Predicting gravity separation gold recoveries

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Abstract — ***A novel methodology for estimating gold recoveries in gravity-separation circuits is presented. The methodology makes use of a population-balance model. The model represents gold liberation, breakage and classification behavior and applies preconcentration and recovery-performance curves to gravity-recoverable gold. The derivation of the necessary parameters is explained and demonstrated using data from an actual gravity circuit and from a projected application.***

The methodology makes it possible to estimate the performance of a gravity circuit having a large gold circulating load and a relatively-low unit recovery (e.g., a jig) or having a high recovery on a bleed of the circulating load (e.g., a Knelson Concentrator).

Introduction

Predicting the gold recovery of a gravity circuit is, at best, a difficult task. A very expensive alternative is to treat a large sample (typically ≥10 tonnes) in a pilot plant having a flowsheet similar to the one projected. However, even this approach is difficult when only part of an actual plants recycling load is to be bled and fed into a gravity unit such as a Knelson Concentrator (KC).

The operation of a pilot plant may not only be prohibitive because of the high cost, but there also may be insufficient sample available. A second approach (used for retrofits) consists of inserting a small-capacity unit into the existing circuit (typically a circulating load) and using the data obtained to predict the gold recovery (Ounpuu, 1992). The difficulty in this approach is again tied to the nature of the circulating load, which never reaches the appropriate steady-state.

In this report, an approach based on population-balance modeling is presented. The laboratory and/or plant data required for the model are reviewed, and case studies are given. Weaknesses of the approach are also evaluated.

Theory

Consider the grinding circuit depicted in Fig. 1. Free gold is generated as the material is ground in the mill. At the discharge of the mill, the free gold is represented as a column matrix $\underline{F}$. A preconcentration step yields a proportion (p_i) of each size class (forming diagonal matrix P) that is then presented to a gravity separator. From each size class, a free-gold recovery of r_i (forming diagonal matrix R) is achieved. Material not presented to the gravity unit (or not recovered from it) is then classified with a fraction c_i (forming diagonal matrix C) being returned to the mill. In the mill, a fraction h_{ii}' of the free gold in size class i remains in size class i in the mill discharge, but fractions h_{ij}' (j = i + 1 ton) report to the (finer) size classes i. Given the above description, it can be shown with basic linear algebra that:

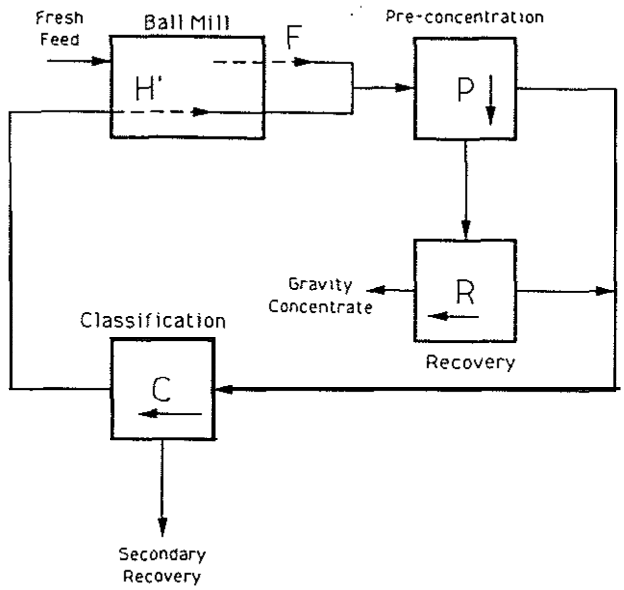

Fig. 1 — Block diagram of primary-gold recovery within a grinding circuit.

$$\underline{D} = PR \cdot [I - HC(I - PR)]^{-1} \cdot \underline{F} \qquad (1)$$

where $\underline{D}$ is a column matrix of the free-gold flowrate into the concentrate (the sum of the d_i gives the total gold recovery). Although Eq. (1) looks formidable, it is readily computed using a spreadsheet.

