GOLD GRAVITY RECOVERY AT THE MILL OF LES MINES CAMCHIB., CHIBOUGAMAU, QUÉBEC

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Abstract

The gravity circuit at Les Mines Camchib Inc. consists of two pinched sluices to bleed a feed to one of two 76 cm (30") Knelson concentrator. The Knelson concentrate is upgraded to smeltable grade with a smaller Knelson (18/30 cm) and a riffleless table. Their tails are column cyanided and washed prior to recycle to the grinding circuit. This paper details size-by-size circuit performance. The Knelsons recover around 90% of free gold (as determined by Mozley Laboratory Separator), even below 37 µm. The sluices are ineffective, with upgrading ratios around 2, but approaching unity in the fine sizes. Much of the effectiveness of the circuit lies in the high gold circulating loads in the grinding circuit, which partly atone for the poor sluice performance, and the ability of the Knelsons to recover free gold. Secondary upgrading recovers slightly above 99% of the primary concentrate gold, 90% of it in a 65% Au concentrate and the balance in a heavily loaded (10 to 20 oz/st) pregnant solution.

INTRODUCTION

Background

The operating performance of the mill of Les Mines Camchib Inc. has been described earlier (Laplante et al., 1989). The presence of significant chalcopyrite (1% and up) makes flotation necessary prior to cyanidation, to limit cyanide consumption and minimize environmental impact. In 1982, the presence of very high gold circulating loads in the grinding circuit lead to the commissioning of a gravity circuit, schematically shown in Figure 1. Gravity recovery of gold has been found beneficial because it increases metallurgical recovery by l-3% (Laplante et al, 1987), as only part of the flotation tail is cyanided, and economic recovery, because of the lower net smelter return of gold in the flotation concentrate (as compared with the gravity concentrate). The latter increase is estimated, with a net smelter return of 95%, at 1.5% when gold recovery by gravity is equal to 30%. In early 1988, a sampling program was initiated to characterize the behaviour of gold in the grinding/gravity circuit. This paper reports on the results of this study.

Methodology

Free gold concentration is normally determined by amalgamation. This procedure, however, suffers some serious drawbacks. It is time consuming, represents a health risk, and may give a biased estimate of recoverability by gravity. For example, thin flakes are readily amalgamated, because of the high specific surface area, but can be difficult to recover by gravity, especially for very thin flakes. Amalgamation does not provide a grade-recovery curve, a useful tool to assess unit plant performance. Thus, the traditional amalgamation route was discarded in favour of a laboratory gravity separator. Below 210 µm (65 mesh), a Laboratory Mozley Separator (MLS) was used on individual Tyler classes; the unsized +210 µm was processed with a Gemini table. The Gemini table could not reproduce the high concentrate grades of the MLS (Liu, 1989), but could process the larger mass (2 kg and more) necessary to yield adequate statistics. Free gold content then becomes more a function of response to gravity concentration than actual liberation; it is a more relevant parameter to analyze gravity circuits.

Each sample extracted from the grinding/gravity circuit was dried, wet screened to 37 μm, then dry screened from 210 to 37 μm. Up to 150 g of each size class was processed on the MLS; for the 150-210 μm, two 150 g samples were processed and the corresponding products combined to improve sampling statistics. Operating conditions were a frequency of 84 min^{-1}, an amplitude of 63 mm, a peripheral wash water flow of 1.0 L/min for the 150-210 μm down to 0.4 L/min below 53 μm, and a longitudinal slope of 3° for the 150-210 μm down to 1" below 75 μm. Light minerals were taken as a tail after 3 to 5 minutes of operation, and a middling (M2) after 12 to 15 minutes; the MLS was then stopped, and the mineral band remaining on the tray split into a concentrate containing about 1% of the feed mass and another middling (Ml) containing most of the sulphides. If the first middling mass was greater than 5% of the feed mass, its tail end was added to M2. Each product was fire-assayed to determine gold content. When plotted as a grade-recovery curve, duplicate tests yielded slightly different points, but lying on the same curve. Curves were fitted to a cubic spline (Whiten, 1971).

RESULTS AND DISCUSSION

Grinding and Classification

The grinding circuit at Camchib consists of one rod mill (4.0 x 3.4 m) and two parallel ball mills (3.7 x 3.0 m) operated in closed circuit with two 51 cm cyclones. Samples of the rod mill and ball mill discharges, cyclone underflow and overflow were extracted over a two-hour period, at a fresh feed rate of 122 t/h. Samples were processed as described above, to characterize overall and gold grinding and classification performance. A metallurgical balance was presented in an earlier paper (Laplante et al., 1989). The circulating is 300% for solids, but 560% for gold (based on the gold content in the rod mill discharge); the cyclone overflow is 66.1% -75 μm (200 mesh). Gold's circulating load is high, but it is down considerably from the 1800-2500% observed prior to installing the gravity circuit.

Figure 2 shows the yield-recovery response of the rod and ball mill discharges and cyclone underflow and overflow (all sizes). Each yield-recovery curve is calculated from corresponding curves for each size class. The poor liberation of the rod mill discharge is evident. There is a definite increase in liberation between the ball mill feed and discharge. The cyclone overflow is not as liberated as the ball mill discharge, because a significant amount of free gold has been recovered in the gravity circuit, and also because free gold is preferentially recycled to the ball mill, thus increasing the degree of apparent liberation in the recycled product.

