

# Testing Requirements and Insight for Gravity Gold Circuit Design

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## ABSTRACT

Gravity gold recovery can be of significant economic importance for gold copper ores that cannot be directly cyanided. Gravity is typically followed by flotation to produce a smeltable copper concentrate. Flotation tails gold values are often too low to justify a cyanidation circuit. Given this typical scenario, a very effective gravity circuit can yield higher overall gold recoveries (i.e. gravity + flotation), and also improve the NSR of gold. Typical scoping work, whilst well suited to investigate the downstream circuit, be it flotation or cyanidation, often overestimates ore variability to gravity recovery and fails to yield the data needed to predict plant gravity recovery and design the optimum gravity circuit. This presentation examines why typical scoping work fails to address gravity recovery adequately and shows how a standard test to characterize gravity recoverable gold (GRG) can provide the missing data. Case studies are presented and used to illustrate the concepts of recovery effort and aggressive gravity recovery.

## INTRODUCTION

Gold gravity recovery has evolved significantly over the past twenty years, largely because of the advent of the Knelson Concentrator. A number of graduate students at McGill University have completed largely applied research on ore characterization (Woodcock, 1994), evaluation gravity circuit of gravity circuit performance (Putz, 1994; Vincent, 1997, Zhang, 1998), or the fundamentals of semi-batch centrifuge operation (Buonvino, 1993; Ling, J., 1998; Xiao, 1998; Huang, 1996).

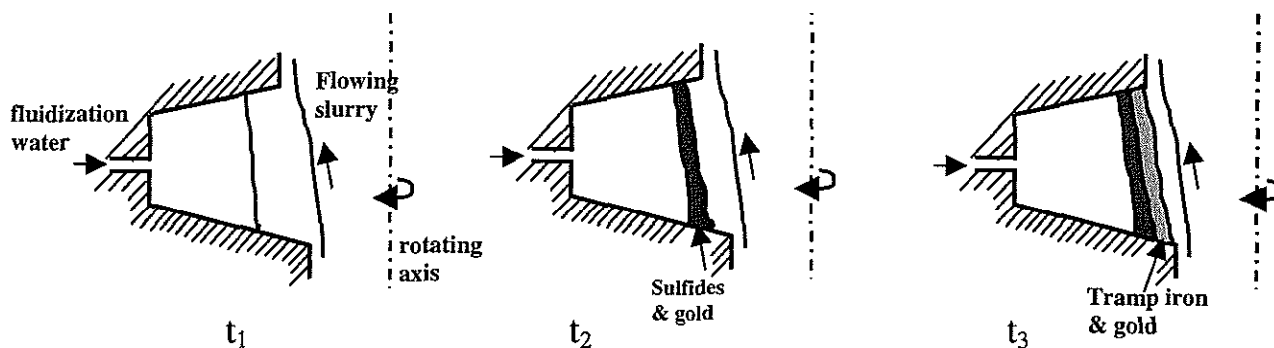
For characterizing gravity recoverable gold (GRG) in ores, the McGill University test was developed in the early 90s and used for over 70 ore samples. The test uses a representative feed as small as 30 kg for high-grade, finer gold, and as large as 150 kg for low grade, coarser gold. The feed samples are composited from drill core of primary (rod, Sag) mill discharge, or screen or screw classifier undersize when primary mills are operated in closed circuit. When plant samples are used (for retrofit or optimization studies), they are usually composited over a full month, to average out day-to-day variations.

The test itself consists of a sequential liberation and recovery with a 3-in Knelson, first at 100% finer than 850  $\mu\text{m}$  (stage 1). A second stage follows, at 45 to 60% passing 75  $\mu\text{m}$ , with part of the tailing of stage 1 as feed, typically 27 kg. A third stage follows, at final grind (typically 80% -75  $\mu\text{m}$ ), using most of stage 2 tailing as feed, typically 24 kg. Details are presented elsewhere (Woodcock, 1994; Woodcock and Laplante, 1993). Of interest here are "simplified" tests that were completed by grinding to final size and processing with a 3-in Knelson, either once or three times (since the original GRG test has three recovery stages). The following findings emerged:

1. Grinding to final size and processing once always yielded lower gold recoveries and a finer GRG size distribution than the standard GRG test, even when much lower feed masses (about 5-10 kg) were used. Clearly, some GRG was ground into finer gold, some gravity recoverable, some not.
2. When using **three** recovery stages at final grind with lower feed masses, total gold recovery could exceed GRG content, especially with low ores having a low sulphide content. Some **non-GRG**, presumably associated with sulphides, was then recovered.

The above conclusions provide a useful background for a critical examination of typical scoping test data, as it pertains to gravity recovery. Scoping test work resembles the simplified test (i.e. single recovery stage, at final grind), and even uses lower feed masses, typically 2 kg. With a low-density gangue, the Knelson will typically

recover about 100 g, or 5% of a feed mass of 2 kg. Typical plant Knelson operation recovers typically 0.06% of the mass, or **one hundred times** less. How does this discrepancy affect the scale-up of scoping work results? Before this can be addressed, a brief discussion on how the laboratory and plant Knelson Concentrators work is warranted.