The above equation is easily derived. But how are its various components evaluated? Answers to this question are case specific. For example, retrofit applications will be able to take advantage of the existing grinding circuit to reliably generate much of the data. Green field applications may have to rely on data "borrowed" from other operations, resulting in a decrease. in reliability.

Matrix F. Matrix $\underline{F}$ (elements f_i) shows what amount of the gold is gravity-recoverable in each size class. $\underline{F}$ is expressed as a fraction of the total gold in the ore (either in %, g/t or oz/st). Summing the f_i (= F_t) yields the total amount of gravity-recoverable gold. The ratio of F_t over the total gold feed grade (F_g) is the mass fraction of recoverable gold. To determine $\underline{F}$, a sample of ore is taken and sequentially broken down to the final (cyclone-overflow) size. After each incremental size reduction, the sample is processed with a laboratory Knelson Concentrator (LKC) to recover its free-gold

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content. The Knelson Concentrator tail (in total or in part) is then ground for further gold liberation and recovery. The rationale for this approach is to minimize overgrinding of the free gold, which would occur if the material where ground directly to the final size. Overgrinding would reduce the gold recovery and would provide an inaccurate estimate of the free-gold sizedistribution. Details of the experimental procedure and the results of a number of tests are presented and discussed in Woodcock and Laplante (1993).

Table 1 Liberation data for AELRD ore (%R: stage LKC recovery; %U: % of the total gold in the ore)

Size (μm)	First Stage %R	First Stage %U	Second Stage %R	Second Stage %U	Third Stage %R	Third Stage %U	Total $\%U_{tot}$
840	13	0.0					0.0
600	1	0.0	31	0.0	•		0.0
420	1	0.0	40	0.1			0.1
300	16	1.0	41	0.6			1.6
210	7	0.4	14	0.3	99	2.4	3.1
150	19	1.5	32	2.0	94	2.2	5.7
105	24	1.3	34	1.9	65	1.6	4.6
75	20	1.7	44	4.0	51	2.6	6.3
53	13	0.7	37	2.1	35	1.5	4.3
37	14	1.1	53	6.3	26	2.6	12.2
25	26	1.8	33	3.2	60	6.9	11.9
-25	1	0.2	6	2.0	8	2.2	4.4
Total		9.9		24.3		22.5	56.7

Table 1 shows the result of sequentially crushing/grinding and processing (through an LKC) a 25-kg sample of ore from Agnico Eagle, La Ronde Division (AELRD). Free-gold recovery and total-gold content are calculated from the size-by-size assays of the three LKC concentrates and third-stage tail. Because the LKC concentrates are fully assayed and because the third-stage tail is fine (generally 80% -75 μm), these assays are reliable. Table 1 shows that the total free-gold recovery is 56% and shows that only 10% of this is achieved after crushing to 100% -850 pm (20 mesh). Most of the free gold first appears below 210 μm (70 mesh). Table 1 shows that the LKC recovery for -25 μm material is very poor, which is typical of ores with a high gangue density. This recovery progressively increases with decreasing feed size (from Stage 1 to Stage 3).

Matrices P and R. P and R are diagonal matrices expressing the probability that free gold in size class i will first be fed (preconcentrated) to the separator and will then be recovered. Both are set when designing a gravity circuit by the selection and size of concentration equipment. Because of the high density of the gangue at AELRD and the relatively fine size of the gold, a 300-μm (50-mesh) screen could be used to remove. the relatively barren oversize. This would bleed part of the feed effectively, increase the total- and free-gold contents of the Knelson Concentrator feed and increase the Knelson Concentrator efficiency significantly (Laplante, Putz and Huang, 1993).

