed through the emergency feeder, ie:

$$[T_{FOB3}/(T_{FOB1} + T_{FOB2} + T_{FOB3})] \times \text{mill feed tonnes} + \text{emergency reclaim}$$

SAMPLING

The Marvel Loch automatic sampler was well designed and has been the subject of previous internal and external reviews. The sampling action is fully automated and the unit is essentially self cleaning particularly when sampling primary ores.

A linear cross-cut sampler (Model 1000 ARM Sampler) was used for primary sampling of the final crusher product on conveyor CV-05. The primary sample was directed by a chute to a sample cone crusher to reduce the particle size. The product from the sample crusher was then directed to a rotary vezin sampler and the secondary sample cut was collected in a rotary sample collector with four containers which indexed automatically. The secondary sampler reject material was returned to the crushing circuit. Operation of the sampling system was controlled through Citec and timing of the primary samplers and the sample collector could be altered to suit sample requirements (Figure 2).

The sampling station is viewable in Figures 3 and 4. The sampler was regularly cleaned and all sample transfer and collection points were free from contamination by clean up solids and water. The primary cutter is shown to have collected some material between the inner and outer liners. The mass of this material and the fact that the sampler was rarely used for any material other than the CJV ore would limit the possibility of contamination. This build-up also reduces the sample cut

