A Comparison of Knelson Concentrator and Jig Performance for Gold Recovery

by

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Abstract

A 7.6 cm Laboratory Knelson Concentrator was used to evaluate the performance of one jig circuit (Snip Operation), four Knelson Concentrators circuits (Meston, Est Malartic, Aurbel and Hemlo) and one classification circuit (Agnico-Eagle). To determine the size-by-size unit performance of all units, total and gravity recoverable gold contents were measured in the feed, concentrate, tails, underflow and overflow. Sample dilution with silica was used as a tool to enhance LKC recovery in samples with a high sulphide content.

Knelson performance was found to vary from plant to plant: overall gold recoveries by gravity were 35-40% for Meston, 30% for Hemlo, 25% for Aurbel and 20% for Est Malartic. All plant KCs proved capable of recovering gravity recoverable gold (GRG) over the full size range of the feed (25-850 μm) but all, except possibly Meston, demonstrated handicaps that limited their gold recovery. Those handicaps showed that gravity recovery was a function of the GRG content of the ore, the feed rate, the fraction of the circulating load treated and the recovery flowsheet. The high GRG stage recovery of Meston, 50-75%, compared to that of Est Malartic (16%), which treats a high gangue density ore, showed that Knelson performance was size dependent.

Size-by-size GRG recoveries were determined by using the difference of GRG content in the Knelson feed and tails. This method proved to be somewhat inadequate due to the variability of the size-by-size data, particularly when the Knelson performance was lower than 50%. A sample of the Knelson concentrate and a measure of its yield are necessary to evaluate recovery.

The behaviour of GRG in the Agnico-Eagle classification circuit was that 99.4% of it reported to the cyclone underflow compared to 98.1% of the total gold and 84.2% for the ore itself.

Snip is one of the few Canadian plants still using a jig for gold recovery. There was virtually no coarse gold in the ore. The overall jig performance in 1992 was found to vary between 2.1 to 3.1%, and then was increased to 3.7% in 1993 because of a yield increase. Total gold recovery was very high because of the circulating load, 3300%. However, the jig failed to recover fine GRG effectively as almost no gold (<1%) finer than 25 μm was recovered. The table rejected almost all the gold recovered by the jig, between 100 and 600 μm, because it was unliberated.

The data generated from the Knelsons and the jig was used in a model designed to simulate an actual grinding and gravity circuit, and to predict its GRG recovery. It describes gold liberation, breakage and classification behaviour, and the GRG recovery performance curve of the chosen gravity unit. The simulation of the Snip circuit reproduced the recoveries obtained at the plant, and predicted that the use of a 20" Knelson, replacing the jig, would bring the recovery from 33% up to 43%.

Résumé

Un concentrateur Knelson de Laboratoire (CKL) de 7.6 cm a été utilisé pour évaluer la performance d'un circuit gravimétrique de récupération de l'or avec un bac oscillant (Snip Operation), de quatre circuits avec concentrateurs Knelson (Meston. Est Malartic, Aurbel et Hemlo) et d'un circuit de classification (Laronde d'Agnico-Eagle). Pour déterminer la performance de chaque unité, les quantités d'or total et d'or récupérable par gravimétrie (ORG) ont été mesurées dans chaque classe granulométrique de leur alimentation, concentré. rejet. souverse et surverse. On a dilué les échantillons très riches en sulfures avec de la silice pour en maximiser la récupération d'or par CKL.

La performance des concentrateurs Knelson a varié d'une usine à l'autre. Les récupérations d'or par gravimétrie ont été de 35-40% pour Meston, 30% pour Hemlo. 25% pour Aurbel et 20% pour Est Malartic. Tous les CKs ont récupéré l'ORG sur toute la plage granulométrique étudiée, de +850 à -25 µm, mais tous les circuits, excepté celui de Meston, souffraient d'handicaps qui limitaient leur efficacité. La récupération gravimétrique dépendait de la quantité d'ORG, du taux d'alimentation, de la fraction traitée de la charge circulante et du schéma de traitement. La récupération d'étape en ORG de Meston. élevée (50-75%) comparée à celle d' Est Malartic (16%) où est traité un minerai très dense, a démontré que la performance du Knelson est affectée par une densité de gangue élevée.

Les récupérations en ORG ont été déterminées en utilisant la différence en ORG de l'alimentation et du rejet du Knelson. Cette méthode n'a pas été entièrement satisfaisante, à cause de la variabilité des analyses d'or, particulièrement quand la récupération du Knelson était inférieure à 50%. Un échantillon du concentré du Knelson et une mesure de récupération poids sont alors nécessaires pour bien évaluer la récupération.

L'étude du comportement de l'ORG dans le circuit de classification d'Agnico-Eagle a montré que 99.4% de ce dernier se rapportait à la souverse, tout comme 98.1% de l'or total et 84.2% du minerai lui-même.

Snip est l'un des concentrateurs canadiens qui utilisent encore un bac oscillant pour récupérer l'or. Il n'y avait pratiquement pas d'or grossier dans le minerai. La récupération d'étape du bac en 1992 variait entre 2 et 3%, pour grimper à 3.7% en 1993, à cause d'une augmentation de la récupération poids. La récupération totale de l'or était beaucoup plus élevée, à cause de la charge circulante de 3300%. Pourtant, le bac ne pouvait récupérer l'or fin (-25 µm) efficacement (moins de 1%). Presque tout l'or récupéré par le bac entre 100 et 600 µm était rejeté par la table à secousses parce qu'il n'était pas libéré.

Les données générées par les Knelsons et le bac oscillant furent utilisées par un modèle créé pour simuler un circuit de broyage et de gravimétrie, et pour prédire sa récupération en ORG. Ce modèle décrit la libération d'or, le comportement en broyage et en classification, et la récupération en ORG de l'équipement gravimétrique. La simulation a reproduit les récupérations observées en usine, et a prédit que l'utilisation d'un Knelson de 20", remplaçant un bac oscillant, pouvait augmenter la récupération de 33% à 43%.

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List of Abbreviations

BMF	ball mill feed
BMD	ball mill discharge
CO(F)	cyclone overflow
CU(F)	cyclone underflow
GRG	gravity recoverable gold
JC1	jig concentrate hutch 1
JC2	jig concentrate hutch 2
JTLS	jig tailings
KC	Knelson Concentrator
LKC	laboratory Knelson Concentrator
PCF	primary cyclone feed
PCO(F)	primary cyclone overflow
PCU(F)	primary cyclone overflow
PKC	plant Knelson Concentrator
SCO(F)	secondary cyclone overflow
SCU(F)	secondary cyclone underflow

Chapter One
Introduction

1.1 Gold Gravity Separation

Since antiquity, gold gravity concentration has always been very common due to its large capacity, low operating cost, freedom from chemical additives and ability to treat a wide size distribution. Gold's high specific gravity (19.3 when pure) compared to that of gangue minerals (2.1-5.0) makes the process very attractive, although gold particle shape, porosity and hydrophobicity can lower recovery[1,2,3,4].