Matrix P is therefore a measure of screening efficiency. Two models are most frequently used to describe size-by-size efficiencies. These are the Karra model (Karra, 1979; Mular and Bhappu, 1978) and the Whiten model (Whiten, 1972: Walter and Whiten, 1977). A third model that can also be used is the Valliant model (Gottfried, 1978; Goodman and McCreery, 1980; Gottfried arid Abara, 1979). Although this third model was developed for coal preparation, the "sluggish" performance curves it yields may more appropriately represent the behavior of fine screens that would probably be overloaded for this application. No data are available as of yet for these fine-screening applications, although even finer plant screens (105 μm) have been shown to provide very selective separation. We will assume, for the time being, the following Karralike separation efficiency:

$$o_i = o_f + \left(1 - o_f\right)\cdot\left[1 - e^{-0.693\left(\frac{x_i}{D'_{50}}\right)^m}\right] \qquad (2)$$

where:

o_i is the fractional recovery to the screen oversize,
o_f is the fraction short-circuited to the oversize,
D',, is the corrected cutsize of the screen, and
m is a measure of separation sharpness, which is assumed to be equal to four.

This is a conservative value, and Plitt and Flintoff (1985) suggest a default value of 5.846 for simulation. 0, can be adjusted to achieve an undersize flow rate of 30 t/h, for which the Knelson Concentrator performance for free-gold recovery is relatively known (Laplante, Putz and Huang, 1993). Since P is the matrix of the size-by-size fraction presented to the separation unit and 0 is the matrix of the fraction reporting to the oversize, it follows that $p_i = 1 - o_i$.

Screens are not the most common preconcentration units (although with a KC, a 1.7-mm screen is almost always used). When one cyclone underflow out of a cyclopak is directed to the recovery unit (which is occasionally the case with a KC), all p_i are equal to 1/n, where n is the number of cyclones of the cyclopak (assuming a constant cyclone-undefflow rate for all cyclones). When a jig is used, all of the stream is often selected for treatment, and, in this case, $p_i = 1$. In KC circuits, sluices are occasionally used (Laplante, Liu and Cauchon, 1990). In this case, pi = Y * u_i, where Y is the yield of the sluice, and u_i is the upgrading ratio of size class i. Because u_i is generally low (and in fact approaches unity below 210 μm), sluices are not recommended for most applications.

An interesting feature of the model is its ability to calculate the free-gold circulating load when not using a gravity circuit (by setting PR = 0). This can be compared to actual circulating loads to validate at least part of the model.

Classification efficiency curve C. C is the classification efficiency curve of the cyclones for free gold. It is calculated from the analyses of three or four of the cyclone streams (typically the underflow, overflow and one or two feeds). Using a procedure developed by Laplante, Putz and Huang (1993), each sample is processed with a laboratory KC (LKC) to determine the free-gold content. The curve is described with Plitt's model (Plitt, 1976). which is in fact similar to Karra's model (Eq. (2)). In Karra's model, the fraction short-circuited also reports to the coarse product (in this case, the cyclone underflow). Separation sharpness is generally lower for gold than it is for the overall ore (as is the D_{50}).

C is an important factor in estimating gold recovery by gravity in most circuits. This is because fine gold that can be recovered by gravity, typically with a PKC, grinds very slowly (with the probabilities of surviving any given pass in the secondary mill equal to 98% to 99%). It is therefore primarily removed from the circuit via either the cyclone overflow (with probabilities of 10% to 20%) or the gold gravity concentrate (where the probability of fine-gold recovery is extremely unit dependent and may be as low as I % for jigs).

Matrix H. The H matrix is probably the most difficult function to estimate in the present exercise. Because it is malleable, gold is ground at a rate noticeably lower than the overall ore (Banisi, Laplante and Marois, 1991). Hence, a separate estimate of S must be generated. Further, its breakage function is highly atypical of most minerals. Additionally, the losses of gravity-recoverable gold have to be estimated.