**Figure 1** Schematic of the Knelson Recovery Cycle for a Single Groove

Both units recover material in grooves that are very partially fluidized, as shown in Figure 1. After a very short feeding time ( $t_1$ ), the bottom of each groove fills with coarser particles with a high terminal settling velocity. The gold content of this material can be lower than that of the feed (Huang, 1996). This material is recovered based much more on its particle size than specific gravity, and will remain at the bottom of the grooves throughout the full Knelson recovery cycle, to report to the concentrate.

Once the bottom of each groove is filled, selective recovery begins: a concentrate bed is formed at the top of the non-fluidized material in each riffle. Gold and dense minerals such as gold-bearing sulphides are recovered. After a relatively short time (at  $t_2$ ), the sulphides begin to experience significant erosion from the top of the concentrate bed and their recovery rapidly plummets. What follows then is very selective recovery of GRG, tramp iron, and some high-density minerals (e.g. scheelite, galena, cassiterite).

Eventually, as gold and very dense and coarse particles (e.g. tramp iron) build up the concentrate bed, they eventually extend far enough into the flow slurry to experience themselves severe erosion, and their recovery begins to drop significantly. This has been identified in previous work as the onset of concentrate bed erosion ( $t_3$ ) (Laplante et al., 1996).

The Knelson recovery cycle must be understood in terms of  $t_1$ ,  $t_2$  and  $t_3$ . For laboratory Knelson tests,  $t_3$  is seldom reached, unless head grades are exceptionally high (e.g. when treating plant Knelson concentrates) or gangue is both coarse (more than 10% in the  $+300 \mu\text{m}$  fraction) and dense (more than 50% sulphides). There is also evidence that plant Knelsons behave similarly (Zhang, 1998). For example, recoveries were sustained for four hours with a 30-in Knelson treating a flash flotation concentrate (dense gangue finer than 300  $\mu\text{m}$ ) (Putz, 1994) at Lucien Béliveau. At Mines Camchib, a 30-in Knelson Concentrator treating a feed screened at 1.7 mm with low sulphide content maintained the same recovery for a 2-hour cycle (Laplante et al., 1990).

For both laboratory and plant Knelsons,  $t_1$  is reached in a matter of seconds, and more than half the mass of the concentrate is thus recovered unselectively.

Laboratory and plant Knelsons recovery cycles can differ with respect to  $t_2$ . For plant Knelsons, even recovery cycles as short as 30 minutes are such that gold-bearing sulphide recovery will always be negligible. It is thus reasonable to assume that virtually all gold recovered in plant Knelson concentrate is GRG. Consider Table 1 that details the performance of the gold room at Omai and Sigma gold mines. Table recoveries in both cases exceed 95%, and gold in table tails is more than 70% gravity recoverable. Where table performance is poorer than at these two sites, gold in table tails is more than 60 or even 90% gravity recoverable (Laplante et al, 1999). This would not be the case of jigs, as they recover gold locked with coarse sulphides; this gold then reports to table tails as non-GRG gold (Vincent, 1997).

Table 1 Metallurgical Performance of Tabling at Omai and Sigma

Product Grade, g/t	Sigma	Omai
Table Feed	5300	14626
Table Tails	119	882
Table Tails non-GRG	15	285
%GRG in Table Feed	99.7	98.1

Laboratory Knelson operation may vary significantly. Typically, scoping tests use feed masses of 2 to 5 kg. As a 3-in Knelson recovers approximately 100 g (more with a high specific gravity gangue), a yield of 2 to 5% is obtained (i.e.  $100\% \times 100\text{g}/5000\text{g}$  to  $100\% \times 100\text{g}/2000\text{g}$ ). At these high yields, one can no longer assume that  $t_2$  has been reached and gold-bearing sulphide recovery is negligible. The same conclusion is reached when the mass recovered selectively in a 3-in Knelson, 25-50 g, is compared to the mass of sulphides in the feed. For a 2-kg feed at 2% sulphides, the 40 grams of sulphides in the feed is dangerously close to the mass that can be recovered between  $t_1$  and  $t_2$ . With the standard 60 kg of the GRG test, the same 2% sulphides weighs 1200 grams -clearly much more than what the 3-in can recover selectively.

For scoping tests, thus, Knelson concentrates must be upgraded if only GRG is to be recovered. This is achieved by hand panning, and more rarely amalgamation. Grinding samples to final size has produced a finer GRG distribution than that of the ore, which will increase panning losses, especially if very low yields and high grades are targeted. Figure 2 shows how yield affects gold recovery (100% = gold in the ore): the effect is very significant, and cleaner recovery is a function of the arbitrarily set final yield, rather than the GRG content of the ore.

Figure 2 Effect of Yield on Gold Recovery  
(Cleaning stage by hand panning: roughing with 3-in Knelson Concentrator)

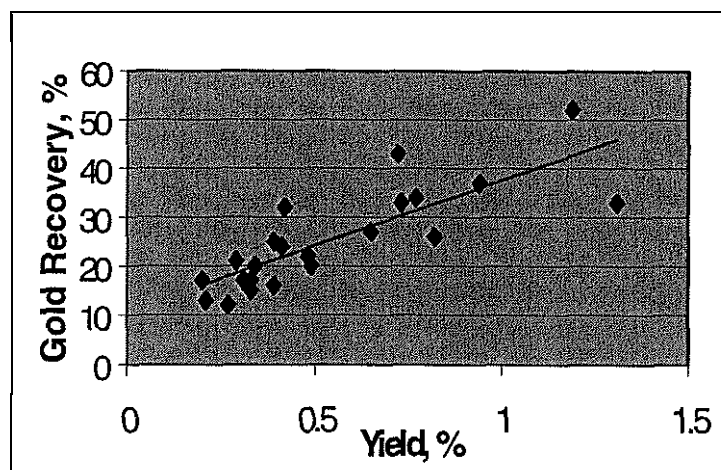


Table 2 summarizes the drawbacks of the typical scoping approach with respect to gravity recovery. Generally, the finer the final grind size, the coarser the GRG, the harder the gangue and the lower the final yield, the lower the gravity gold recovery.