Up to the first half of the 19th century, panning was the main gold recovery method, particularly in Russia, which supplied 60% of the world's gold production. In the second half of the 19th century, as a series of gold rushes swept the world (California, South America, Australia and New Zealand), gravity remained the dominant recovery method but other techniques were developed. In North America, panning was superseded by cradles and long toms (consisting of screens and sluices). During the country's gold rush era, new gravity concentration equipment was developed to treat a wide range of ore types on larger scales and was combined with amalgamation to recover gold as early as possible in a flowsheet. Despite advances in gravity concentration and amalgamation, the two processes were unsuitable for the recovery of fine gold and gold associated with sulphide minerals. These drawbacks prompted the search for an effective hydrometallurgical or pyrometallurgical process[5].

Cyanidation, the dissolution of gold in an aerated cyanide solution, proved to be the most successful process. Its commercial use began in 1889 and spread rapidly. The cyanidation process established hydrometallurgy as a distinct subject within mineral and metal processing. Gold precipitation with zinc was later introduced commercially for the treatment of cyanide leach solutions in 1890 and was subsequently applied widely in the industry. Commonly known as the Merrill-Crowe process, it evolved to be highly efficient with dissolved gold recoveries as high as 99.5% although 99% is more typical and some plants suffer from recoveries as low as 97-98% - a significant incentive to maximise gravity recovery. At the beginning of the 20th century, a typical flowsheet, particularly of those found in South Africa, included screening, manual sorting of waste rock, stamp milling, amalgamation, cyanide leaching, solid/liquid separation by filtration to produce a gold-bearing solution, and gold recovery by precipitation with zinc[5].

Flotation was introduced, between 1910 and 1930, for the treatment of base metal sulphide ores. It was quickly used for the recovery of free milling gold (since gold is naturally floatable without collector) and the recovery of gold-bearing sulphide for smelting or roasting followed by cyanidation (since with fine unliberated gold, <10 µm, associated with sulphide cyanidation performance is typically poor). Cocurrently, many advances in gravity concentration techniques were made, such as the use of jigs, Johnson drums and shaking tables within grinding circuits for the recovery of coarse gold. In 1922, direct amalgamation of the mill product was replaced in South Africa by the use of corduroy strakes, which preconcentrated the amalgamation feed and significantly reduced the amount of mercury used. The change was encouraged largely for health and safety reasons[5].

In the 1970s, the use of carbon circuits, with stripping, acid wash and reactivation, was first used at plant scale at the Homestake Gold Mine with dissolved gold recoveries reaching 99.5-99.8%. Although the ability of carbon to adsorb the aurocyanide complex had been known for a long time, the inability to desorb the carbon and the need to oxidise it had precluded its economic use until the 1950s. The solutions to that problem were found to be the processes: desorption and electrowining. Various applications of carbon adsorption (CIP, CIL, and CIC) now dominate the field of dissolved gold recovery. The ability of carbon to recover gold at low concentration contributed greatly to the success of heap leaching and the improvements of carbon technology to the point of replacing almost completely the zinc precipitation process, except where the Ag:Au ratio is particularly high[5,6].

Cyanidation and flotation advances have led to a decline of gravity technology. For example, in the 1980s, about 20% of the South African gold were produced from gravity concentration; in the early 1990s, gravity recovery had all but disappeared.

Unlike the gold ores of South Africa, the mineralogy of Canadian ores is highly variable, with a wide range of minerals often present, such as base metal sulphides, pyrite and arsenopyrite. The gold is finely disseminated and the ore must be finely ground, typically to 80% <75 μm, to achieve the liberation needed for satisfactory cyanidation. Gold dissolution is frequently in excess of 90% and sometimes exceeds 95% with a relatively long residence time of 30 to 48 hours. Carbon adsorption of dissolved gold has received widespread acceptance in recent years and zinc precipitation is now confined to older plants. In a typical CIP plant, gravity is not seen as beneficial for reasons such as installation costs and complexity, security risks, difficult sampling and metallurgical accounting procedures. Those drawbacks are usually combined with the perception that gravity does not increase overall recovery particularly when treating free milling ores[7].

Still, in recent years, this view has been challenged and gravity has regained attention and a role in many Canadian gold mills; typically, part of a ball mill discharge or a cyclone underflow is treated prior to flotation or/and cyanidation. Users of gravity concentration maintain that:

- The earlier the gold can be extracted, the sooner it is smelted, refined and sold, maximising the smelter return.
- Overall plant recovery can be improved by extracting coarse gold prior to the leach circuit where it may have insufficient contact for dissolution. It can also reduce the head grade of the cyanidation circuit feed and hence any potential for solution gold losses. A shorter leaching time can also be achieved.
- Overall plant recovery can be increased (up to 3%) by removing gold too coarse to float and flotation time to reach desired tailings grades can be reduced.
- The high gold circulating load of grinding circuits can be reduced. Build-up and overgrinding of the dense and malleable gold can be decreased.
- Low gravity plant installation costs (less than 3% of total) are possible[7,8,9].

Gravity cannot replace flotation and cyanidation but it can reduce their circuit size, reagent usage and the resulting environmental impact. In North America, gravity circuits based upon the use of jigs/tables or Knelson/tables are frequently used for hard-rock operation. Spirals and Reichert cones are also used, but to a lesser extent. The Knelson Concentrator (KC) has now established itself as the better choice over the jig owing to two factors contributing to its success: its mechanical and operational simplicity and reliability, and its ability to achieve excellent gold recoveries over a wide size range (Figure 1.1). Some plants which had jigs before are now using Knelsons[7,8,9,10,11]. Some of these will now be discussed.