A typical population-balance grinding model is one that relates the size distribution of the discharge of a mill ($\underline{m}_d$) to the size distribution of the feed ($\underline{m}_f$), and the residence-time distribution in the mill and the breakage and selection function of the material (S and B). Of these parameters, S and B are the most critical. The model can be resolved into a simple equation of the following type (Austin et al, 1984):

$$\underline{M}_d = \begin{vmatrix} h_{11} & 0 & 0 & 0 & \cdots & 0 \\ h_{21} & h_{22} & 0 & 0 & \cdots & 0 \\ h_{31} & h_{32} & h_{33} & 0 & \cdots & 0 \\ & & & & & 0 \\ h_{n1} & h_{n2} & h_{n3} & h_{n4} & \cdots & h_{nn} \end{vmatrix} \times \underline{M}_f \qquad (4)$$

In this equation, the h_{ij} are a function of the above parameters. Each h_{jj} (on the main diagonal) is the fraction which both enters and exits in size class j. The terms below each h_{jj} (h_{ij}, $i>j$) is the fraction of material which enters the mill in size class j and leaves in size class i. Because we assume that no material can exit the mill in a size class coarser than the one in which it entered ($h_{ij} = 0$ for $i<j$), the upper triangle of the matrix is null. Note that, unlike most applications, we include in the matrix the finest size class, i.e., the pan(n), to model its gravity recovery. Because of mass conservation, the sum of each column should be equal to one. However, since gravity-recoverable gold is modeled (not total gold), we must make provisions for gravity-recoverable gold to become unrecoverable, due to smearing, overgrinding or excessive flaking. As a result, the sum of each column will be less than one.

To estimate the breakage function of the gravity-recoverable gold, a procedure was developed whereby the gravity-recoverable gold in a single size class is first isolated (using a LKC) and then ground for a short time (to minimize secondary breakage) in a laboratory ball mill with barren silica sand. After grinding. the product is screened, and all of the material that is finer and coarser than the initial size class is removed and replaced with an equivalent mass of silica sand. This procedure is repeated ten times with the intermediate and final products being processed with the same LKC. The concentrate and tails are then analyzed. The distribution of gold in the mass finer than the original gold-bearing class and the gold recovery by LKC yields an estimate of the breakage function (B) and an estimate of the amount of gold that is gravity-recoverable, Details of the work are in Laplante and Noaparast (1994).

Table 2 shows the result of this procedure for three size classes of Casa Berardi (CB) and one of Cominco's Snip Operations (CSO). The CSO results include a repeat. Results are somewhat erratic, especially between the two CSO repeats of the 210 to 300 µm size. Additionally, there is a significant fraction of the gold that is flattened and actually reports to the size classes coarser than the original (most report to the adjacent coarser size class).

The following conclusions can be drawn:

- For material remaining in the size class, between 99% and 100% can be recovered by gravity (with the possible exception of the 38 to 53 pm size class from CB).
- For material finer than the original size class, between 75% and 90% will report to the adjacent finer size class with a high recovery (typically 98%).
- For material coarser than the original size class, the recovery is also high (from 95% and 100%).
- With decreasing particle size in the progeny, the recoveries decrease down to 10% to 92% in the -25 µm size.

The above results are fragmental, but recent work (Noaparast and Laplante, 1994) has confirmed its general trends. For purposes of this work, the following assumptions were made:

- Panicle migration to coarser size classes is neglected.
- The gravity-recoverable gold for the -25 µm progeny is assumed to be 50%.
- The gravity-recoverable gold broken into the adjacent (next lower) size class is assumed to be 98% (except for the 25 to 37 µm size).
- The relationship between the gravity-recoverable fraction and the geomean particle size is assumed to follow a modified Rosin-Rammler equation. The value of m and D_{50} is adjusted to comply with the second and third assumptions above.
- The breakage function is assumed to be normalizable and is the averaged results of the five tests.
- The gravity-recoverable gold that is unbroken is assumed to be 100% gravity recoverable at the discharge of the mill.

Figure 2 shows the resulting breakage function (B). The results of Banisi, Laplante and Marois (1991) are shown for comparison.