Table 2 impact of the Typical Scoping Approach on Gravity Recovery

Action	Problem	Bias on GRG
Grind to final size	GRG overground	Underestimate
Knelson with high yield	Recovery of non-GRG	Overestimate
Panning concentrate To a low yield	Fine GRG lost	Underestimate
Amalgamate concentrate	Non-GRG recovered and Coarse unliberated GRG lost	Underestimate Or overestimate

To the above problems must be added the sampling problem of using a low feed mass. The central limit theorem dictates that the variance of the pure sampling error is inversely proportional to the mass extracted. Thus, a 2-kg feed sample will have a sampling variance 25 times that of the 50-kg sample normally used for GRG work. In the presence of significant gold above 150  $\mu\text{m}$ , scoping tests with 2-kg samples measure the variability of extracting 2-kg samples, and not the variability of GRG content in the ore. Fundamental sampling standard deviations that have been measured at 5% with the GRG test would reach 25% with 2-kg samples. These “errors” are not linked to any flaw in testing or sampling protocol other than the small mass extracted.

The sampling problems associated with small sample weights can be illustrated using a “synthetic” ore containing 1.5 g/t Au, with 0.5 g/t of non-GRG, 0.5 g/t of very fine, easily sampled GRG, and 0.5 g/t of coarser, difficult to sample GRG, contained in a single size class. This is quite a typical result (the mean GRG content of all ores tested is 66%). We will assume that all sampling errors come from the coarser GRG, and will vary the size of this GRG 300-425  $\mu\text{m}$  (average mass: 0.5 mg) to 850-1200  $\mu\text{m}$  (average mass: 5 mg). Table 3 shows that head grade would vary significantly when the coarser GRG is very coarse. The probability distribution of finding a coarse GRG particle in a 2-kg sample is then very erratic. For this case study, a Poisson distribution was assumed (a best-case scenario).

Table 3 Effect of Coarse GRG particle Size and Mass on the Standard Deviation of the Head Grade and %GRG in 2-kg Samples Averaging 1.5 g/t Au and 0.5 g/t Coarse GRG  
(Gold particle mass from Banisi, 1990)

Size Class of Coarse GRG, $\mu\text{m}$	Particle Mass, mg	Average Number of Coarse GRG Particles	Standard Deviation	
			Grade, g/t	%GRG
850-1200	5	0.2	1.12	14
600-850	3	0.33	0.88	14
425-600	1.3	0.77	0.57	12
300-425	0.5	2	0.35	a

For coarse GRG in the 850-1200  $\mu\text{m}$  class, an average of 0.2 particle per sample means that 82% of samples would contain no particles (grade: 1 g/t; %GRG: 50%). Also, 16% of samples would contain one particle (grade: 3.5 g/t; %GRG: 86%) and 1.6% would contain 2 particles (grade: 6 g/t; %GRG: 92%). **None of the samples would have either the correct head grade or GRG content.** The higher-grade samples would have more GRG, and the average GRG content would have to be calculated by weighting the GRG content of each sample with its head grade, otherwise a negative bias on the GRG content would result.

The example shown in Table 3 shows a relatively low variance because it was assumed that each sample has 0.5 g/t of non-GRG and 0.5 g/t, of fine, easily sampled GRG, setting a lower limit for GRG content to 50%. If the non-GRG or fine GRG content is reduced or assumed variable, the standard deviation of the GRG content would increase significantly. This is the case of ores that contain very little fine or erratically distributed fine GRG, and a GRG content that covers the range of nearly none to nearly 100% GRG is occasionally encountered in scoping work. Thus Table 3 does not represent the worst case scenario. It does show that even when the GRG content does not vary wildly, head grade can, in this case from a relative standard deviation of 23% to 75%. Using the larger sample size of the GRG test eliminates this problem, unless gold is coarser than 850  $\mu\text{m}$ .

The above discussion is not a condemnation of present scoping work practice. Rather, it preambles the following points. First, different data are needed to predict plant gravity recovery and design the grinding circuit. Second, scoping work aims at providing information for the recovery process downstream of gravity recovery, rather than gravity recovery itself. Third, the use of gravity recovery in general and the Knelson Concentrators in particular for scoping work can in some cases be refined.

The GRG test addresses the first point: it measures the natural size distribution of GRG, recovers only GAG and all of the GRG. This is illustrated in Figure 3, that shows the best and the worst performances obtained with the test. The test has clearly the ability to reject nearly all of the gold that is non-GRG (e.g. a GRG content of 3.5% at 61% -75  $\mu\text{m}$ ) and recover all that is GRG (e.g. a GRG content of 97% at 72% -75  $\mu\text{m}$ ). The natural size distribution of GRG, coupled with the economic incentive of gravity recovery, is then used to choose the magnitude of the gravity recovery effort and how it is best effected.