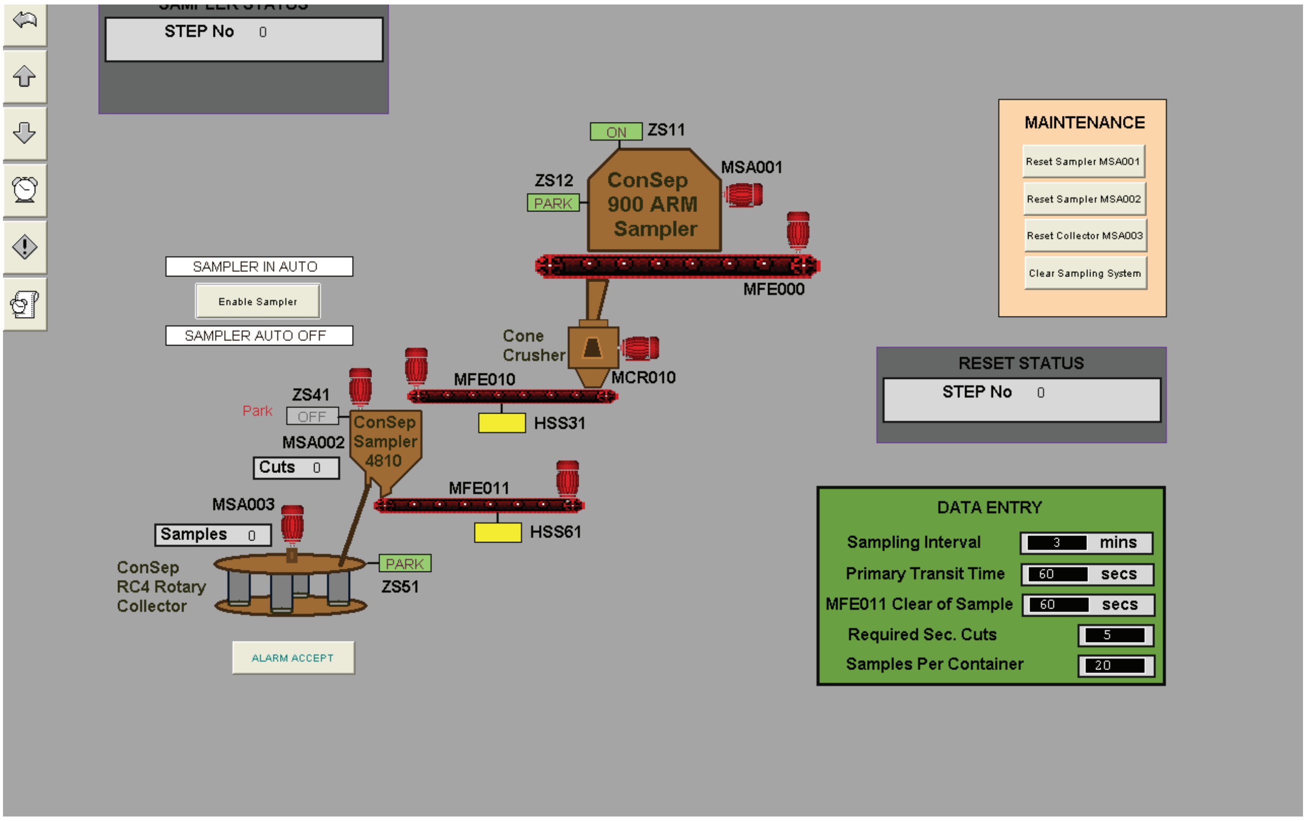

FIG 2 - Sampler Citec display.

FIG 3 - (A) Sample plant overview; (B) primary cutter; (C) primary sample point; and (D) primary sample transfer chute.

size; however, it is still significantly greater than the minimum width of three times the maximum particle size.

Initially it was noted during the operation of the primary sampler swing arm, that approximately ten to 20 per cent of the cut was not removed from the conveyor belt. If there is size segregation on the conveyor belt, and a bias in the gold distribution towards either the fine or coarse size fractions, it is possible there will be an error in the sampling. Site metallurgical personnel were aware of this issue and have taken belt cut sizings and compared these to the primary sampler swing arm sizing. They have also taken assay by size fraction of the belt cut sample and the primary sampler swing arm sample. The results showed no significant difference in the sizings for the two samples. Similarly the results showed no relationship between size and grade. The inconclusiveness of the sizing and assay by size fraction data is related to the variance. To minimise this variance and provide statistically significant evidence that the residual sample not being removed from the belt is or is not causing an error in the sampling would require an extensive sampling campaign. It is the authors' opinion that it is unlikely that the incomplete sample collection by the primary swing arm is likely to result in any sample assay bias.

In general the physical condition of the sample plant, maintenance and operation was acceptable and not likely to be causing any bias in the samples taken.

FINE ORE BIN

The CJV ore was fed exclusively into fine ore bin (FOB) number three (Figure 5). Prior to the shutdown of the crushing circuit when treating CJV ore, the conveyors and all surge bins are run completely empty and all chutes and bins inspected. Once all the ore has been run out of the circuit, the tripper was reset to feed into FOB 1 or FOB 2, and the loader driver instructed to begin feeding Marvel Loch ore into the Primary Crusher. The reverse of this process occurs at the start of the processing of the CJV ore.

The normal process was that once the CJV FOB reaches +90 per cent full, the shutdown sequence commences. If FOB 1 and 2 are not available, CJV ore was drawn from the bottom of FOB 3 and stockpiled in an isolated area. This stockpiled ore was fed back through the emergency feeder when required. When the ore was fed through the emergency feeder, a weightometer reading was taken at the start and end and tonnage allocated to the joint venture.

Whilst onsite, changeover procedures and emergency reclaim procedures were observed on day shift and night shift. Operator understanding of the importance of maintaining separation between the CJV and other ores was acceptable and procedures were followed, ensuring all tonnages were correctly allocated.

FIG 4 - (A) Sample crusher; (B) secondary sampler housing; (C) sample collection pots, and (D) secondary vezin cutter.

FIG 5 - Fine ore bins, Cornishman Joint Venture (on right).

WEIGHTOMETERS

Figure 6 illustrates the layout of the weightometers in the fine ore storage section of the SOG Marvel Loch plant.