The H matrix must then be calculated. It is proposed that the selection function of the gravity-recoverable gold for a given circuit be based in accordance with the findings of Banisi, Laplante and Marois (1991) at Hemlo, where 95% of the secondary ball mill feed was amalgamable, and up to 91% could be recovered by a Knelson Concentrator. They found that, at the coarser end of grinding (1.2 mm, 14 mesh), the selection function of gold was 20-times lower than that of the ore, and, in the finer range (37 µm, 400 mesh), it was six-times lower. Other selection functions can be calculated assuming a log-linear function with particle size. The retention time can be either measured or assumed (its impact is not significant, provided that a consistent estimate is used according to Laplante, Finch and del Villar (1987). The selectionfunction is then used with the breakage function of gravity-recoverable gold to calculate a transfer matrix H.

First case study: Casa Berardi

The Casa Berardi grinding flowsheet, at the time of sampling, consisted of a simple primary-SAG mill in open-circuit and a ball mill in closed-circuit with two stages of cycloning (with both underflow repotting to the ball mill).

Table 2 — Breakage Function Determination for Gravity-Recoverable Gold (C5: 150-210 µm; C7: 75-105 µm; C9: 37-53 µm; C4: 210-300 µm; %R: stage LKC recovery; %U: % of the total gold in the ore)

Size	CB-C5		CB-C7		CB-C9		CSO-C4a		CSO-C4b	
(µm)	%R	%U	%R	%U	%R	%U	%R	%U	%R	%U
+300	66.8	0.3	-	-	-	-	100.0	44.0	99.6	15.3
210-300	95.9	2.4	95.4	0.4	90.2	0.2	100.0	35.8	99.4	48.9
150-210	99.8	60.4	91.6	0.9	90.3	0.4	100.0	18.0	98.6	27.0
105-150	98.4	16.8	98.3	4.3	90.6	0.4	100.0	0.8	74.2	2.2
75-105	92.4	2.6	100.0	76.7	89.5	0.7	100.0	0.6	90.6	1.1
53-75	87.7	2.4	99.6	11.7	95.7	8.0	94.8	0.3	93.4	0.6
38-53	61.2	3.1	95.3	1.6	95.7	77.9	80.5	0.2	61.0	0.8
25-53	53.4	3.7	97.0	1.5	96.6	9.9	66.8	0.2	32.5	0.6
-25	50.3	8.4	92.0	3.0	76.9	2.5	48.5	0.2	11.9	3.4
Total	91.9	100.0	99.3	100.0	95.2	100.0	99.8	100.0	94.8	100.0

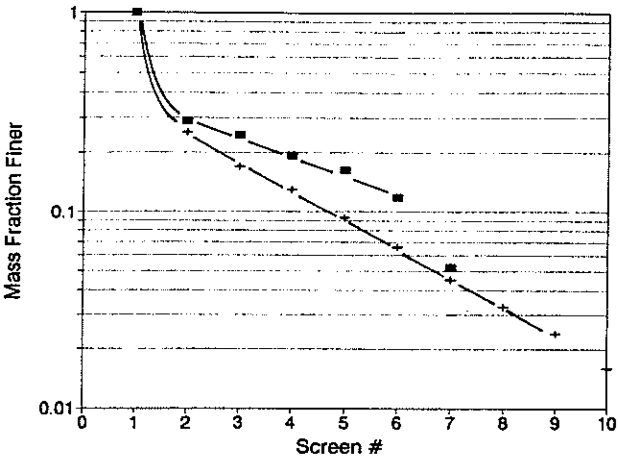

Fig. 2 — Breakage function used for this work, (squares) compared to the results of Banisi, Laplante and Marois, 1991 (crosses).