The GRG test effectively “takes the pressure” of scoping work to provide gravity data for circuit design and performance prediction. Scoping then fully focuses on the downstream recovery process, and the prior gravity recovery step is used solely to provide it with a feed with a realistic gold content and size distribution. The latter point is important, as typical flotation or cyanidation feeds are produced by hydrocyclones, and have a much finer gold size distribution than what is obtained by batch grinding in a laboratory rod mill. In this sense the typical use of gravity recovery in scoping work is largely vindicated.

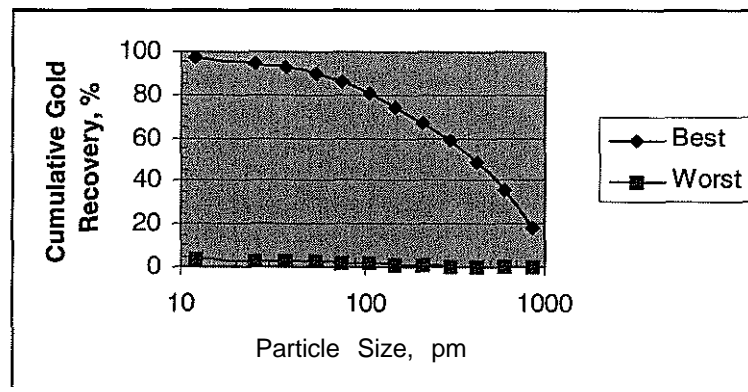


Figure 3 Extremes in GRG Response

Scoping protocols, in some cases, could be slightly improved. For example, if the same feed material is to undergo different tests at the same grind, it is better to prepare a common feed and perform the gravity step, inclusive of cleaning, and then split the gravity tails in sub-samples for the various test conditions. The Knelson Concentrate yield will then be lower (because of the increased mass of the common head sample) and the gravity tail sub-samples would have much more similar head grades and gold liberation. This would be useful for either flotation or cyanidation testing, and is particularly appropriate in the presence of coarse GRG.

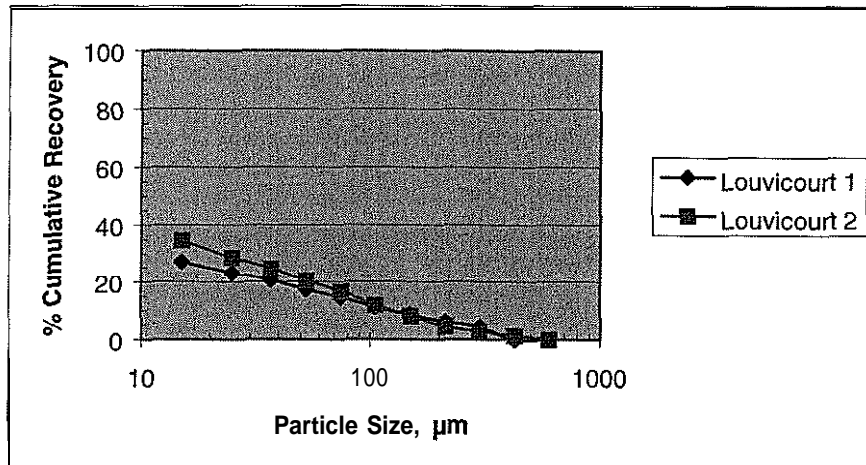
Since this paper focuses on laboratory test work for gravity circuit design, the experimental section will consist of a number of GRG tests performed on Au-Cu ore samples. Greenfield, retrofit and optimization cases will be presented, along with test results and diagnoses, with special emphasis on the interaction between flotation and gravity, and scale-up problems.

#### CASE STUDIES

Of all the GRG tests performed at McGill, 30 were on Cu-Au or Cu-Zn-Au ores. The ones presented here cover the full range of responses.

##### **Louvicourt Mine**

Two GRG tests were performed and yielded very similar results, despite very different head grades, 1.7 and 9.3 g/t. The GRG size distribution of both tests is shown in Figure 4. This ore represents the most common response of massive sulphide ores, a GRG content too low and too fine for effective gravity recovery. These problems make full scale operation very challenging, to the extent that only a small proportion of the GRG can be recovered in the plant. At Louvicourt Mine, the high gangue specific gravity and high silver content of the gold particles further impede full scale recovery, which was nevertheless attempted, with extremely poor results (Zhang, 1996). The addition of flash flotation cells in the circulating load was eventually used instead, with significant metallurgical improvements in gold, copper and zinc recovery, and a decrease in reagent consumption. The circulating load of GRG dramatically decreased, especially below 106  $\mu\text{m}$ . It was observed that most GRG reported to the flash flotation concentrate, from which it could be recovered by Knelson Concentrator. This was not implemented, as it did not result in a higher net smelter return.