Figure 3 compares the classification performance for the overall ore, total gold, "free" gold (gold content in a MLS concentrate mass of 1%). Gold is classified at a considerably smaller cut-size than ore. The cut-size, D,, is estimated from the corrected classification efficiency (Plitt, 1976). The D_{50} of the ore is 104 μm. Total and free gold's very high recovery to the underflow precludes calculating their D_{50} with any accuracy (Plitt recommends deleting all values of corrected efficiency above 95% to calculate the D_{50});

screening to a size limit much finer than 37 µm would be required to do so.

Pinched Sluices

The pinched sluices were designed on-site, to replace static sluices that were labour-intensive, and virtually inefficient below 74 µm. The first design called for a single launder, 6.1 m long, and narrowing from 0.7 m at the feed end to 0.2 m at the concentrate port. The sluice bottom is V-shaped and the concentrate port diamond shaped, with a gate to control concentrate flowrate. Upon commissioning, it was observed that the recovery of -74 µm gold increased significantly, but the large degree of loading of the sluice was apparent. For the second ball mill, the design was modified: a second trough similar to the first was added, hence producing a "double-sluice".

Samples of the feed, concentrate and tails of the single and double sluices were taken under normal operating conditions, during the same two hour period as above. Thus, the sluice feed rate can be estimated from the circulating load. The sluice concentrate samples were timed, to estimate flow rate. The tails flow rate can be calculated by difference. The samples were processed as described above. The algorithm NORBAL2 (Spring, 1985), was used to adjust size-by-size and overall grades. Results for the two sluices, plus a third test performed at a higher feed density (66.2% solids) with the double sluice, are shown in Table 1. The double sluice produces a higher yield at a higher upgrading ratio, at a significantly higher gold recovery. Increasing feed density with the double sluice from 63 to 66% w/w results in a slight improvement in gold recovery. This result is probably not statistically significant, but is in good agreement with other sluice results in the same density range (Nudo, 1987; Chuan and Ithnin, 1983). The third test gives the highest gold recovery, 17%, with the highest upgrading ratio, 2.3. The double sluice is a superior unit, because each trough is fed only half the rate of the single unit. Performance remains poor, as is typical of pinched sluices, but even more so because the loadings are considerably above those normally used in pinched sluices. For example, the Wright Concentrator trays are only slightly smaller than the Camchib sluices, but their capacity is typically 5-10 t/h (Chuan and Ithnin, 1983).

TABLE 1 Sluice Metallurgical Performance

	Test 1	Test 2	Test 3
Sluice Type	Double	Single	Double
Feed Density (% Sol.)	63	66	66
Yield (%)	7	5	7
Upgrading Ratio	2.1	1.7	2.3
Overall Au Recov. (%)	15	8	17
Size-by-size (µm)			
+210	31	14	46
150-210	12	5	10
105-150	9	4	12
75-105	11	5	9
53- 75	3	6	8
37- 53	8	7	8
-37	5	3	12

Table 1 also gives the size-by-size recoveries. Results are erratic, but the trend is clear: recovery increases with increasing particle size. Recovery of -37 μm gold is lowest. This trend is disquieting for two reasons: first, fine gold is most likely to be classified to the cyclone overflow and second, the Knelson's ability to recover fine gold is not fully exploited. Again, test 3, with the double sluice at a higher feed density, yields the best recovery for fine gold. These operating conditions should be probed further.

76 cm Knelson Concentrators

At Camchib, the two sluice concentrates are combined, screened to 1.8 mm (10 mesh), and the undersize fed to one such unit. Every 1 to 2 hours, feed is transferred to a second unit, and concentrate is discharged from the first. Thus, one unit is always on stand-by, and operations are continuous. Four sampling tests were performed: a first at low feed rate (one sluice operating), a second at standard operating conditions, a third at low backwater pressure (40 kPa, 18.0 t/h) and a fourth at standard condition (80 kPa, 19.6 t/h). Results of the second test were highly erratic, and will not be discussed here. The sampling procedure for tests 3 and 4 was modified: increased masses were extracted and timed samples of each sluice concentrate and screen oversize were taken.

For each test, operation over a full 90 minute cycle was characterized. The tails sample was broken into sub-samples of O-10, 10-30, 30-50, and 50-90 minutes. For tests 3 and 4, the feed sample was also split into sub-cycles. All samples were processed as described above.

Overall Results

Table 2 gives overall results. Recovery is highest for test 1, at standard back-water pressure and low feed rate. Recovery drops with increasing feed rate (test 2), and further with decreasing back-water pressure (test 4). These results must be interpreted carefully, as the proportion of free gold in the Knelson feed is likely to vary substantially. Concentrate mass was estimated for test 3 at 45 kg. This corresponds to a weight recovery of 0.15% with respect to the Knelson feed and 0.025% with respect to the circuit feed.