Table 3 — Gold Size Distribution in the Casa Berardi Sample (Woodcock and Laplante, 1993)

Size, µm	Gravity recoverable, %	Non gravity recoverable, %
420 – 600	0.2	—
300 – 420	0.3	—
210 – 300	1.2	—
150 – 210	2.2	0.3
105 – 150	4.7	0.3
75 – 105	12.0	2.2
53 – 75	11.0	2.8
37 – 53	12.8	4.0
25 – 37	8.9	2.3
– 25	18.6	16.2
Total	71.9	28.1

Table 4 — Diagonals of the P (preconcentration) and R (recovery) matrices

Size, µm	P	R
+420	0.000	0.55
300-420	0.000	0.55
210-300	0.034	0.68
150-210	0.105	0.77
105-150	0.139	0.84
75-105	0.150	0.89
53-75	0.152	0.89
37-53	0.153	0.88
25-37	0.153	0.86
-25	0.153	0.70

Samples of the mill discharges and cyclone underflows and overflows were extracted to establish a size-by-size mass balance for total and gravity-recoverable gold through the circuit. Results showed a large circulating load of gold (5800% compared to 517% for the actual ore) with the ball mill discharge assaying 80 g/t and the feed assaying 7 g/t. The primary-cyclone underflow (114 g/t) showed the highest upgrading. Agar (1993) studied the same circuit and obtained very similar results.

Matrix F. The sequential grinding/recovery test yielded a gold recovery of 6.8 g/t out of a feed of 9.5 g/t (an estimated 71.6% gravity-recoverable gold). Table 3 details the size distribution of the gravity-recoverable gold. Table 3 shows that one-fourth of it is -25 µm, which would be difficult to recover for units with poor fines recovery capability.

Matrices P and R. The diagonal matrices of P (preconcentration) and R (recovery) are shown in Table 4. To estimate P, the R_0 of Karra's equation was allowed to vary (with $m = 4$ and $D_{50} = 200$ µm, the equivalent of a 50-mesh screen) to achieve, with the known ball mill discharge flow of 345 t/h, a Knelson Concentrator feed rate of 35 t/h. The R values were taken from Laplante, Putz and Huang (1993) for a PKC operating at 35 t/h. They may be somewhat conservative, since the unit was processing -1.7-mm feed (rather than -0.21-mm feed).

Classification efficiency curve C. The values of C are estimated from the actual circuit. Since both primary- and secondary-cyclone underflows are directed to the ball mill, the performance curves for both stages were determined and combined:

$$C_i = C_{i1} + \left(100\% - C_{i1}\right) \times \frac{C_{i2}}{100\%} \qquad (4)$$

where c_{i1} and c_{i2} are the recovery to the underflow for the primary and secondary stages, respectively.

Although the gravity-recoverable data were slightly noisy, they were in good agreement with the total-gold data, especially in the range of interest (<300 µm). Total-gold data will be used to estimate the gold circulating load, albeit somewhat conservatively. Figure 3 shows the joint recovery to the underflow of the two cyclone stages, which is particularly large (97% to 100%) above 25 µm. The performance curve of the ore is also high, resulting in a high circulating load (>500%).

Matrix H. Figure 4 shows the measured selection function of the ore and the total gold. The selection function of gravity-recoverable gold, estimated from that of the ore and Banisi, Laplante and Marois (1991), is also shown. The residence-time distribution used to calculate H was that of two small perfect mixers (PM, 20% of the total mean residence time, t) in series with one large PM (70% oft) and one plug-flow unit (10% of t). Detailed results are found in Laplante (1993b).

Predicted circulating load and gravity recovery. Table 5 shows the predicted and measured loads. The predicted circulating load was obtained by setting PR equal to zero. Both loads are high, but the simulation can only predict approximately 75% of the measured load. The predicted load is higher than the measured load in the two finest sizes (the sizes below 37 µm, 400 mesh). This is probably due to some of the gold becoming unrecoverable without being ground out of these two size classes, a phenomenon which is not accounted for in the present model.

Gold recovery was predicted for two cases: the first case being the combination of a screen at the ball mill discharge (BMD) and PKC operated at 35 t/h, and the second case being the PKC operated at 20 t/h from a bleed of the BMD (6% of the circulating load). Results are given in Table 6. What is shown is that, whereas the screen-PKC can achieve close to the maximum recovery (given by the gravity-recoverable content of the ore), the PKC alone fails to recover a significant fraction of the gold (with only about one-third being recovered). Both arrangements recover significant gold values.