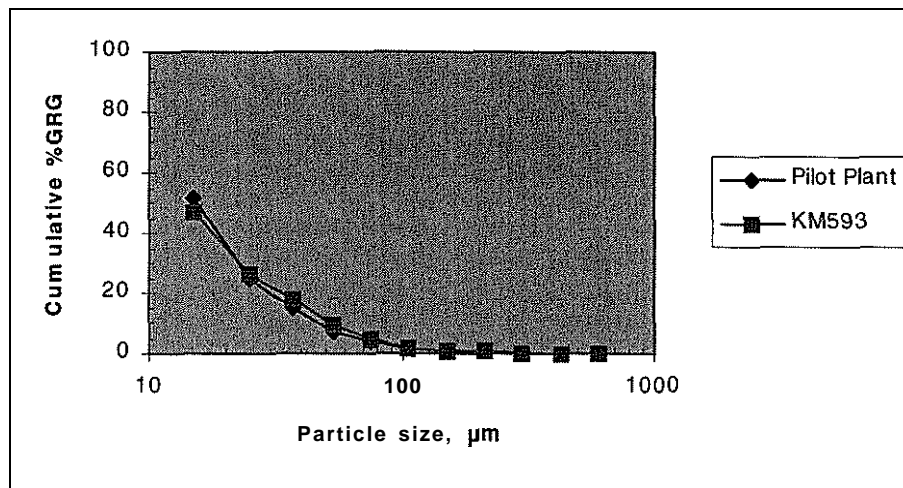


**Figure 4** Cumulative GRG Distribution for the Two Louvicourt Mine Tests

The Louvicourt case study is a typical condemnation test. It is by no means the only case where the GRG test yielded negative results, despite which further gravity efforts were made, to no avail. A word of caution is warranted here. The GRG test represents the **ultimate** gravity recovery effort, and prediction of plant performance must take into account the gangue specific gravity, the GRG size distribution, its silver content, and the recovery effort. As little as 25% of the GRG can be recovered in actual plant practice, and as much as 90%.

**Alumbra**

The Alumbra GRG tests yielded very consistent results, shown in Figure 5 (see also Keran et al, 1998). Virtually all the GRG is finer than 106 µm, and almost is finer than 20 µm. The GRG content is 49% at a nominal 80% -75 µm. The grinding flotation approach selected at Alumbra was a coarse primary grind at 150 µm, a bulk flotation in tank cells to recovery about 15% of the weight, followed by regrind at 80% - 37 µm.



**Figure 5** Cumulative GRG Distribution for the Two Alumbra Tests

The gravity flowsheet at Alumbra was finalized before GRG results became available. Thus, rather than discussing the existing circuit, a conceptual gravity flowsheet that would be ideally suited to the GRG size distribution will now be presented.

Gravity recovery from the primary grinding circuit should stress screening rather than Knelson Concentrators, because of the fineness of the GRG and the coarseness of the circulating load. Two stages of screening could be used, the first to cut around 800-1000  $\mu\text{m}$ , and the second around 150  $\mu\text{m}$ . This second stage would be very similar to screening in taconite plants. The second screen undersize would then be fed to Knelson Concentrators.

Secondary gravity recovery from the regrind circuit would be vary much like what is actually done, using 30-in Knelsons without pm-screening, but at higher rotating velocities, because of the fineness of the GRG. Alternately, a Falcon SuperBowl could be used, since its rotating velocity is higher than the Knelson's.

The proposed gold room would then be considerably simpler than the existing design, on account of the much finer gravity concentrate. The primary and secondary concentrate would be pumped to a large receiving tank and allowed to settle, to be processed with rougher and cleaner tables. A safety screen would be installed to remove oversize from the table feed due to screen failure in the primary grinding circuit. The cleaner table tails would be recycled to the concentrate hopper, and the rougher tails to a scavenging centrifuge (e.g. a 12-in Knelson). The Knelson tails would be pumped back to the regrind circuit. Because some gold is coated by very magnetic maghemite (Keran et al, 1998), magnetic separators should be used only very sparingly, if at all.

The Alumbra ore, because of the extreme fineness of the GRG and high tonnage, is a very challenging gravity application. From the nominal 50% GRG content, it is estimated that only half would be recovered at plant scale with the proposed flowsheet. Much like Louvicourt, this ore may be better suited to flash flotation than gravity recovery.

The existing gravity experience at Alumbra has served to confirm that (i) despite the coarse primary  $P_{80}$  and GRG fineness, gold still accumulates in the primary circulating load and (ii) the 30-in Knelsons in the regrind circuit recover significant amounts of gold. Both are notable advances in gold gravity recovery practice.

#### Troilus

The Troilus mill processes 15000 Vd of low Cu-Au grade porphyry deposit. The general flowsheet is similar to Alumbra's, although the GRG content is higher and coarser (Figure 6). Gravity recovery takes place in the primary circuit, with four 30-in Knelsons, whose concentrates are pumped to a gold room for tabling. Table tails are processed with a 12-in Knelson Concentrator, and then returned to the grinding circuit. The 12-in Knelson concentrate is further upgraded on a smaller Gemeni table.