TABLE 2 76 cm Knelson Tests Overall Results

	Test 1	Test 3	Test 4
Back Water Pressure (kPa)	80	73	40
Back Water Flow Rate (t/h)	-	24	21
Feed Rate (t/h)	13	18	20
Total Gold Recovery (%)	71	62	53
Free Gold Recovery (%)	91	82	93
Upgrading Ratio	326	480	336

Free Gold

The performance of the Knelson is better assessed if free gold recovery can be estimated. The following approach was taken: free gold was assumed to be present in the MIS at a very low mass; since mass recovery in the Knelson concentrate is negligible, it can be assumed that free gold present in the Knelson feed and not recovered will be present in the corresponding mass in the MIS concentrate (of the Knelson tail). Free gold recovery is equal to:

$$R_f = 100\% * [l - (M' * G,') / (M' * G,')]$$

where

R_f: free gold recovery
M': MIS concentrate mass % in which free gold is recovered
G_t': gold content of the MLS concentrate of mass % M' in the Knelson tail
G,': gold content of the MLS concentrate of mass % M' in the Knelson feed

The MLS mass recovery in which all free gold is recovered, M', is not known. It should logically correspond to the maximum R,, since any higher value of M' will include locked gold which is not recovered as effectively. This is illustrated in Figure 4, which shows gold recovery converging to slightly above 90% for tests 1 and 4. For test 3, gold recovery begins drops with decreasing M' from 1 to 0.1% recovery. This behaviour is most likely attributable to a poor operation of the MLS for the Knelson feed, with a poor recovery of free gold in the concentrate.

Cycle Time

Figures 5 and 6 show total and free gold recovery for tests 1, 3 and 4 throughout the full 90 minute cycle. The general increasing trend for total gold recovery is not considered significant. Recovery should drop sharply if the Knelson becomes overloaded, or progressively if gold recovered in the riffles has even a low probability of being lost again. Neither is observed, suggesting that captured gold is well shielded in the bed contained within the riffles. The obvious conclusion is that cycle time could be extended at no cost to recovery, to achieve higher upgrading ratios and decrease the extent of retreatment downstream.

Particle Size

Figures 7 and 8 show the effect of particle on recovery for total (tests 1 and 3) and free gold (tests 1, 3 and 4). The recovery of total gold decreases with increasing particle size.. Free gold recovery, however, shows no trend, although data display significant scatter; the line represents the average of all points. These two figures demonstrate that the technique used, although still in need of refinement, can adequately separate between liberation and equipment inefficiency. An interesting size class is the -37 µm, which shows a slight decrease in recovery in Fig. 5, but none in Fig. 6. The free content was evaluated with the MIS V-tray, which is designed for coarse feeds; undoubtedly, some fine free gold was lost. It is likely that if the flat tray had been used, the estimated free gold content of the feed and tail would have been higher, and calculated free gold recovery lower. The particle size at which the Knelson efficiency drops noticeably is below 37 µm, and most likely between 10 and 15 µm.

The Gold Room

Knelson concentrates are pumped to a controlled access area, the gold room, where further processing takes place. At the time of the study, the Knelson concentrate was being treated with a smaller (19 cm) unit; with a roughing/scavenging pass. The rougher and scavenger concentrates are combined and fed to a riffleless table; the concentrate is slowly skimmed and the middlings repeatedly recycled. The table and 19 cm Knelson tails are combined and (presently) send to cyanidation columns.

19 cm Knelson

A single sampling test was performed, for which the 19 cm Knelson feed, rougher and scavenger concentrates and scavenger tail were sampled. The high feed grade, 303 oz/st, eliminated all nugget effects. Overall recovery is 89.9% Au, with a combined concentrate of 1786 oz/st, whilst the final tails remains high in gold content, 36 oz/st. The scavenging step is necessary to keep the gold content of the tails low. The scavenger concentrate has a higher gold content than the rougher feed, 629 oz/st, which justifies combining it with the rougher concentrate (Agar, 1980). Figure 9 shows that recovery increases with particle size, even down to 37 µm. Much of the coarse gold particles lost contain definite chalcopyrite and pyrite concentrations, as determined by energy-dispersive analysis on a scanning-electron microscope. However, the lost gold is clearly present as flakes (Fig. 10), which indicates that sulphide inclusions, when present, are minor constituents. It is believed that the failure of very coarse flakes to trickle down to the recovery zone is the main mechanism that explains losses to the tails. The high recovery of the -37 µm would support this. This high recovery must not be interpreted to mean that the Knelson can recover <u>all</u> -37 µm material effectively, as the feed here is the concentrate of a larger Knelson. Rather, it indicates that the size limiting fine gold recovery is equal or smaller for the smaller Knelson unit. The 19 cm Knelson has since been replaced with a larger (30.5 cm) unit, which achieves a higher recovery in a single pass.

Riffleless Table

Evaluating the performance of the riffleless table proved difficult, because of the repeated recycling of the middlings, which contain a significant fraction of the total gold. For the first pass, a 1786 oz/st feed yielded a 20.1 oz/st tail, a 14247 oz/st concentrate (44.3% Au). From a process view point, this compares well to the 19 cm Knelson, which had a higher tails grade in a larger mass. Thus, most of the gold contained in the gold room tails comes from the Knelson tail. This tail was first combined to the flotation concentrate, because of its high gold content (up to 40 ozs/st); it was then shipped to be refined separately, but is now cyanided in columns and returned after washing to the grinding circuit, with a gold content generally below 1 oz/st. As the 76 cm Knelson grade varies between 200 and 400 oz/st, the combined recovery of the gold room and cyanidation columns is well above 99%.