There are three feeders that all discharge onto CV-4. At the beginning and end of each shift weightometer readings on the three feeders and CV-4 are recorded. Tonnes allocated to the joint venture based on the ratio of ore are drawn from the FOB 3 feeder multiplied by the tonnes on CV-4. Relative accuracy checks can be performed by isolating two of the three feeders from the fine ore bins and comparing the weightometer reading to that on CV-4. Any ore fed through the emergency reclaim bin is recorded separately and allocated to the relevant source.

Whilst onsite feeders from FOB 1 and 2 were isolated to check the correlation between the weightometer on the feeder from FOB 3 and the weightometer on CV-4. The two weightometers had a variance of approximately one per cent when observed. This is considered acceptable.

The weightometer on CV-4 was calibrated and serviced monthly. The weightometers on the feeders from the fine ore bins were compared to the weightometer on CV-4 monthly and if significant variance was noted they were also calibrated. Span and zero errors were within two per cent which was considered acceptable.

Load cells weighing the ore hauled from the Cornishman ROM pad to the Marvel Loch ROM pad provided an additional check ensuring ore was not misclassified. During the processing of Batch 99 the variance between the hauled and crushed ore was less than three per cent.

In general the operation of the Marvel Loch laboratory was very good. Sample preparation and processing procedures were followed as described, and equipment was maintained in good condition and the possibility for contamination was minimal.

CJV samples were transferred to the laboratory at the end of each shift by the crusher operator. Sample ovens and processing equipment was observed to be well maintained and adequately flushed and cleaned prior to pulverising and splitting.

The sample preparation area is shown in Figure 7. The sample preparation procedure (Figure 8) was observed to be followed correctly and there was minimal potential for sample contamination.

FIG 7 - Sample preparation area.

ASSAYS

Assay grades for Batch 99 are shown in Table 1. Sample variance for individual shifts was significantly above the 15 per cent tolerance level; however, averaged out over the whole program the variance was 13.4 per cent.

The Marvel Loch laboratory assay procedures were observed to be as per procedures in the CJV agreement. Discussions with the site metallurgical staff indicate that grades from fire assay are consistently returning higher results than site aqua regia digest results. This inconsistency was considered to be real and fire assay of aqua regia (AR) digest residues were undertaken to determine if incomplete digestion was the cause. The results are shown in Table 2.

In the checked samples, acid digest was leaching around 94 per cent of the gold in the feed samples and 83 per cent of the gold in the tailings samples. This is not uncommon in laboratory acid digest of ore samples. This was an area where the confidence in the gold allocation could be improved.

The Perth laboratory assay procedures processes a 3 kg sample by removing the gravity gold, leaching the gravity tail, then fire assaying the leach tail. The Perth laboratory gravity leach results return on average, a slightly higher gold grade than the site aqua regia digest. Historically the correlation was improving with the exception of period 10 (Figure 9) where the Perth laboratory grade was more than 1 g/t higher than the site AR digest grade.