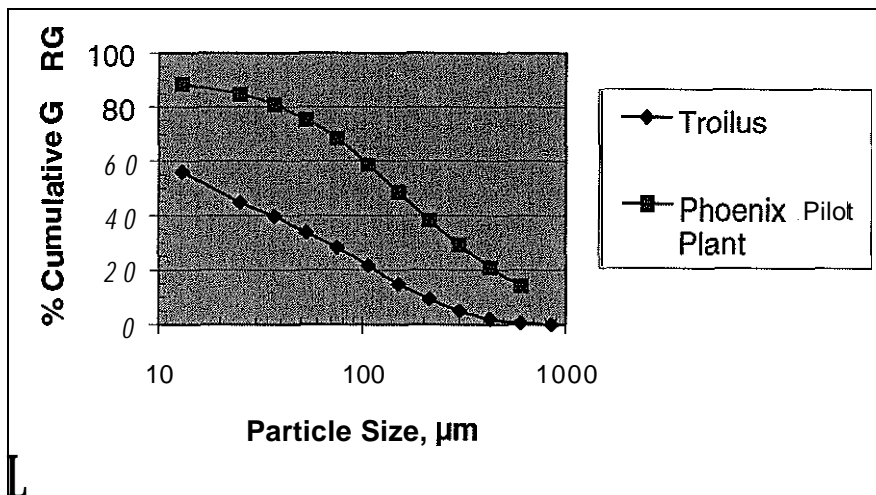


Figure 6 %Cumulative Recovery for Troilus and Phoenix Samples

Gravity recovery now averages around 30%, whereas GRG content is approximately 55%. The difference is linked to the gravity recovery effort, which will be discussed in the next section.

The economic impact of gravity recovery is particularly significant at Troilus or for Troilus like ores because of the flotation circuit high upgrading ratio, with the potential of significant loss of gold in the cleaning circuit and the higher net smelter return (NSR) of gold recovered into bullion. The initial plant design incorporated a small CIL circuit to treat approximately 500 Vd of cleaner tails. The circuit was briefly run, but it soon became apparent that not enough gold was recovered to justify its operation, which was discontinued.

Obviously, for greenfield applications, aggressive gravity recovery could be used to minimize or altogether eliminate the need to cyanide a middling stream.

**Battle Mountain -Phoenix Pilot Plant Sample:**

This sample is presented here not as representative of the full ore body, but as an example of a Au-Cu ore with a high GRG content, what can be achieved in a pilot plant, and a caveat of sampling problems. The sample was extracted by selecting three locations from a pilot plant head sample of a few tonnes, and taking a representative sample of each, to obtain a total weight approaching 100 kg.

The very high GRG content, shown in Figure 6, was first thought anomalous, because the calculated head grade was twice that of the pilot plant measured over a few days. Clearly, the sampling protocol was deficient, and a much higher number of sampling locations should have been used. However, the pilot plant gravity recovery, when optimized, ranged from 85 to 90%, very close to the calculated GRG content.

The ore head grade is around 0.10% Cu, which is too high for direct cyanidation (mostly because of the environmental liability and the presence of secondary copper minerals). A circuit flowsheet with very aggressive gravity recovery is ideally suited for this type of ore. Should high gravity recoveries be achieved, it would be much easier to operate the flotation circuit selectively to minimize smelting charges and maximize net smelter return. Aggressive or "extreme" gravity recovery will be discussed in the next section.

**DISCUSSION**

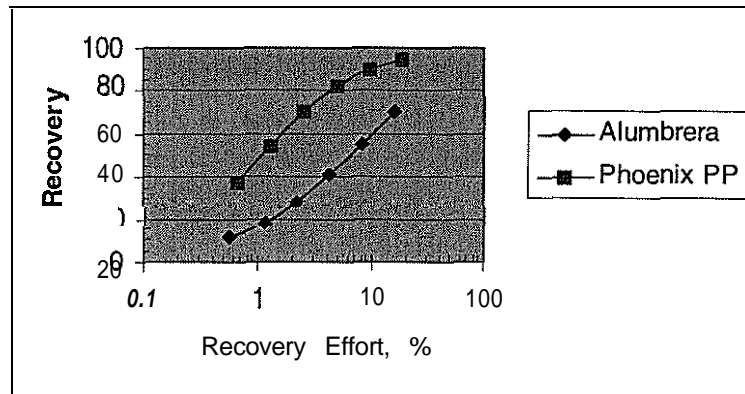
**Au-Cu Ores**

The four case studies presented are typical in that whole ore cyanidation was not retained. This flowsheet choice is the likely result of environmental constraints, especially where zero discharge cannot be implemented because of climatic considerations. A combination of gravity and flotation is then the most attractive route.

Actual plant recovery is a function of the GRG content and size distribution, and the gravity recovery effort, which is defined as the product of the circulating load treated by the primary gravity unit by the GRG recovery of this unit, corrected for gold room recovery. For example, let us consider Troilus: 5% of the circulating load is treated, and it is estimated that the screen-Knelson combination recovers 60% of the liberated GRG in its feed. Gold room recovery is assumed to be 90%, based on the use of the 12-in Knelson as a scavenger. Thus the

recovery effort is  $5\% \times \frac{60\% \cdot 90\%}{100\% \cdot 100\%} = 2.7\%$ . Should the recovery effort be doubled (as it is between

adjacent points in Fig. 7), the gravity recovery increase would be modest. This is shown in Figure 7 for the finest and coarsest GRG distributions presented here (simulation based on Laplante et al, 1995). Details of the simulation are shown in Appendix 1.



**Figure 7** %GRG Recovered as a Function of the Recovery Effort for the GRG Size Distributions of Alubrera and Phoenix Pilot Plant Samples (Details in Appendix 1)

The extent of the gravity recovery effort should be based on economic and metallurgical considerations. Generally, the higher the gold grade and the flotation upgrading ratio (i.e. the lower the Cu head grade), the higher the gravity incentive. Even when part of the flotation tails is cyanided, gravity recovery can still have a significant economic impact. Typically, gold payments for flotation concentrates are both partial and delayed.