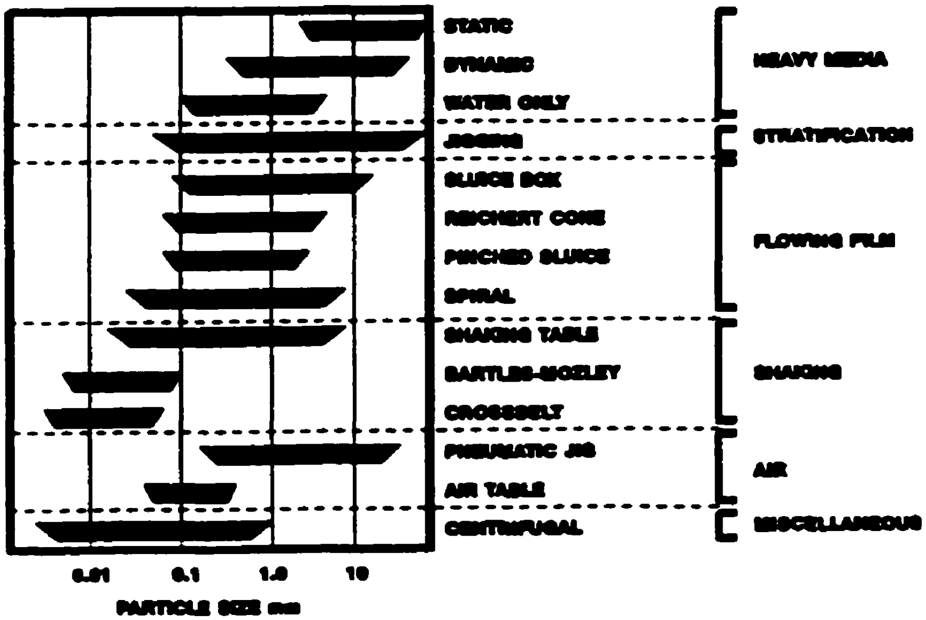


Figure 1.1 Operating range of gravity concentrating units

1.2 Gold Operations that Replaced Jig(s) by Knelson Concentrator(s)

<u>Hemlo Gold Mine</u>

At the Hemlo Gold's Golden Giant mill, the primary mill discharge was pumped directly to one Yuba-Richards jig; due to mechanical problems, its operation was discontinued with no apparent loss of recovery. A 76 cm PKC eventually replaced the jig and its performance will be evaluated in this report[12].

<u>Les Mines Casa Berardi</u>

At Casa Berardi, gold is recovered by cyanidation in a CIL circuit. Much like at Hemlo, the jig present at the plant start-up was stopped and then removed from the circuit because of operational problems. A 76 cm PKC processing a bleed from the primary cyclone underflow (PCU) was eventually installed to replace it. Overall gravity recovery was found to be higher than 30%[11].

Yvan Vézina/Chimo

At the Yvan Vézina mill, a jig installed at plant start-up failed to perform adequately due to tramp iron originating largely from the SAG mill. It was later replaced by a 76 cm PKC processing the flash flotation cell concentrate. The mill was subsequently moved to the Chimo minesite, where the existing gravity circuit includes a 76 cm PKC processing a bleed from the PCU and a 51 cm PKC processing the full flash flotation concentrate[11]. The Chimo plant shut down in early 1997.

1.3 Objectives

The overall purpose of this report is to compare the performance of jigs to that of Knelsons. The problem is not trivial, as the apparent superiority of the Knelson is tempered by its inability to process the full circulating load of the grinding circuit (unlike the jig). There might be conditions such that processing the full circulating load overwhelms the benefit of the Knelson's better recovery in the fine sizes. This warrants close examination. The methodology used consists of the following steps:

- to generate a data base on jig and Knelson concentrator performances by using existing data (Aurbel) or sampling industrial circuits such as:
  - (1) various PKCs circuits (Hemlo, Meston, Est Malartic),
  - (2) the classification circuit of Agnico-Eagle, and
  - (3) the grinding and gravity circuits of Snip Operation
- to use the data in an algorithm to model a gravity circuit that uses a jig (Snip) and to simulate the replacement of the jig with a PKC.
- to explore the relationship between gold recovery and GRG size distribution and the fraction of circulating load treated.

It is expected that the study will yield a better understanding of how the recovery units perform and should be used. The industrial participants to the study should benefit in that these results might indicate how their circuits could be improved.

1.4 Structure of Thesis

Chapter two provides the background on what gravity recoverable gold (GRG) is and how its content is determined. Plant and laboratory units used to recover GRG will be presented.

Chapter three describes the sampling campaigns of four Knelson circuits in order to gather GRG performance data and to illustrate how GRG measurements can be optimised with high density samples with a case study, the primary classification circuit at Agnico-Eagle, division LaRonde.

Chapter four presents the sampling campaigns at Snip operation. After a description of the grinding and gravity circuits, the sampling scheme is explained. Sampling data are then used to estimate unit's performance and gold's behaviour in the circuit.

In chapter five, an algorithm that predicts how much GRG can be recovered in a grinding circuit that includes gravity will be described, using Snip as a case study.

Conclusions, recommendations and future work will be presented in chapter six.

Chapter Two

Background

2.1 GRG

The term "gravity recoverable gold" (GRG) should not be confused with the term "free-milling" gold. Free milling gold refers to the gold that can be readily extracted (95%) by cyanide, typically when the ore is ground to a size of 80% <75 µm. GRG refers to the portion of gold in an ore sample that can report to a gravity concentrate at a very low yield (<1%) and very high grade (typically more than 10000 ppm). This includes gold that is not totally liberated and is part of a particle that is of such density that it reports to the low yield concentrate, but it excludes fine, completely liberated gold that does not have the proper characteristics (shape factor or size) to do so. The amount of gold that can be recovered by cyanidation is generally much higher than the GRG content[3,14].

In this report, GRG is measured using a 7.5 cm laboratory Knelson Concentrator (LKC). It has been shown that the LKC can recover, at a very low yield of 0.2 to 0.5%, 95% of the gold recoverable by amalgamation. The yield is so low that it is assumed that at least 95% of the gold recovered is GRG[3,14].

However, the LKC, as any other gravity unit, can fail to recover GRG when the gangue becomes very coarse and/or very dense. In a grinding circuit, a coarse feed (F_{80} > 400 µm) is generally a SAG or a rod mill discharge, and to a lesser extent, a ball mill discharge or a cyclone underflow (generally finer due to the circulating load). For such feeds, the usual solution is to remove the oversize, +850 µm for low density gangue and +300 µm for high sulphide gangue, prior to processing with the LKC. Dense feeds usually come from massive sulphide deposits but can also be the results of processing, e.g. flash flotation concentrate, and table tails. For those feeds, an alternative to the removal of the +300 µm fraction is dilution with silica flour to achieve the desired density for maximum GRG recovery[3,14,15,16].

2.2 Sample Size

Gold gravity concentration circuits have historically been difficult to evaluate for a number of reasons. Slurry sampling is an essential tool for the evaluation of plant performance but it is error prone, especially when GRG is present, as it is less likely to be uniformly dispersed in the flowing slurry. When sampling, great care must be taken to obtain a truly representative sample. Precision and accuracy are difficult to achieve due to the occasional occurrence of coarse gold, called the nugget effect. Large samples (10-20 kg and sometimes more) are required to make the assessment of gold content statistically sound[3,14].