Second case study: Meston resources

This second study is shorter than the first because it follows essentially the same steps, and much of the circuit data have already been reported in Laplante and Shu (1991). Table 7 shows the matrix F, the diagonals of the matrices R (Laplante et al., 1994) and C used for the calculations. P was set to a simple bleed of 11% of the circulating load. This was because the sluices used were found to provide no detectable upgrading below 210 pm, where most of the gravity-recoverable gold was found (Laplante, Liu and Cauchon, 1990). The C matrix was estimated for free gold and shows some noise, as indicated by the fact that the recovery of the -25 µm is actually above that of the 25 to 37 µm. The selection function of the ore was measured and that of free gold was estimated from Banisi, Laplante and Marois (1991).

Results are shown in Table 8. Table 8 compares the circulating load at the cyclone underflow, which was slightly higher for the simulation (509% vs. 461%). Similar results were obtained for the ball mill discharge (569 vs. 544%). The simulated gold recovery is 48%, which compares well to the final gravity gold recovery of 43%, reported for October 1993.

Discussion

Appiicationsofthemethodology. The methodology can be used to assess the impact of gravity recovery, to design flowsheets, to select and size units and to optimize gravity circuits. Table 6 can be used as an illustration. It shows that a combination of screen and Knelson separation can recover 64% out of the 72% of the gold that is recoverable. Mostly because of secondary upgrading, the actual gold recovery would be lower (e.g., increasing the concentrate grade from 1% to 50% Au). However, a 50% gold recovery (with a cleaning recovery of 78%) does not seem unrealistic, given a typical gold room performance of 80 to 90% gold recovery (Laplante, Liu and Cauchon, 1990). It also shows that a PKC alone, especially if operated below its normal capacity, would not perform as well, largely because the recovery in the finer sizes would be lower.

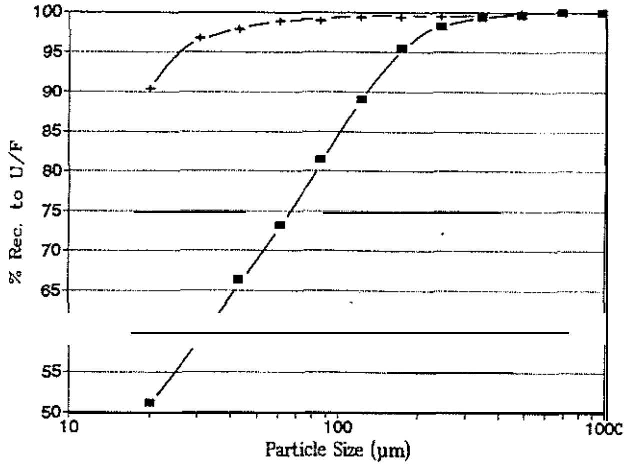

Fig. 3 — Combined performance curve of the primary and secondary cyclones at Casa Berardi (squares: are; crosses: total gold).

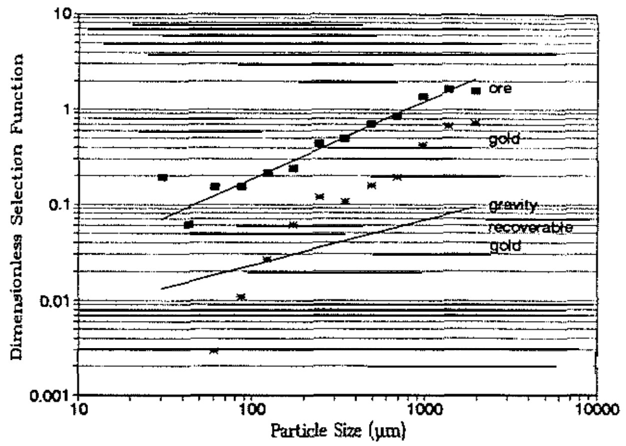

Fig. 4 — Dimensionless selection function at Casa Berardi.