FUTURE WORK

Circuit Modifications

Results of this study suggest that significant improvements in gold recovery can be achieved. Approximately 40% of the gold escaping the gravity circuit via the cyclone overflow is free. The pinched sluices are clearly the weak link. Installing a second sluice to "double" the single unit would increase recovery appreciably, as the double unit shows increases of both yield and upgrading, compared to the single unit. Yield can be increased because the present feed rate of the 76 cm Knelson, 20 t/h, is double that of its rated capacity, 40 t/h. Additional but more modest improvements can also be achieved if sluice feed density is optimized.

Failure of the Knelson concentrator to show decreases in recovery even after 90 minutes of operation suggests that the load cycle could be much longer. Present 'loadings of 10 to 20 gold ounces per cycle occupy a very small fraction of the total riffle volume. The practice of using one unit on stand-by to provide continuous operation is therefore questionable. Both units could be used simultaneously, as the water requirements, 50 t/h for both units, are well below the 200 t/h present in the cyclone overflow (approximately 100 t/h solids at 33% solids weight by weight). Thus, much of the present water addition comes nor from the rod mill discharge nor from gland water. With the two 76 cm units operated at 80 t/h, sluice yield could be increased from the present 6-8% to 25-30%.

Ultimately, it could be argued that sluices should be replaced with more efficient units. Reichert cones have been used successfully for such applications (Lammers, 1984). It is unlikely, however, that their installation would be warranted in this case, especially if sluice yield can be increased substantially.

New Developments

Immediate improvements to the circuit were suggested above. The inability to predict what will be the magnitude of their impact stems largely from ignorance -- ignorance of gold's grinding and classification behaviour. Both are very different from those of other minerals. The ability to recover gold well within the sub-sieve range will make it necessary to characterize its behaviour in that range.

The MLS has proven effective as a "perfect" separator. In reality, its use for the purpose of this study is far from perfect. The approach used often yielded noisy data, was time-consuming and costly. Already, the advent of a 7.6 cm Knelson makes it possible to process quickly 5 to 15 kg samples and recover "free" gold in a 100 to 150 g concentrate.

This technique is now being evaluated. It is already apparent that the Knelson is well suited for streams such as primary cyclone underflows and ball mill discharges, but much less for cyclone overflows.

The effectiveness of the cyanidation column suggests that the gold room could be entirely by-passed in favour of column cyanidation followed by electrowinning. This option is also under investigation.

The positive impact of gravity at Camchib is undeniable. The role of gravity in conventional cyanidation mills is much less obvious. From gold's behaviour in classification (Fig. 2), it can be concluded that even where there is significant coarse gold, losses due to its incomplete dissolution are negligible, because it will leave the grinding circuit as fine gold. Indeed, a mill such as Golden Giant has removed its gravity circuit, which recovered very coarse gold, with no apparent loss of recovery (Larsen, 1989). A technico-economic study of gold gravity circuits is warranted. Such a study would aim first to characterize the performance of existing circuits, then to evaluate the impact of gravity on overall metallurgical and economic performance.

CONCLUSIONS

The gravity circuit at Les Mines Camchib Inc. makes a significant contribution to the economic performance of the mill. Part of this stems from the increased value of gold recovered as bullion, as compared to flotation concentrates. Another contribution is an actual increase in metallurgical recovery, estimated at l-3%.

The primary gold recovery circuit incorporates pinched sluices and two 76 cm Knelson concentrators. Its effectiveness relies on the high gold circulating load and the ability of the Knelsons to recover around 90% of the free gold captured by the sluices, over the full size range. The performance of the sluices is poor; some enrichment takes place above 210 µm, but little below.

Secondary upgrading makes use of smaller Knelson units and a riffleless table, whose tails are column cyanided. Overall recovery is slightly above 99%, of which about 90% is by gravity.