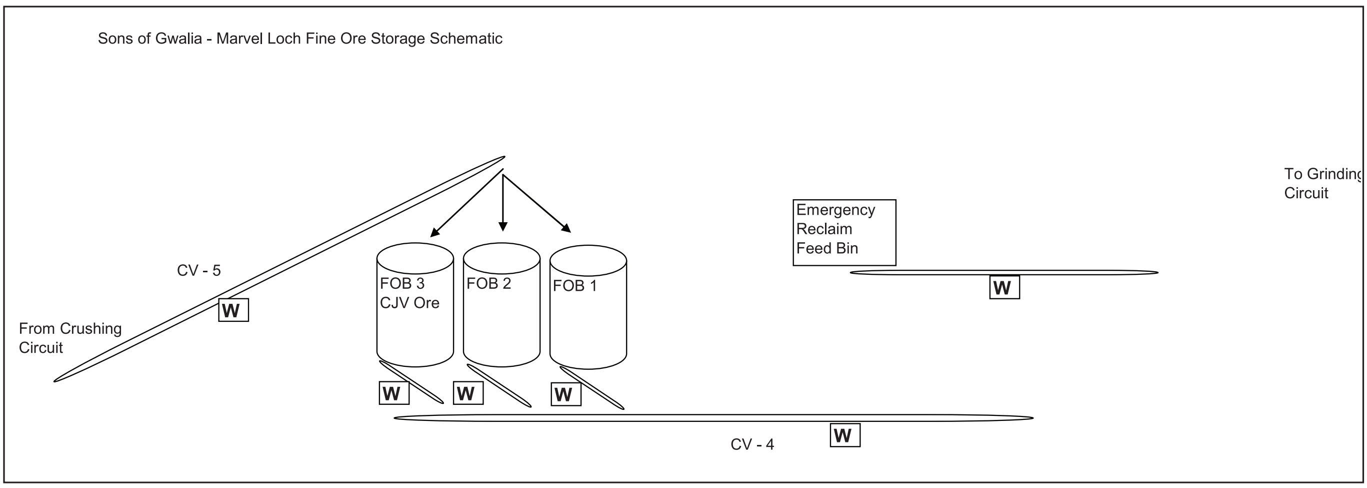

FIG 6 - Marvel Loch weightometer configuration.