Gold recovered by gravity receives a very high payment and can be delivered quickly. The difference in NSR present value can be as high as 8%, a strong incentive for gravity recovery.

### GRG Content vs. Plant Gravity Recovery

The GRG test is carried under controlled and optimum conditions<sup>1</sup>, to insure its reproducibility and validity. As a result, it yields a target that can be approached, but never reached. Generally, the coarser the GRG, the lower the gangue density, the higher the recovery effort and gold room efficiency, the closer plant recovery will approach the GRG content. All four variables are important and all are highly variable, although the last three are design variables that can be set to match the economic incentive of gravity recovery. As a result, plant gravity recovery has been measured to be as low as 20% and as high as 85% of the GRG content. For coarse GRG and low gangue specific gravity, recoveries of two thirds of the GRG are often encountered, even when gravity recovery is not fully optimized. Optimized circuits can yield GRG recoveries exceeding 80%.

The example of Kundana Gold Mine is instructive. Figure 8 shows that GRG at Kundana is both plentiful (86% of the gold in the ore) and relatively coarse (42% GRG above 106 µm). Primary recovery is effected in a 20-in Knelson that processes about 120 t/h (rated capacity: 20 t/h). Typical plant gravity recoveries averaged 60% before a Gekko InLine Leach Reactor (ILR) was installed to process table tails (i.e. about two thirds of the GRG). The increased recovery from the ILR has not been precisely accounted for (the liquor is directed to the existing electrowinning cell), but overall recovery has increased 1.5%, which strongly suggests that the ILR's contribution to "gravity" recovery is significant. Even a contribution as low as 15% would push gravity recovery to about 75% of the gold, or 85 to 90% of the GRG.

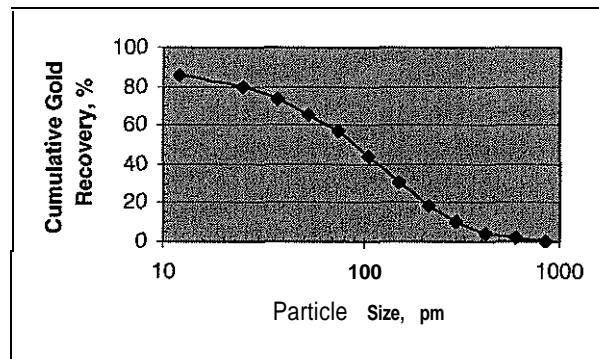


Figure 8 Cumulative GRG Distribution at Kundana

### “Extreme” Gravity Recovery

The previous section raises the question of how close the GRG content can be approached in full plant operation. The question is closely linked to the very difficult challenge of estimating a priori rather than a *fortiori* the economic benefit of gravity recovery. The economic benefit of gravity recovery most easily predicted is the increased NSR of gravity gold over gold reporting to a flotation concentrate. Actual increases in overall recovery are difficult to predict, and even when detailed metallurgical data covering the time period before and after the commissioning of a retrofit gravity circuit are available, difficult to back-calculate. With flotation circuits, increases of 3% in overall recovery for simple Cu ores for with feed grades of 0.3% Cu or with Zn-Pb have been reported (Laplante, 1987). Similar increases in NSR are also easily achieved, for example with a gravity

recovery of 40% and an increased NSR of gravity over flotation of 7.5% ( $3\% = 40\% \times \frac{7.5\%}{100}$ ). This combined benefit of 6% would constitute a very significant economic incentive for gravity recovery.

Where the gravity recovery incentive is high, how can the envelope of gravity recovery be pushed? No universal circuit can be proposed, but the following considerations should guide design:

1. A very high proportion of or the entire circulating load should be treated.

<sup>1</sup> Optimum feed rate and fluidization flow, short recovery cycle and sequential liberation and recovery to minimize over-grinding.

2. Centrifuge units should be fed at high feed rate to maximize how much gold is recovered in each unit.
3. The usual bleed from the cyclone underflow fed to the centrifuge(s) and by-passing the ball mill would increase pumping costs; treating ball mill discharge becomes the preferred option.
4. Careful screening of the Knelson feed becomes important, and a stand-alone vibrating screen may be preferable. Magnetic separation before the centrifuge units could be attractive, especially downstream of a Sag mill.
5. Cyclone performance should be optimized to maximize the gold circulating load.
6. Gold room recovery should be maximized.
7. The availability of the gravity circuit should be maximized.

The first two points stem from economic considerations: the cheapest way to achieve a given recovery effort is to feed a high proportion of the circulating load to units that have a high feed rate, even at the expense of stage recovery. This minimizes the number of centrifuge units needed. For a recovery effort approaching 15%, at least half the circulating load should be used. If half the cyclone underflow is bled and by-passes the ball mill, half the material is pumped and cycloned without any comminution, an obvious lack of efficiency. Recovery from the mill discharge rather than cyclone underflow eliminates this problem. The Kundana operation embodies the first three principles proposed: all of the ball mill discharge is treated in a Knelson Concentrator fed at many times its rated capacity (i.e. a 20-in Knelson fed at 120 t/h).