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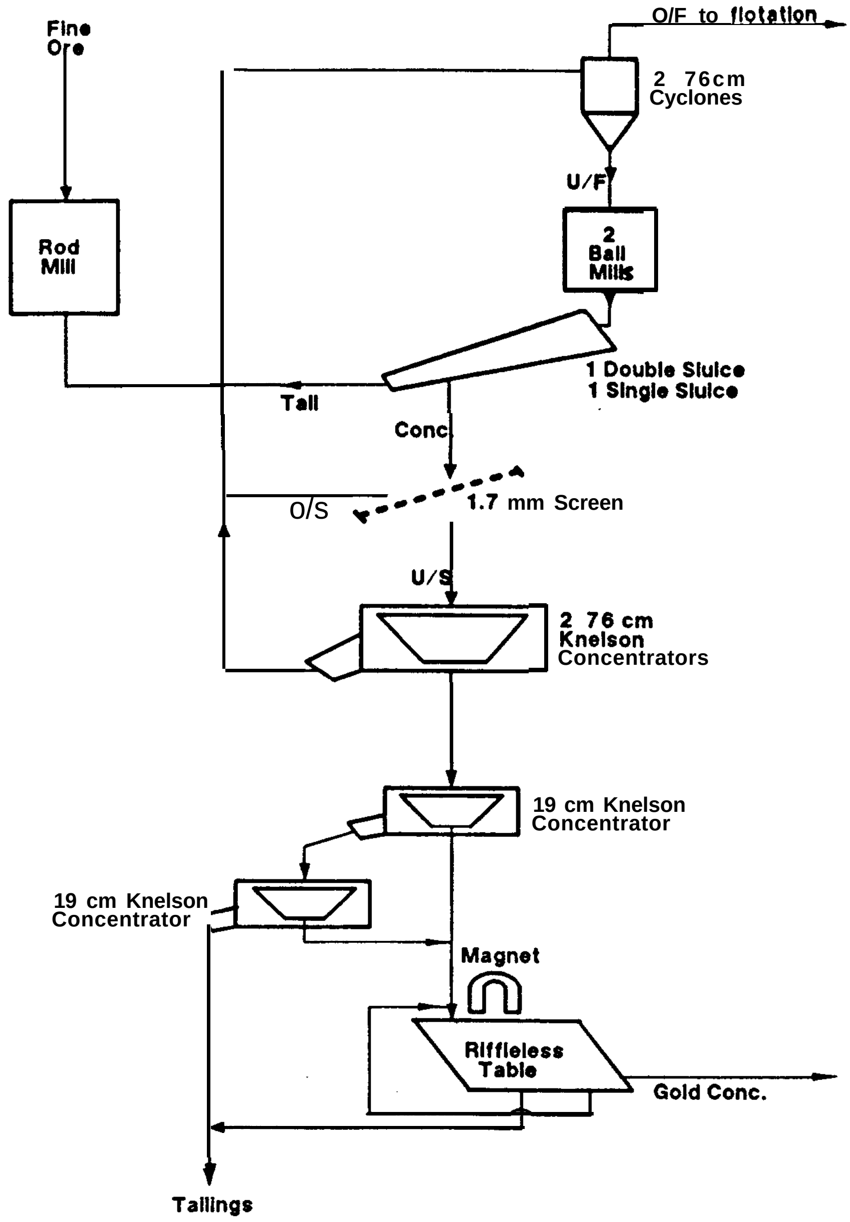


Figure 1 Gold Gravity Circuit at the Time of the Study (From Liu, 1989)

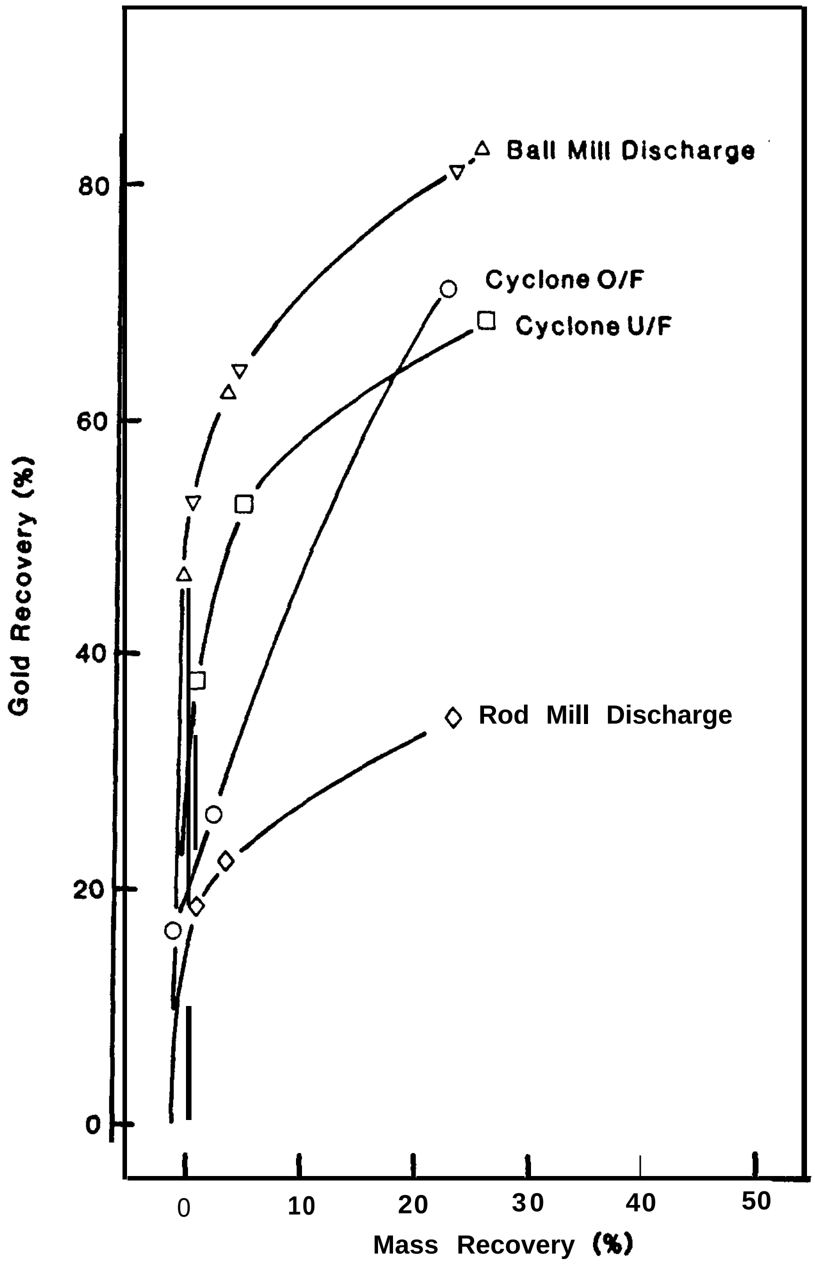


Figure 2 Gold Recovery Vs. Yield for Five Grinding Circuit Streams (Inverse Triangles: First Ball Mill Discharge; Upper Triangles: First Ball Mill Discharge)
(From Laplante et al., 1989)