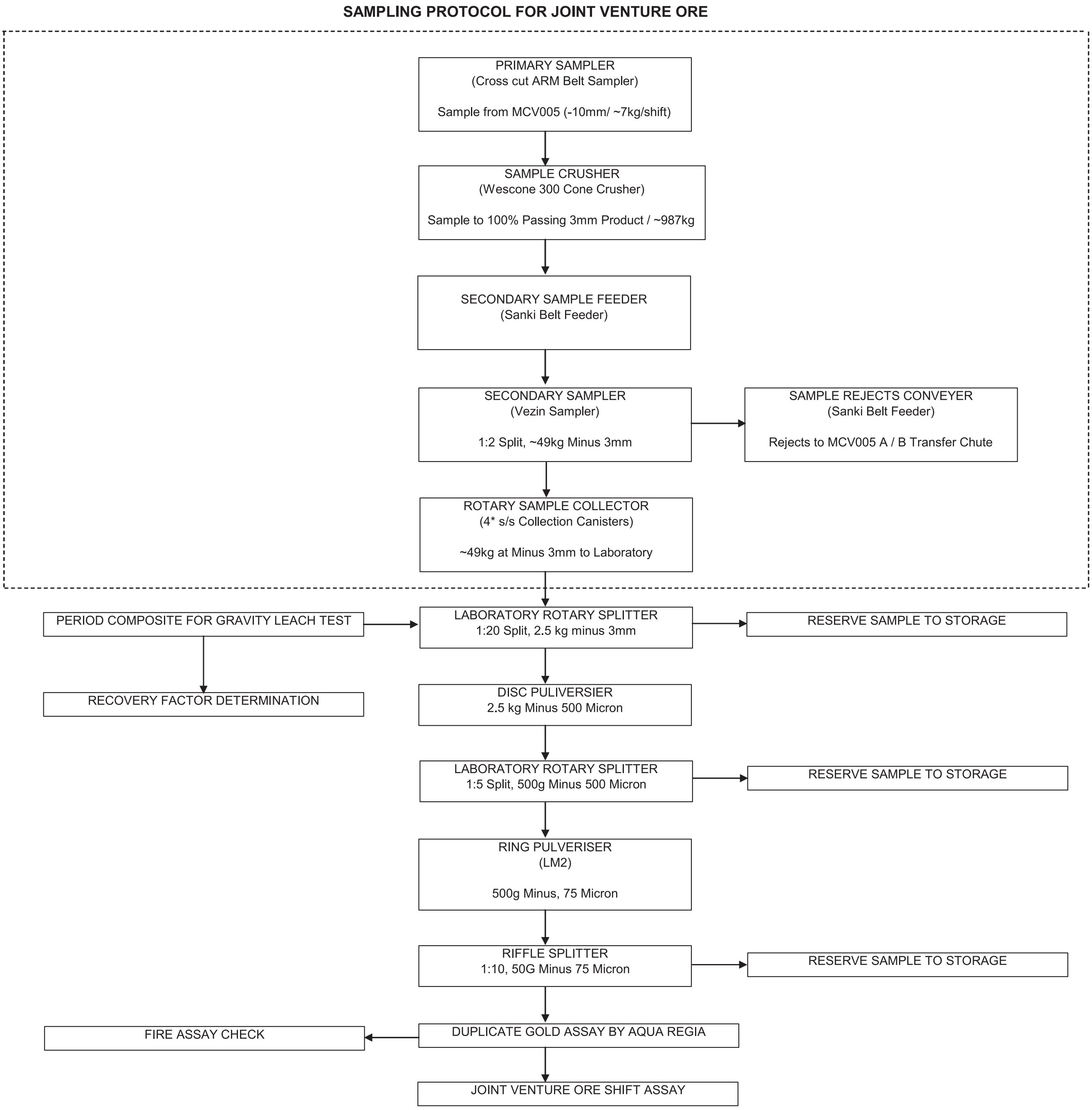

FIG 8 - Sample preparation protocol for Cornishman Joint Venture ore.

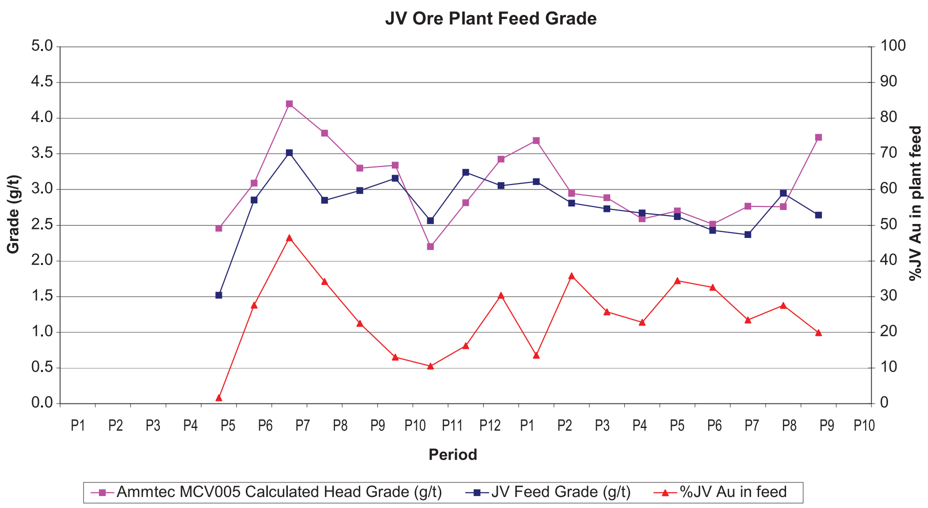

FIG 9 - Gravity leach versus site acid digest assay.

TABLE 1
Batch 99 shift assay grades.