The occurrence of GRG can be thought to follow a Poisson distribution. Consider a sample that contains on average n flakes. Actual samples will indeed average n gold flakes, with a standard deviation of $\sqrt{n}$. This describes the fundamental sampling error and does not include assaying and screening errors nor systematic errors stemming from inappropriate sampling methodology. For the same grade and mass, finer feeds yield an increasing number of gold particles and thus a lower fundamental sampling error. Thus, coarser size classes normally dictate what the minimum acceptable sampling mass should be. It has been proposed that the maximum size class for which reliable GRG content information could be thus generated be below 850 μm[3,14].

Figure 2.1 offers useful guidelines for sample mass selection and realistic sample accuracy expectations. As an example, if the GRG is below 300 μm (0.5 mg gold particles) and the grade is above 3 g/t, a sample size of 5-20 kg would be representative. This sample size would also yield good size-by-size information (relative error <10%) when grades are 20 g/t or higher[3,14].

2.3 Sample Processing Method

The classical method to process those larger samples is to precede each reduction in weight by a reduction in top size. Every step of reduction of the fragment size and every division of a sample into subsamples introduces additional sampling errors. The variance of the complete process is the sum of the variance of each individual step. This approach, however, cannot be directly applied when studying gold gravity circuits, as size-by-size information, critical to a good understanding of gravity concentration, is lost during the comminution steps[3,14,15,16].

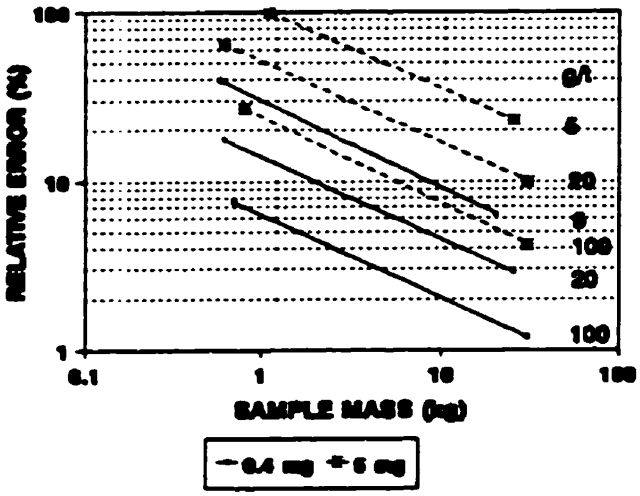


Figure 2.1 Relative sampling error of gold content as a function of gold grade and particle size[3]

If all coarse gold particles could be concentrated in a small mass, assayed separately, then recombined mathematically with the grade of the material from which the coarse particles were removed, the error associated with the overall grade of the sample would be significantly lower. Size-by-size analysis of the gold thus recovered would preserve important size-related information. The LKC has been found to be a particularly effective tool to concentrate liberated gold particles (i.e. GRG) into a small, assayable mass. It can process up to 100 kg of material, which is more than adequate for most ores or plant stream samples to minimise the nugget effect, and concentrates the free gold in a small mass (typically 85-110 g) which can be entirely assayed. This will be discussed in detail in Chapters 3 and 4. Large sample masses can then be completely assayed for GRG. The tails contain virtually no GRG and can be sampled and assayed with less error than the feed[14,16].

2.4 Laboratory Techniques and Devices Used to Measure GRG

The method to recover and measure GRG has traditionally been amalgamation. More recently gravity devices such as flowing film concentrators (the Mozley Laboratory Separator. MLS; superpanners). shaking tables. laboratory jig and even flotation have been used[1,3,8,9,14,15,17].

2.4.1 Amalgamation

In mineral processing. amalgamation is the process of separating gold and silver from their associated minerals by binding them into a mixture with mercury. The wetting of gold into mercury is not alloying but a phenomenon of moderate deep sorption involving a limited degree of interpenetration of solid gold and liquid mercury. In all wetting phenomena. the surface tensions of the substances involved influence the nature of the reaction: gold is readily wetted by mercury because of the higher surface of tension of mercury. Due to the specific gravity of gold (19.3) compared to mercury (13.5) gravitational forces act to immerse the gold in the mercury and may be the most important forces at work. Two important conditions for efficient amalgamation are that the surface of both gold and mercury must be clean and the mercury must offer an adequate receiving surface to the particles of gold. Although the amalgamation process is relatively simple. unsatisfactory results may be obtained by:

- lack of suitable contact between gold and mercury
- gold grains too fine or flat gold grains. which cannot penetrate the mercury
- gold present as telluride or locked in sulphide
- gold grains that have tarnished or contaminated surfaces with oil. grease. talc or sulphur
- impure or floured mercury which cannot open its surface to gold.

Due to health and workplace concerns. and lack of facilities to perform mercury amalgamation. its use is declining. Current practice is limited and its approach in many laboratories is slightly different. Once a sample has been amalgamated, the tailings and feed samples are assayed and the free gold content is determined by difference. Thus no mercury distillation (by far the most hazardous step) takes place. Unfortunately, this approach does not eliminate the nugget effect (when assaying the feed)[3,14,18].

When determining the behaviour of gravity recovery in a circuit, evaluation with amalgamation can overpredict GRG content: very thin flakes of gold that would be refractory to gravity recovery will be readily amalgamated. The method can also underpredict GRG content as some coarser gold can resist amalgamation if its surface is coated with a contaminant or is imperfectly liberated. In practice, amalgamation overestimates GRG in the fine sizes, and underestimates it in the coarse (i.e. Knelson tails have higher gold content than amalgamation tails in the fines, and lower in the coarse). Another disadvantage is that some gold particles are either completely or partially dissolved in mercury; then they tend to coalesce during the mercury-gold separation; thus, their original size distribution is lost[3,14,18].

2.4.2 Flotation

Graham[3,19] investigated gold recovery by sampling the gravity and flotation gold circuit of Echo Bay Minerals' Manhattan. Batch flotation in a laboratory Denver cell (0.033 m^3) was used to process a -600 μm (-28 mesh) Wilfley table concentrate sample (a pyrite gold concentrate). Soda ash and sodium cyanide were used to depress pyrite. The gold was also depressed initially but after 30 minutes of additional conditioning time it began to reactivate and float. The gold floated in stages where the finer gold particles floated first followed by the coarser gold particles and finally the very coarse particles (+210, +297 and +420 μm) began to float. A recovery of 96% was achieved in a yield of 1.8%. The method appears successful although it was reported to be very operator sensitive. Also after two and half-hour of flotation, the gold size distribution appeared reduced, presumably because of partial cyanidation. Graham concluded that accurate metallurgical accountability was unattainable, even in carefully controlled batch flotation tests, when coarse gold is present.