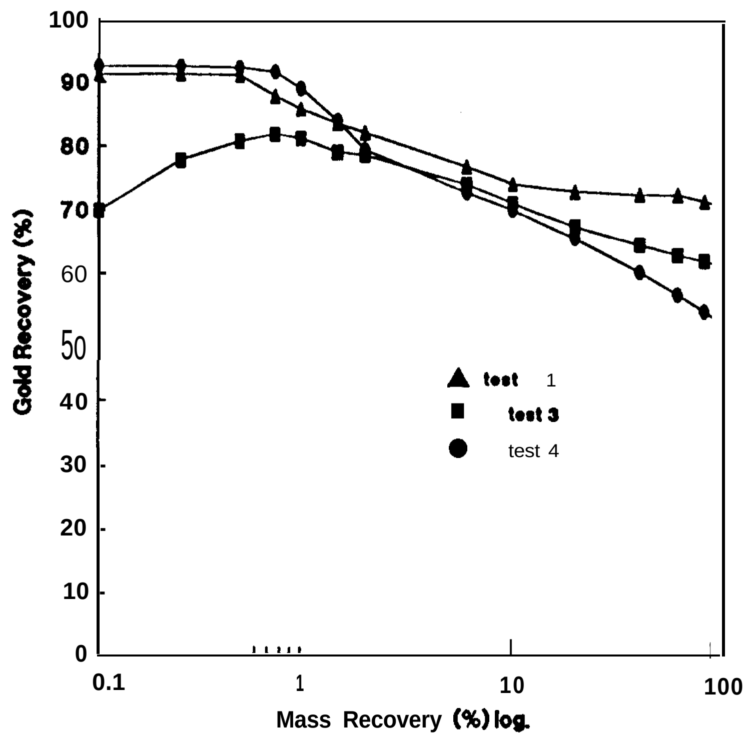
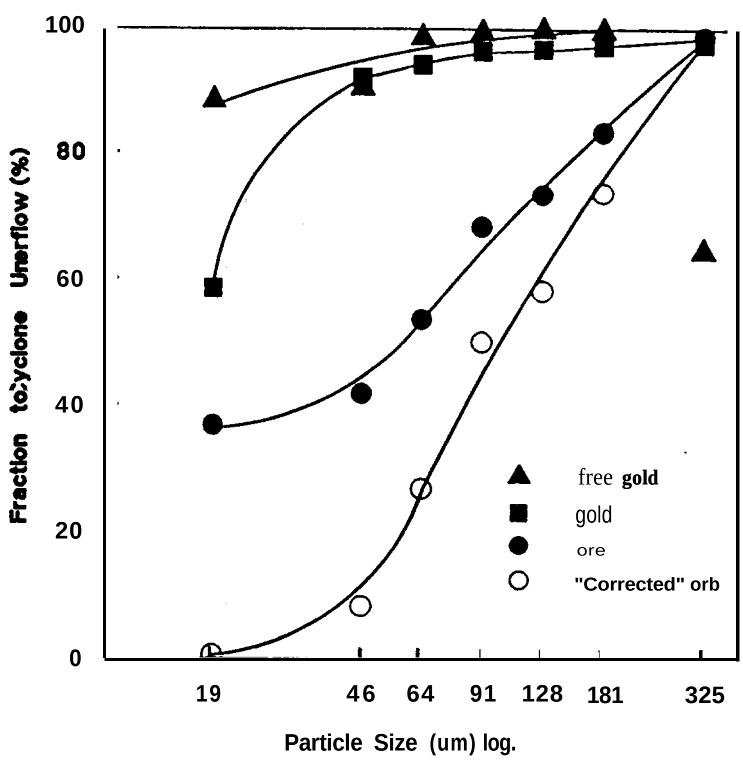
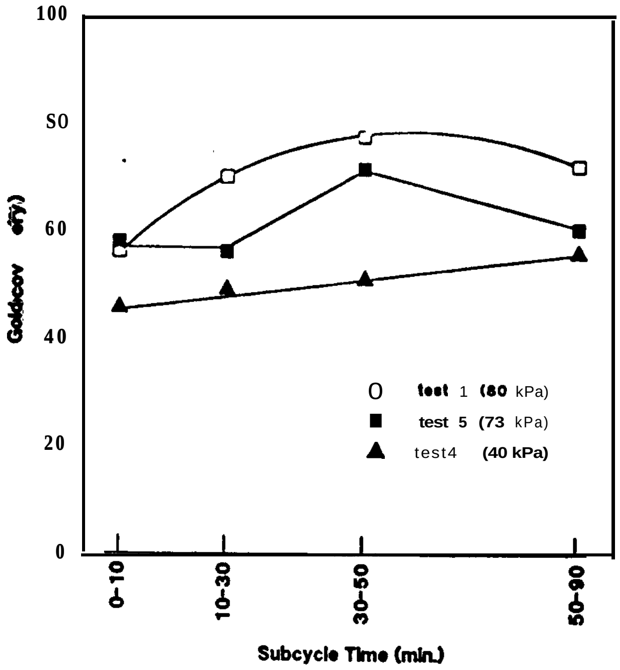
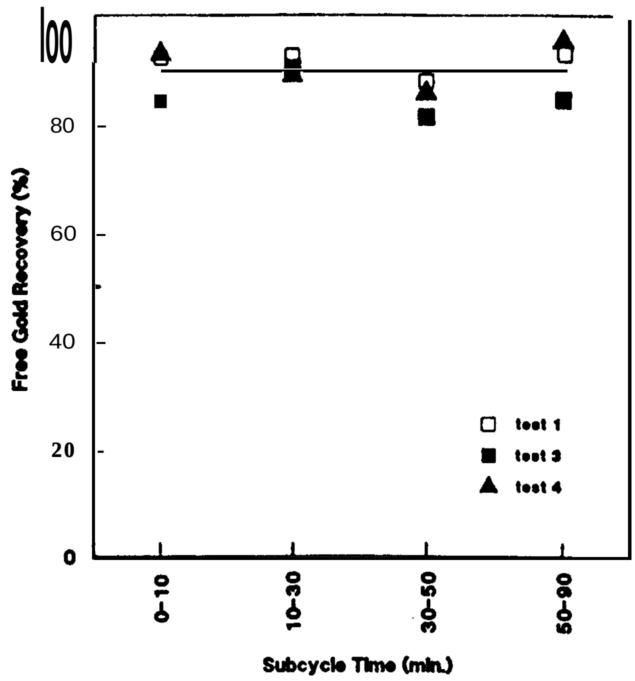