Shift	Tonnes	Average	Assay 1	Assay 2	Assay 3	Assay 4	Assay 5	Assay 6	% Variance
15/4 DS	1349	2.83	3.25	2.53	3.53	2.33	2.24	3.12	8.8
17/4 DS	1096	3.08	2.44	3.82	3.21	2.45	3.74	2.83	15.3
17/4 NS	1502	2.50	3.62	2.01	2.49	2.57	2.19	2.11	9.7
18/4 DS	1703	4.31	4.40	3.89	4.16	3.34	4.58	5.47	11.6
19/4 DS	1632	3.31	2.12	3.18	4.23	2.39	2.41	5.55	84.4
22/4 DS	2267	3.04	3.10	2.78	2.09	4.41	2.79	3.11	18.9
24/4 NS	349	2.97	2.94	3.00	2.67	3.30	3.19	2.75	2.0
25/4 DS	481	2.97	2.94	3.00	2.67	3.30	3.19	2.75	2.0
26/4 NS	1471	4.49	4.77	4.19	4.11	4.70	5.59	3.59	10.0
28/4 DS	1330	4.48	4.06	4.89	5.52	4.28	3.62	4.52	10.9
29/4 DS	1062	5.93	5.59	6.81	5.79	5.40	6.03	5.98	4.3
29/4 NS	1311	2.43	2.57	2.21	2.81	2.29	2.38	2.30	1.9
30/4 NS	1610	3.13	3.08	3.38	2.85	2.52	3.23	3.75	5.9
1/5 NS	1289	3.86	3.05	3.68	4.07	4.09	3.88	4.38	6.9
2/5 NS	827	3.11	3.77	2.41	4.35	2.75	2.44	2.98	16.2
3/5 DS	1126	3.81	3.49	4.20	3.28	3.98	3.28	4.63	8.7
3/5 NS	810	3.81	3.49	4.20	3.28	3.98	3.28	4.63	8.7
7/5 NS	1171	3.05	2.78	2.47	2.98	2.85	4.33	2.89	15.3
Total	**22 386**	**3.51**							**13.4**

TABLE 2
Fire assay of acid digest residues.

Sample	Solid AR assay	Residue fire assay	Total	% Leached
CJV Head	3.873	0.279	4.152	0.93
CJV Head	2.527	0.188	2.715	0.93
CJV Head	3.446	0.191	3.637	0.95
Tail	0.154	0.029	0.183	0.84
Tail	0.117	0.020	0.137	0.85
Tail	0.108	0.026	0.134	0.81

If the CJV grade was consistently being underreported, the mine call factor would increase with the amount of CJV treated and there would be a relationship between the per cent CJV ore in the feed and the mine call factor.

As shown in Figure 10, there was no relationship between the mine call factor and the percentage of CJV ore in the mill feed. This supports the belief that the CJV grades being reported were true and accurate.

PERTH LABORATORY RECOVERY DETERMINATION

The Perth laboratory was an independent commercial laboratory with quality assurance standards. Its facilities were not inspected. Assay and recovery test variance is shown in Table 3.

METALLURGICAL ACCOUNTING

Metallurgical accounting practices are consistent with the JV agreement (see Table 4 for elaboration). There was a consistent positive variance in the calculated grade against the assayed grade. This means that the gold poured at the end of each month exceeded that estimated from the head grade minus the tail grade assay multiplied by the tonnes milled. This is shown in Figure 11.

This variance was consistent and independent of the percentage of Cornishman ore in the mill feed. This would indicate that the method for analysing for gold in the feed was under reporting. This was consistent with the variance between the fire assay grades and aqua regia digest grades and the gold grades of the aqua regia digest residue, however inconsistent with Figure 11 with mine call factors being independent of per cent CJV ore in the feed.