2.4.3 Laboratory Shaking Tables

Tables and panners sort material by using a combination of flowing film concentration and other mechanisms (i.e. inertia, jigging). Coarse light particles are separated from small dense particles when they are introduced into a film of water flowing down an inclined surface[3].

The MLS is fast, practical and capable of treating a large number of small sample (100 g) of ore. Its primary role is for flowsheet design. The unit consists essentially of a separating tray sloping slightly in one direction, oscillating in a simple harmonic motion in another direction and capable of recovering particles below 100 μm better than jigs, sluices, cones and spirals[2,3,15]. Liu[1,20] tested its performance as a standard unit to assess plant gravity performance and its ability to separate gold from sulphide by estimating GRG content in processed streams from Les Mines Camchib. The procedure consisted of wet and dry screening two to three kilos of material and processing 75 to 150 g of each size class with the MLS, recovering four different products to generate a grade yield curve. The approach was time consuming and required a large number of assays to determine yield curves. Also, results were found not entirely reproducible even if the MLS gave an accurate indication of GRG content.

Banisi[15,17] also assessed the efficiency of the MLS. The same procedure was used on the primary and secondary cyclone overflows (PCO and SCO respectively) of the Golden Giant Hemlo gold mill with the MLS and a LKC for comparison. The MLS actually outperformed the LKC on the SCO by about 5% at equivalent yield, because overflows are the most refractory streams to gravity recovery due to incomplete liberation, gold particle shape and very fine size. However, the LKC bettered the MLS with the PCO. Putz[3] also processed samples (jig concentrate and table tails) from the Dome mill with a MLS when sample mass was insufficient to process with a LKC. All samples were screened at 600 μm to 38 μm. Each size class was processed separately on the MLS to produce a concentrate of less than one assay ton and a tailing sample. Results were noisy due to the small masses processed (<300 g) and operator dependent as the

MLS slope had to be adjusted for each sample to optimise separation. Another problem was the capacity of the unit. only 150 g could be processed which for coarse classes is clearly insufficient for good statistical reproducibility. It appears that the MLS is most effective when the mass is too small for the LKC. the grade high enough in order to get good statistics and the particles coarse enough to be recovered by the MLS[1,3,16,17,21].

Hand panning is one of the oldest methods of carrying out physical testwork on samples of up to a few kilograms. However. it is not reproducible and is clearly unsuitable as a measure of separation[1]. Agar[21] reported use of a superpanner as an ideal separator (complete separation of the valuable material from gangue). Two stages of superpanning separation were used with very low weight recoveries (50-60 mg) in the individual size fractions of the final concentrate. High concentrate grades (19% and 38% Au) were achieved. Since a superpanner is also a flowing film concentrator. similar problems to the MLS could be experienced. Superpanners are more difficult to operate than the MLS and process even smaller masses.

Shaking tables are generally incapable of efficient separation below approximately 75 µm. An exception may be the Gemeni table. as it was designed especially for gold recovery. It is also capable of processing large quantities (1-5 kg) of material. The mechanism of separation of the device is explained later. Liu[1] used a Gemini table to estimate GRG content and determine its performance as a measure of GRG. The procedure consisted of processing about 450 g of unprepared feed to produce four products (tail. two middlings and a rougher concentrate) and to process the concentrate again to yield four more final products. Results were disappointing: the table failed to yield a concentrate of very high grade or a tail of very low grade. However. the unit recovered 78% of gold in a 12% yield from a riffleless table middlings without elaborate effort to optimise operation. which suggests it has a good potential as a production unit.

2.4.4 Laboratory Mineral Jig

A laboratory jig is mostly used when information on the performance of a plant jig is needed. The mechanisms of separation will be explained later on. Putz[3] processed the oversize product of very coarse samples (rod mill discharge, primary and regrind cyclone underflow, jig tails and table tails from the Dome mill) in a Denver laboratory mineral jig. Jig bedding consisted of +4 mm steel shot ranging in mass from 100 to 300 g and all +4 mm material was removed from samples prior to jigging. Samples were processed to obtain a concentrate of less than one assay ton (29.166 g). Resultant tailings and concentrate were screened into five size fractions (2.4 mm, 2.0 mm, 1.2 mm, 840 μm and 600 μm) to eliminate the nugget effect and assayed. The jig proved to be cumbersome, requiring for each sample constant attention and changes in the amount of ragging, stroke length and hutch water.

2.4.5 Knelson Concentrator

The laboratory Knelson concentrator (LKC) consists essentially of a riffled cone rotated at high speed, with a drive unit (Figure 2.2). It uses the principles of hindered settling with interstitial trickling enhanced by a centrifugal force of 60 times that of gravity (60 Gs) (generated at 1700 rpm for the 7.6 cm LKC). Feed (20-40% solids) is introduced by gravity through a vertical tube to the base of the rotating bowl where it rapidly gains rotational speed and progresses upward and over the rim of the bowl to the tailing launder[6,10,19]. Due to the tapered cross-section of the cone, part of the rotation kinetic energy is translated into a flowing velocity. As the slurry flows over the concentrator riffles, denser particles can trickle into their active zones where they are recovered once feeding is stopped and the cone retrieved from the concentrator. Lighter particles are carried by water to the top of the unit while a constant volume bed is formed between the cone riffles. Due to the high centrifugal force, surface chemistry effects such as surface tension on the air-water interface are negligible[3,10].

Clean water is injected through holes in the inner bowl of the concentrator to prevent compaction of the concentrate bed. Water is injected tangentially, counter-current to the rotation of the bowl (Figure 2.3). Water addition is the key to the

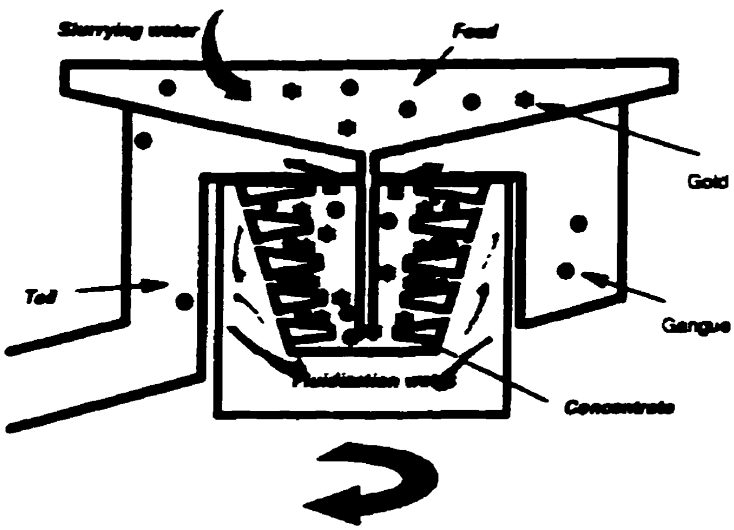


Figure 2.2 Schematic diagram of the Laboratory Knelson Concentrator[1]