The fourth principle is widely applicable. For example, 20-in and 30-in Knelson Concentrators can come with their own screen fitted above the unit. This may be economical, but it is not optimal when the gravity incentive is high. A vibrating screen could present more tonnage to the Knelson at the optimum size distribution. For fine GRG recovery in a high specific gravity gangue, the screen could deliver to the Knelson a feed finer than the usual 100% -1.7 mm. If significant tramp iron or magnetite is present in the ore, it is better to remove it from the unit feed. Low intensity magnetic separators are cheap and easy to operate. At Omai Mines, they serve a very useful duty removing tramp iron from the concentrate of the Reichert cones used to upgrade the Knelson feed.

Figure 7 also shows that gravity recovery reaches 90% of the Phoenix GRG content when the recovery effort equals 10%. This is contingent on a cyclone performance that keeps GRG in the circulating load. For the Phoenix pilot plant work, cyclone performance was excellent, as even below 25 µm 88% of the GRG was directed to cyclone underflow. Clearly the link between projected gravity recovery, the recovery effort and the GRG content must take classification efficiency into account.

Gold room recoveries have been reported as low as 50%, and have been measured as high as 95.97%. Having a gold room recovery of 50% is worse than cutting the primary unit recovery by half, as the gold lost in the gold room is generally fine and more likely to exit the grinding circuit than the gold recovered. Intensive cyanidation and direct electrowinning was practiced in South Africa, and should be revisited not only because it provides extremely high recoveries, but also because an automated unit like the ILR makes it applicable to smaller plants and eliminates all manipulation of the Knelson concentrate.

The seventh point is often overlooked. Figure 5 shows that when the recovery effort drops by half, gravity recovery does not drop by nearly as much (this explains why many ineffective circuits are still tolerated). However, if the gravity circuit is fully shut down, gravity recovery drops by 100%. Poor operational availability is often the most significant source of poor gravity performance. Thus gravity circuits should be designed with reliability and flexibility in mind, especially when parallel units are used. For example, piping should make it possible to direct screen undersize to more than one centrifuge unit, and vice-versa. Gravity circuits should have enough headroom to make maintenance easy and minimize the risk of nearly horizontal pipes that often sand. When comparing two centrifuges with similar duties (e.g. Knelson vs. SuperBowl), reliability and the ability to accept high feed tonnages should be a critical issue. A vibrating screen is much more likely to fail or require maintenance than a Knelson, and choosing the right screen and installing it properly may have a very significant effect on circuit availability.

## CONCLUSIONS

Typical Au-Cu projects are likely to use a combination of gravity and flotation that makes the economic impact of gravity recovery critical, because gravity affects significantly both metallurgical and economic recovery. The optimal economic gravity recovery effort may vary from none to "extreme gravity", whereby a very aggressive gravity approach should be used. This contribution showed that typical scoping work, whilst essential to understand ore variability, is not ideally suited for generating the best estimate of full scale recovery or the data needed for optimal gravity circuit design. A GRG test performed on a single or very few composites serves both purposes and complements typical scoping work very effectively. There is a growing body of

evidence that suggests that aggressive gravity recovery can yield full-scale performance approaching the GRG content.

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## APPENDIX

When gravity recovery takes place at the discharge of the secondary mill, the recovery of GRG in each size class, D, can be calculated as follows:

$$D = R * (I - B * C * (I - R))^{-1} * F$$

Where

R:	diagonal matrix of the size-by-size stage GRG recovery
I:	identity matrix
B:	lower triangular matrix describing the grinding of GRG
C:	diagonal matrix of the size-by-size GRG partition curve
F:	column matrix of the size-by-size GRG content

Matrices R, B and F are fractional; matrices D and F are either fractional or in percent of the total gold content in the ore. The recovery matrix in this case includes three sub-processes, bleeding of part of the mill discharge to the primary unit, GRG recovery in the primary unit and GRG recovery in the gold room. For the simulation, the following information was used:

Table 4 Diagonals of R and C; F Matrices (in %) for Phoenix, Alumbrera

Size, $\mu\text{m}$	R	C	Phoenix	Alumbrera
600-850	0.384	0.999	14.1	0
425-600	0.493	0.999	7.0	0
300-425	0.585	0.999	8.2	0
212-300	0.626	0.999	9.0	0
150-212	0.686	0.999	10.2	1
106-150	0.739	0.999	10.1	1
75-106	0.749	0.998	9.9	2
53-75	0.713	0.995	6.9	4
37-53	0.648	0.990	5.5	5
25-37	0.520	0.970	3.9	11
-25	0.372	0.900	3.8	26

The GRG recoveries given in Table 4 were multiplied by the fraction of the mill discharge treated (from 0.01 to 0.32) to calculate the actual R matrix. The recovery effort was calculated as the fraction of the GRG fed to recovery that was actually recovered, for all size classes. The B matrix, taken from Vincent (1997), is as follows:

Size, $\mu\text{m}$	600-850	425-600	300-425	212-300	150-212	106-150	75-106	53-75	37-53	25-37	-25
600	.945										
425	.036	.954									
300	.004	.031	.960								
212	.003	.004	.027	.966							
150	.002	.002	.003	.023	.971						
106	.002	.002	.002	.002	.020	.975					
75	.001	.001	.001	.002	.002	.017	.979				
53	.001	.001	.001	.001	.001	.002	.014	.979			
37	.001	.001	.001	.001	.001	.001	.001	.014	.975		
25	.000	.001	.000	.001	.001	.001	.001	.001	.010	.980	
-25	.001	.001	.001	.002	.002	.002	.002	.002	.002	.007	.980