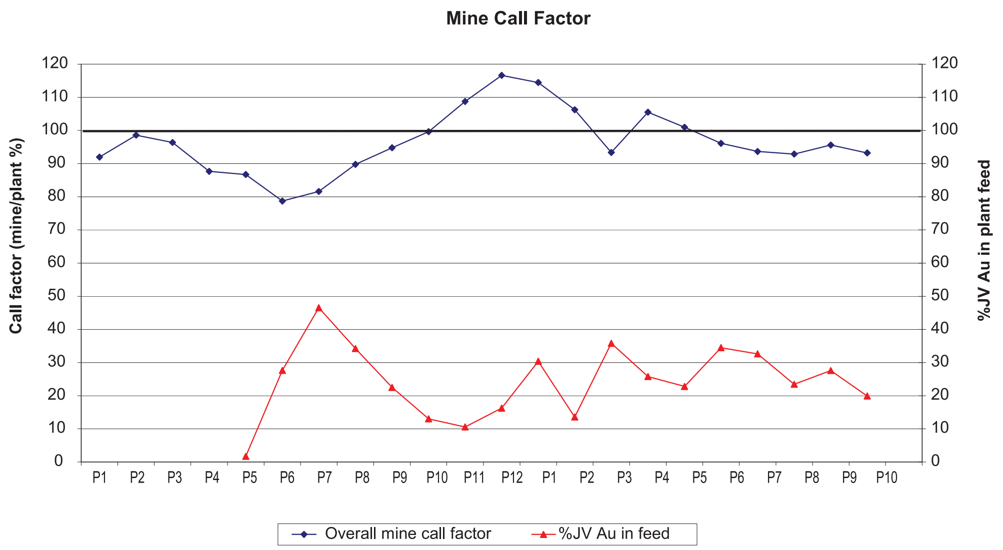

FIG 10 - Mine call factor versus percentage Cornishman Joint Venture in mine feed.

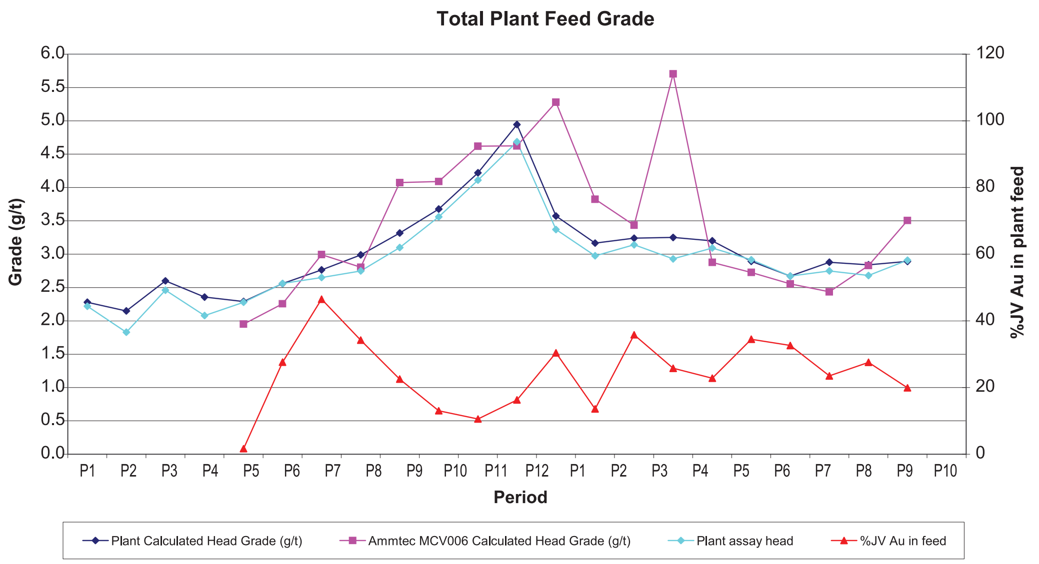

FIG 11 - Metallurgical accounts head grade comparison.

TABLE 3

Perth laboratory gravity leach variability.

Cornishman Joint Venture composite sample						
	Period 8		Period 9		Period 10	
	Head (g/t)	Recovery (48 h)	Head (g/t)	Recovery (48 h)	Head (g/t)	Recovery (48 h)
MCV005 Test 1	3.52	95.38	2.57	96.39	2.85	95.76
MCV005 Test 2	3.94	96.28	2.95	97.29	2.68	95.10
Average	3.73	95.83	2.76	96.84	2.77	95.43
% Variance	2.4	0.4	2.6	0.4	0.5	0.2
Mill feed composite sample						
	Period 8		Period 9		Period 10	
	Head (g/t)	Recovery (48 h)	Head (g/t)	Recovery (48 h)	Head (g/t)	Recovery (48 h)
MCV006 Test 1	3.63	94.92	2.62	95.07	2.48	93.23
MCV006 Test 2	3.36	96.71	3.04	95.04	2.39	94.80
Average	3.50	95.81	2.83	95.05	2.43	94.02
% Variance	1.2	1.7	3.3	0.0	0.2	1.3

TABLE 4
Metallurgical balance.