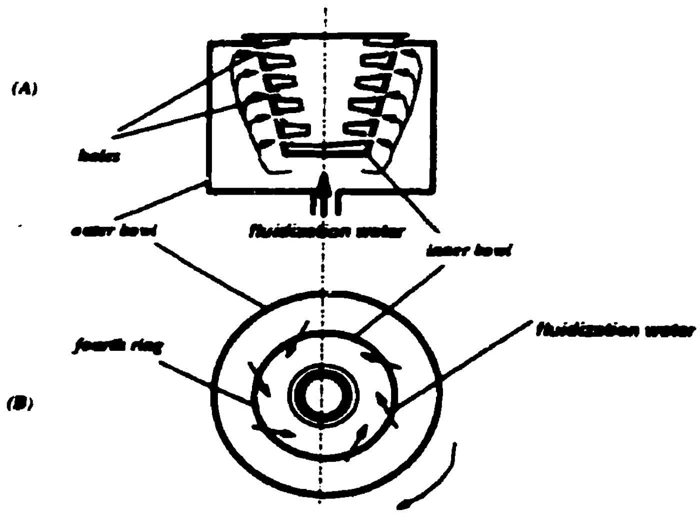


Figure 2.3 The cross-section of the bowl and the supply of high pressure fluidizing water from a top view of the last ring[1]

performance of the LKC: fluidization allows fine gold to penetrate the bed under high 'g' forces since it essentially controls the concentrate bed bulk specific gravity and porosity. Excessive fluidizing water can flush gold out of the concentrate bed, while insufficient water will fail to fluidize the concentrate bed adequately. Higher fluidizing water flowrates are required as gangue becomes coarser or denser[3,6,10,11,22].

The addition of water prevents the material from attaining the same speed as the cone, thereby producing a shear rate that dilates the flowing slurry (Bagnold effect) and favours the recovery of fine dense particles. This rotational shear is very similar to that used by Bagnold to demonstrate the existence of dispersion induced by shear. The force generated in the LKC bed is, according to the formula

$$F_c = 4\pi^2 m n^2 r \tag{2.1}$$

where F_c: centrifugal force

m: particle mass (g)

n: rotational speed (rpm)

r: bowl radius (m)

It has been claimed that a more effective separation is attained at 60 Gs than at gravitational acceleration because of the increase in the specific gravity difference between gold and gangue. More specifically, the increased terminal velocity enhances the percolation trickling, a mechanism critical for fines recovery[3,6,10,11,22].

The efficiency of the unit is affected primarily by the feed rate and is sensitive to the size distribution and density of the gangue. Feed that is very dense can be diluted with silica to achieve the desired density for maximum GRG recovery. Huang[23] has shown that gold recovery increases with decreasing size when diluting a high grade sulphide sample (F_{80}<400 μm) down to a density below 3.2 g/cm^3. It was shown that gold recovery would significantly decrease at a feed size (F_{80}) above 1 mm for a silica gangue or a gangue density above 3.2 g/cm^3. For massive sulphides (4.5 to 6.0 g/cm^3), a dilution of 4:1 (silica to feed material) is adequate to bring the density down to 3.2 g/cm^3. Less dilution may be acceptable for material with different blends of heavies and lights.

Putz[3] also diluted plant Knelson feed and tails in a 2:1 and 4:1 dilutions (70 mesh and 25 mesh). Results show that dilution produced higher gold recoveries, especially in the fine size range below 100 µm: improvements increased with decreasing particle size. It seems that the most significant impact of dilution is density reduction rather than size reduction[3,6,11,24].

The LKC can readily measure the amount of GRG in a stream and be used as a "perfect separator" to study another gravity unit like a jig, a sluice or even a full scale PKC, provided that its performance is superior to that of the plant units. Whereas it is impractical to dilute feed in plant practice, size preparation (i.e. removal of oversize) can improve PKC performance. Recycled streams (ball mill discharge, cyclone underflow) were found to be better candidates than cyclone overflow where the gold is too fine and flaky to be recovered[3,7,11,15,20,24].

A methodology to determine the amount of GRG in an ore was developed using the LKC. The procedure is based on sequential comminution (the first at 100% -850 µm, the second at 50% -74 µm and the third at final grind typically 75 to 90% -74 µm) and recovery steps with a LKC. The mass processed depends on the gold grade and particle size, and commonly varies from 25 to 100 kg. Woodcock[14] applied the technique on ores of existing mills as well as developmental orebodies. Conclusions were that the knowledge of the size distribution of the GRG in the ore could focus on the design of a gravity circuit or eliminate it as an option, prior to any extensive pilot plant testing. The information can also be used to evaluate circuit performance. This procedure yields an essential component for an algorithm that will be used to predict the amount of GRG that can be recovered by installing a gravity circuit. This algorithm will be described later on and illustrated with one case study.

2.5 Plant Units for Gold Recovery

2.5.1 Jigs

Although jigging is one of the oldest methods of gravity concentration, its principles are still not completely understood. It can concentrate a fairly wide range of material, from 200 to 0.1 mm in size. Jigs are used in many applications, especially for treating coal, alluvial deposits and coarse free gold in North and South American grinding circuits[2,3].

A jig is essentially an open tank filled with pulsated water, with a horizontal screen at the top and provided with a hutch compartment for concentrate removal (Figure 2.4). Jig cycles are made up of a pulsion and a suction strokes producing a harmonic motion. Minerals of different specific gravity are separated in a fluidized bed by a pulsating current of water creating stratification while hutch water is added to reduce the rate of the downstroke and aid the stratification. In a controlled manner, the pulsation stroke allows the mineral bed to be lifted as a mass and then dilated as the velocity decreases: the heavier, smaller particles penetrate the interstices of the bed and the larger high specific gravity particles fall, and stratification occurs while the suction stroke slowly closes the bed. Stratification is also affected by the length, frequency and cycle pattern of the jig stroke. The jig also separates the stratified layers into two discrete products[2,26].