Figure 3 Determination of Free Gold Recovery for Tests 1, 2 and 4 (From Liu, 1989)

Figure 4 Cyclone Performance Curves (From Liu, 1989)

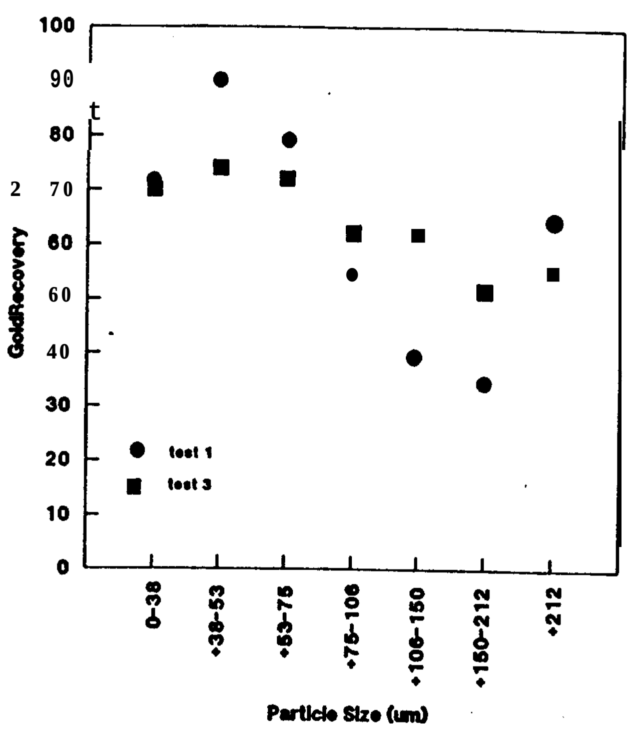


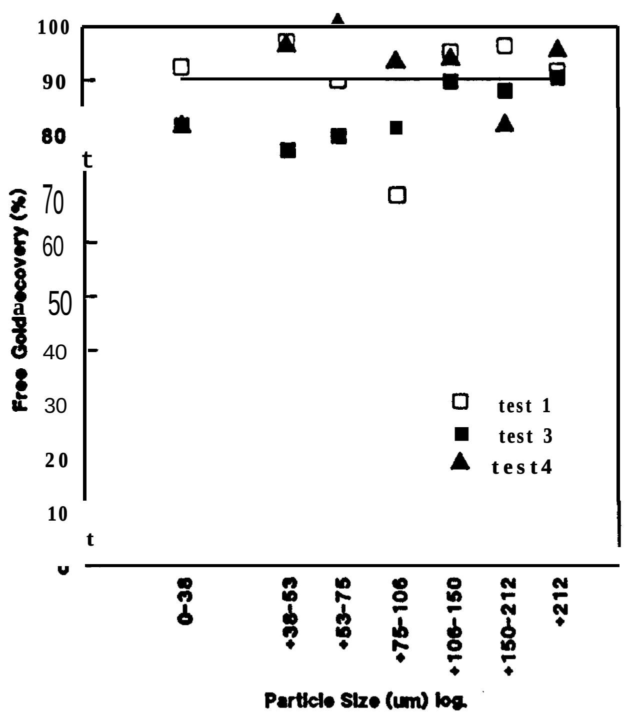




Figures 5 and 6 Total and Free Gold Recovery Over the 76 cm Knelson Cycle

Figures 7 and 8 Size-by-Size Total and Free Gold Recovery for the 76 cm Knelson
(All From Liu, 1989)