2003 - 2004	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8	Period 9	Period 10	Period 11	Period 12	YTD
Dry tonnes crushed	62 219	22 873	92 996	51 010	51 130	93 107	72 842	51 483	49 880	42 137	50 338	66 309	706 324
Wet tonnes milled	62 588	24 387	90 807	53 734	53 821	94 666	72 630	54 216	55 465	37 459	53 036	64 554	717 363
Moisture	3.69	2.84	3.63	4.68	4.36	4.92	3.97	3.46	3.10	2.08	3.39	7.14	4.10
Dry tonnes milled	60 359	23 713	87 630	51 330	51 570	90 227	69 855	52 403	53 800	36 697	51 298	60 252	689 135
Average grade	3.06	3.11	2.81	2.73	2.67	2.622	2.428	2.368	2.947	2.642	3.205	2.476	2.73
Test work recoveries													
Benchmark rec JV %	96.96	94.09	95.27	95.66	93.88	96.14	95.82	93.75	96.18	95.59	95.86	94.53	95.42
Benchmark rec all ore %	96.74	95.45	95.24	95.99	94.19	95.21	92.60	91.06	94.42	95.10	93.37	93.77	94.43
Actual overall plant rec %	94.67	94.07	93.73	94.49	94.13	94.03	92.97	93.85	93.62	92.05	91.33	90.72	93.34
Calculated JV plant rec %	94.88	92.72	93.75	94.16	93.82	94.95	96.20	96.62	95.37	92.53	93.76	91.46	94.31
JV allocated ounces	5625.18	2199.91	7419.62	4245.16	4147.09	7222.26	5245.60	3854.85	4861.26	2884.18	4955.94	4386.93	57047.98
Estimated ounces	5501.67	2201.7	7344.2	4184.0	4102.04	7058.43	5082.2	3718.32	4750.83	2905.2	4926.4	4470.3	56245.2579
Adjustment	123.51	-1.78	75.41	61.20	45.05	163.83	163.38	136.53	110.43	-20.99	29.50	-83.33	802.72
Adjusted ounces rec	5801.59	2323.42	7417.84	4320.57	4208.29	7267.31	5409.43	4018.23	4997.79	2994.61	4934.94	4416.43	58110.44
Estimated rec %	92.80	92.80	92.80	92.80	92.80	92.80	93.20	93.20	93.20	93.20	93.20	93.20	92.98
Check JV ounces rec	5501.67	2201.69	7344.20	4183.96	4102.04	7058.42	5082.22	3718.32	4750.83	2905.17	4926.44	4470.26	56245.22
JV ounces in feed	5928.52	2372.51	7914.01	4508.58	4420.30	7606.06	5453.03	3989.61	5097.46	3117.14	5285.88	4796.41	60489.50
Check	5928.53	2372.51	7914.02	4508.58	4420.30	7606.07	5453.03	3989.61	5097.46	3117.14	5285.88	4796.42	60489.55

Note: rec = recovery

CONCLUSION

Mixing gold ores has been avoided in the past due to metallurgical accounting risks. The design of an engineered sampling system allows the head assay to be determined with a high degree of certainty. Cross checks and balances provide reassurance that integrity is maintained when mixing the ores and processing together.

ACKNOWLEDGEMENTS

The authors would like to thank Troy Resources, St Barbara Limited, all colleagues, metallurgists at various sites, METS staff and other consultants for their contribution and the management of METS for their permission to publish this paper and constructive criticism of various drafts.

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