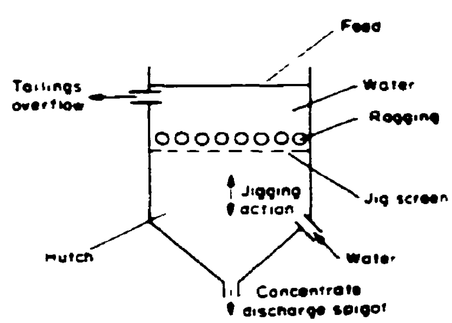


Figure 2.4 Basic jig construction[25]

The first common approach to describe particle motion takes into account the various phases of the jig cycle and the dominant settling mechanisms. The mechanisms are as follows: differential acceleration at the beginning of the fall, hindered settling, and interstitial trickling (Figure 2.5). The particle bed dilates and moves upwards until the velocity is reduced to zero during the upward stroke of the jig cycle. At that instant, particles can be considered as starting to fall from rest with initial accelerations, and hence velocities, which are a function of particle densities and independent of particle size. If the repetition of fall is frequent enough and the duration short enough the distance travelled by dissimilar particles will depend upon their initial accelerations rather than their terminal velocities, resulting in stratification on the basis of specific gravity. Most of the stratification occurs during the period when the bed is open and results from differential trickling accentuated by differential acceleration. Consolidation trickling occurs when the bed is compacted and places the fine/dense material on the bottom and coarse/light material on the top. Since the two effects arrange the particles in diametrically opposite ways, suitable adjustment of the cycle should supposedly balance the effects and result in an almost perfect stratification according to mineral density. Hindered settling in a jig can only take place if the pulp has a high density[2,26].

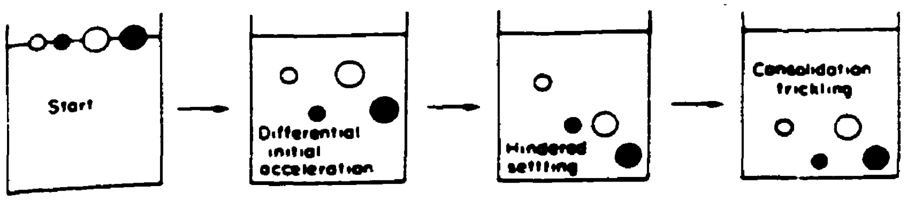


Figure 2.5 Idealised jigging process[25]

Another approach to the analysis of jigging is called the centre of gravity theory, or the attainment of minimum potential energy levels, in which water pulsation is purely use to open the bed and allow the release of its potential energy while denser particles are able to move down through it[2,26].

Different portions of the jig cycle are considered important: Bird[2,26] believed that separation takes place on the suction stroke; Mayer[2,26] believed separation occurs during the downstroke as the particles are resettling though the fluid. It has been suggested that for producing higher grade higher frequencies and lower amplitude are preferable since small rapid movements provide best absolute separation; conversely higher amplitudes and lower frequencies give a more open bed and allow more rapid particles movement and thus enhance recovery. A relatively deep layer of light minerals also enhances recovery of dense minerals while thickening of the dense mineral layer aids the grade.

Burt[2] believed the length and frequency of the stroke are inter-related. Closed-sized coarse feeds, with a high proportion of heavies, require large amplitude and a long cycle time. Fine feeds, with a wide size range and low heavy mineral content, need small amplitude and a short cycle time. For clean concentrate production, a compact bed is required and achieved with a short rapid stroke, while high recovery is obtained with a mobile bed achieved by long slow strokes.

Jig capacity varies depending on the jig configuration (rectangular or circular), ore feed size, and adjustments of stroke length and speed. Generally, capacity is described as the optimum throughput that produces an acceptable recovery and is determined by the area of the screen bed. Coarser grains can usually be fed in larger volumes than fine grains in relation to the area of the jig bed. Higher-density minerals can be fed in larger volumes also. Flat-grained particles tend to slow the concentration rate, an important consideration for gold, which flattens in the grinding process. Jig feed rates need to be constant because too much feed will dampen the jigging process while under-feeding will waste energy and diminish its efficiency. It is also important to have a constant pulp density of the feed, typically 30-50% solids. Hutch water addition is another important factor in jigging. Jigs treating coarse material require more hutch water than those treating finer material do. Ragging is used in jigs to allow finer particle sizes to be treated, as it prevents the light, fine particles from penetrating completely through the ragging interstitial during the downstroke; they are then rejected from the ragging by the upstroke[2,26].

2.5.2 Knelson Concentrators

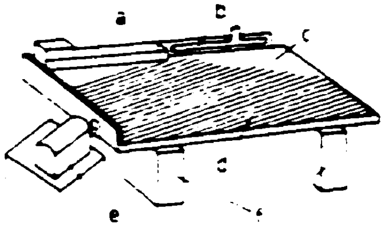
Created twelve years ago by Knelson Gold Concentrators, this device can now be found in over 60 countries and accounts for more than 800 separate installations. The main advantage of the unit is its ability to recover GRG over a wide size range, typically 25 to 850 μm, with recovery falling around 25 to 37 μm due to mechanical limitations in recovering fines. There are six standards Knelson concentrators models (3, 7.5, 12, 20, 30 and 48 inches), the latter three being plant units and the 30" PKC rated for 40 t/h. The

PKC can be fed a preconcentrate to increase overall efficiency: a bleed of the circulating load (ball mill discharge, cyclone underflow) or the undersize of the screened circulating load, or the concentrate of another gravity unit such as a sluice or a Reichert cone. The concentration mechanisms of a plant unit are similar to those of the laboratory unit previously detailed[10].

Recent additions are the centre discharge (CD) model manufactured only for the plant units. The CD can be totally automated and integrated into any existing computerised circuit. Removal of concentrate is accomplished automatically in less than two minutes, with feed diversion, reduction of fluidizing water pressure and bowl rotation speed. As rotation speed falls, the concentrate is flushed from the rings past the feed deflector and piped directly to a secure gold room. Operation is resumed and feed is directed back to the concentrator[10]. At Lac Minerals, a 30" CD PKC replaced a jig, recovered 40% of the gold from the head, and gave a high upgrading ratio (1000:1) with one single stage compared to a jig and table combination (200-300:1) which required more frequent final clean-up[13]. Chapter three consists of the characterisation of the performance of the 76 cm (batch and CD) PKC treating different ores: high grade, low grade, high sulphide, and low sulphide.