Limitations of the approach. The proposed approach remains to be extensively validated. This will be accomplished in the near future when the procedure is applied to plants with gravity-recovery circuits or when circuits for which predictions have been made are installed. Already, the ability of the model to simulate a significant fraction of the circulating load (about 75% in the case of Casa Berardi) is encouraging because it indicates that the phenomena responsible for the buildup of gold in the circulating load are reasonably accounted for. Failure to predict the full circulating load is not a problem and could well be construed as a "safety factor" in the prediction of gold recovery. Overprediction of the circulating load in the two finest size classes could lead to an overestimation of fine-gold recovery. Because the model does predict gold recovery on a size-by-size basis, it will be possible to verify this point specifically and account for it, either directly in the model or by correcting predictions.

A few of the concepts need refinement. The estimate of the selection function, for example, should be linked to a more. consistent parameter than the selection function of the ore (a specific-power index, for example). The database on

Table 5 — Predicted and measured circulating loads (100%: total gold content of the ore)

Size, µm	Measured	Predicted
+840	2	0
600-840	37	1
420-600	35	5
300-420	23	10
210-300	88	35
150-210	166	88
105-150	469	213
75-105	664	502
53-75	740	632
37-53	801	599
25-37	217	360
-25	126	235
Total	3430	2676

Table 6 — Predicted gold recovery for a 300 pm (50 mesh) screen and 76 cm Knelson at 35 t/h and the same unit operated at 20 t/h without a screen (100%: total gold content of the ore)

Size, µm	Screen+PKC	PKC
600-840	0.0	0.0
420-600	0.0	0.1
300-420	0.0	0.2
210-300	0.6	0.8
150-210	2.4	1.6
105-150	5.0	3.9
75-105	11.6	9.2
53-75	11.3	9.6
37-53	12.4	9.9
25-37	6.6	6.6
-25	12.1	6.5
TOTAL	64.0	46.6

Table 7-Free gold vector f (% of the total gold in feed) and diagonals of the matrices C (cyclone performance, fraction to u/f) and r (free gold recovery, %, of a 76 cm pkc) used for the simulation

Size, µm	F	C	A
+600	0.1	0.999	48
420-600	0.7	0.999	58
300-420	2.2	0.999	65
210-300	2.6	0.998	68
150-210	5.6	0.993	73
105-150	8.0	0.966	77
75-105	9.6	0.978	78
53-75	6.7	0.952	75
37-53	10.1	0.952	72
25-37	6.8	0.960	65
-25	12.9	0.840	62

Table 8 -Measured and predicted circulating load at the primary cyclone underflow (100%: total gold in ore)

size, pm	Measured	Predicted
+600	15	0
420-600	4	5
300-420	12	17
210-300	22	29
150-210	44	54
105-I 50	62	76
75-105	82	90
53-75	83	62
37-53	61	85
-37	76	92

the breakage function and recoverability of the progeny of gravity-recoverable particles must also be extended. Finally, recovery data R should be available for a variety of units, e.g., jigs, spirals and cones. An important uncertainty has to do with the very meaning of free-gold recovery. Is it a measure of the capability of the machine to recover all gravity-recoverable panicles, or is it a fraction of the gravity-recoverable gold? In other words, does a recovery of 70% (per pass) mean that the unit will recover 70% of all of the gravity-recoverable gold present (as determined with the existing lab procedure) or does it mean that the unit will always recover the easiest 70% of the recoverable gold and never recover the most difficult 30% of the recoverable gold? These questions will require additional work. But they are not much different than the questions that needed to be answered for other population-balance models that have been used for the scale-up and optimization of mineral processing plants.

Conclusion

A popularion-balance model to predict gold-gravity recovery was presented. The model includes the necessary concepts of liberation, grinding, classification and recovery. Although the model remains to be validated, it predicted the very large circulating loads observed in an actual plant with acceptable accuracy. This was accomplished even though one of the most important variables used, the amount of gravity recoverable gold, was derived from a laboratory test rather than from plant performance.

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