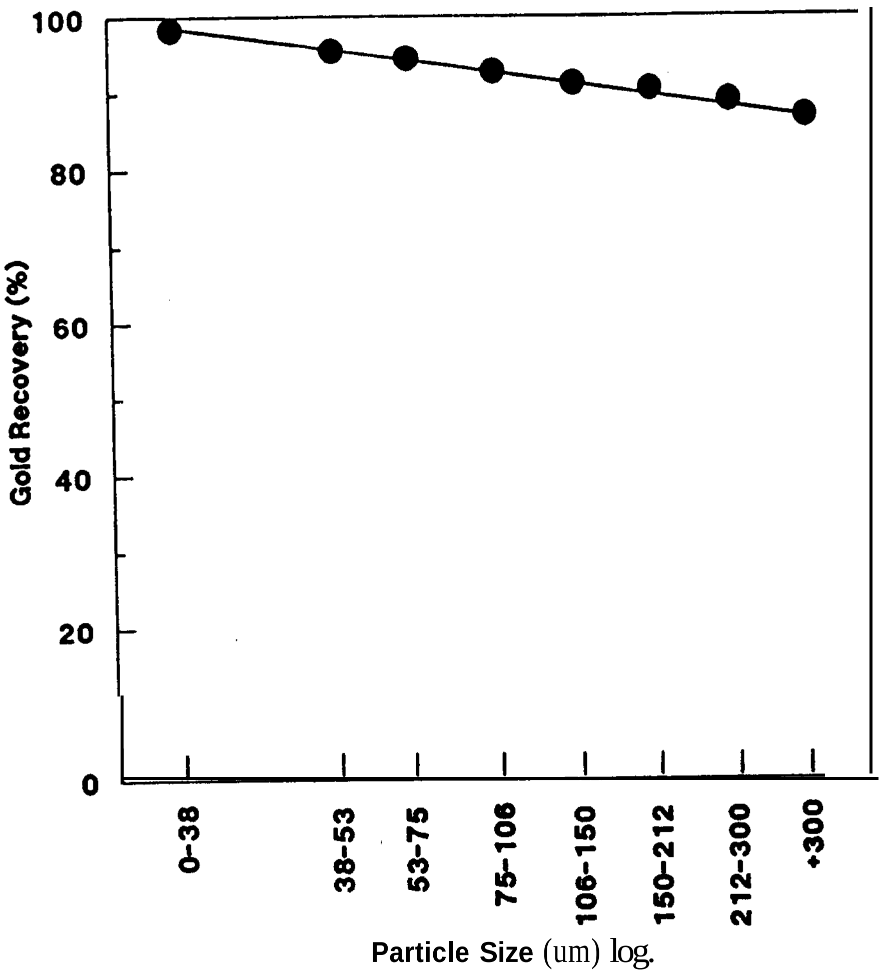


Figure 9 Size-by-Size Gold Recovery for the 19 cm Knelson (Rougher and Scavenger)
(From Liu, 1989)

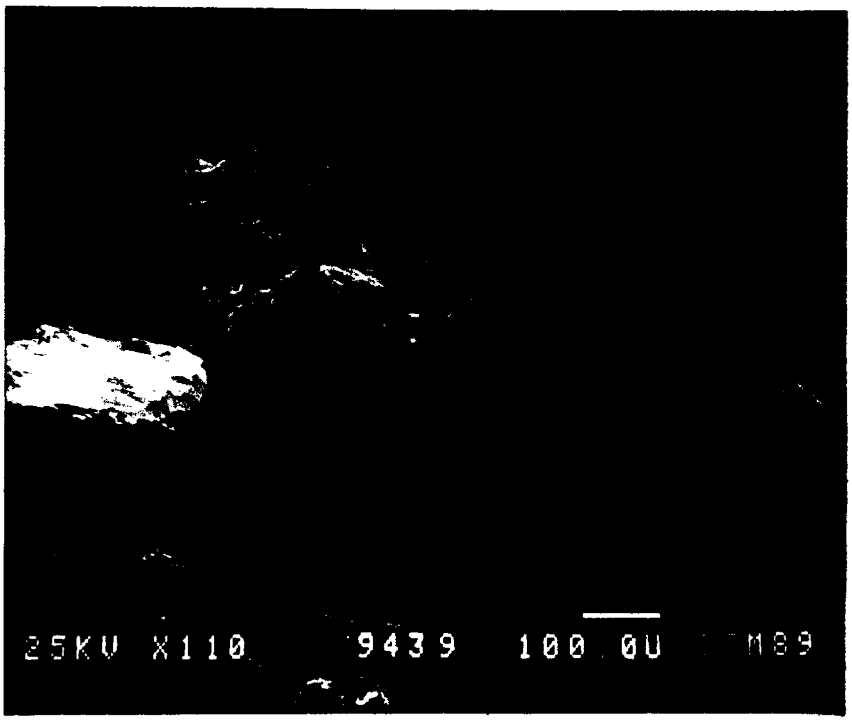


Figure 10 Scanning Electron Microscopy Photograph of Two Gold Flakes Lost to the 19 cm Knelson Tails