2.5.3 Tables

The shaking table remains one of the workhorses of the mineral processing industry (Figure 2.6). It is used mostly for secondary upgrading, typically giving concentrates assaying 40% to 80% Au. Performance ranges from recoveries in the low 80% with conventional tables to the mid 90% with a Gemeni. Simple deck tables have relatively low capacity for their cost and space requirements[2,26]. A shaking table consists of a slightly inclined deck on to which the feed is introduced at the feed box and distributed along part of the upper edges and spread over the riffled surface (rubber, fiberglass) as a result of a longitudinal vibration and wash water. This action not only opens the bed to allow dense particles to sink but also by its asymmetry provides particle transport along the table (Figure 2.7). The product discharge occurs along the opposite side and end[2,26].



a) Feedbox. b) Wash water. c) Deck. d) Riffles. e) Shaking mechanism. f) Supports

Figure 2.6 Shaking table[25]

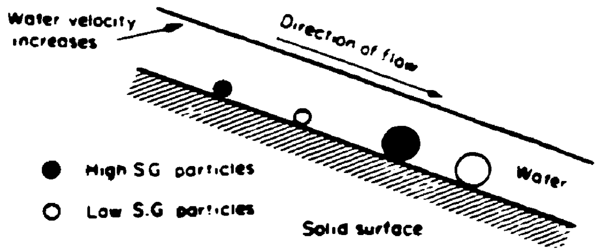


Figure 2.7 Idealised tabling process[25]

The Wilfley table was the first to use this differential shaking motion then followed the Deister. Many gold operators now favour the Gemeni table, of recent design, because of its metallurgical performance and ease of operation. It was designed especially for gold recovery, with longitudinal and traverse slopes on a central convex grooved surface. As the table shakes, gold particles settle in the grooves and slip down to a concentrate launder. Gangue material crosses over the heavier mineral bed and is washed down to a tail launder. Its distinct advantage over the other tables is its capability to produce a clean GRG product. However, its capacity is only about 300 kg/h when feed grade is high, which can make feed rate control difficult[1,2,26].

2.6 Gold Behaviour in Grinding Circuits and Plant Knelsons

The LKC had been used to study the behaviour of GRG in grinding circuits. A first study took place at Hemlo[15,17] whose grinding circuit consisted at that time of three ball mills in series, the last two in reverse closed circuit with two cyclopaks. LKC results were used to make a circuit mass balance which showed a high gold circulating load of

6700%: classification of gold took place at a much finer cut size than the ore. 20 μm versus 57 μm and since 80% of the primary cyclone underflow was coarser than 53 μm. 99% of the GRG was recirculated. Conclusions were that this stream was a good candidate for gravity recovery. possibly with a Knelson concentrator. The high circulating load suggested that it would not be necessary to process the full stream. A PKC was subsequently installed in the grinding circuit[1]. and its performance will be evaluated and discussed in this report.

The gold gravity circuit at Les Mines Camchib (now Meston Resources) was the first hard-rock application of the Knelson[1,20]. A first detailed study was performed using the MLS. as the LKC had not yet been commercialised. The gravity circuit then consisted of two sluices feeding two 76 cm PKC. for coarse gold removal. which themselves fed a 19 cm KC used as cleaner. Classification of gold was found to take place below 38 μm. The two Knelsons achieved 58 to 71% total gold recovery. and 82 to 93% GRG recovery. The 19 cm KC achieved about 90% recovery with two passes and was later replaced with a 30 cm unit which yielded better recovery in a single pass. The efficiency of the circuit at that time was limited by the pinched sluices. which performed poorly. recovering only 8 to 17%. Total gold recovery in the gravity circuit was found to be about 30%. Recommendations were that an increase (50%) in the feed rate to the Knelsons would result in a substantial increase in total gold recovery even if the free gold stage recovery were to drop below 80%. because of the increased throughput.

A second study[20] (the first where the LKC was used to investigate a gravity circuit) showed that an increase in the feed rate to the Knelson had indeed decreased Knelson stage recovery of GRG to 62% but with an overall increase in total gold recovery (about 35-40%. based on monthly metallurgical accounting). As in the first study. GRG recovery was found not to decrease during the Knelson recovery cycle. Testwork at various PKC feedrates (76 cm units) will be evaluated and discussed in this report.

Buonvino[22] also used the LKC to measure GRG content in the grinding circuit of Agnico-Eagle, where a gold-chalcopyrite ore with 50% sulphide (mostly as pyrite) was treated through sluices in the grinding circuit before flotation. A mass balance of the circuit was performed with the LKC results. Problems with sampling of the primary cyclone samples (feed, O/F and U/F) could not yield a reliable partition curve for GRG. Because this information is critical in assessing the circulating load of GRG, testwork presented in Chapter 3 will aim at establishing a partition curve for total gold and GRG. Buonvino found that an abundance of free gold was needlessly locked up in the regrinding loop and was creating a gold circulating load of 3720%: 75% of gold in the secondary cyclone underflow (SCUF) reported to the +53μm (270 mesh) and only very fine gold (-25 μm) was successfully removed from the circuit.

Putz[3] used a 7.5 cm LKC to evaluate the performance of two gold gravity circuits, Lucien Béliveau and Dome Mines. The initial circuit at Lucien Béliveau consisted of a flash flotation cell whose concentrate was fed to a 76 cm PKC. Gold recovery in the 76 cm PKC averaged 45%. Gold was recovered consistently in all size fractions greater than 38 μm while recovery dropped to 22% below 38 μm. It was also discovered that much of the gold in the ball mill recirculating load was too coarse for significant recovery by the flash flotation cell but could be recovered by gravity from the cyclone underflow or the ball mill discharge itself. Later on, the flash flotation concentrate was directed to a hydroseparator whose underflow fed a spiral. The spiral tailings and the hydroseparator overflow reported to a 51 cm (20") PKC. The change in the circuit configuration reduced total gold recovery to 32%. This reduction was attributed to a decrease in the quantity and average size of GRG being fed to the PKC, and the smaller size of the PKC.

At the Dome mill, the primary gravity circuit consisted of four duplex jigs. Testwork was done to view the potential improvements[3]. A high gold circulating load, 1800%, was found and suggested that the jigs did not recover gold adequately despite the high GRG content. All four jigs produced varying grades and size distributions of GRG. GRG content in the jig concentrates varied between 61 and 93%, indicating that some jigs tended to recover unliberated gold associated with pyrite. Very little fine gold was found in the jig concentrates, indicating that its recovery was extremely low. Unfortunately, because the eight concentrates were extremely different in weight and gold content, and could not be weighed, an exact size-by-size recovery could not be estimated. A PKC has since replaced the jigs and recovers more gold, even when treating only 13% of the circulating load.

The Dome work suggested that even when GRG is very coarse, jigs could not outperform a Knelson Concentrator. In this work, jig performance will be further probed at the Snip Operation mill, and the performance of a 20" PKC simulated as a replacement for the jig.

Chapter Three

Test Work on Various Ores

One of the main objectives of this project was to generate a data base on plant Knelson Concentrator (PKC) performance focussing on three important variables: gangue density. size distribution and feed rate. The evaluation of plant units at various locations was done using the LKC to process samples extracted from the gravity circuit. The description of the circuit at each mill. the sampling and sample processing program. and an evaluation of the results will now be given.

3.1 Meston Lake Resources

3.1.1 Introduction and description of the mill

The Meston Lake Resources mill is located near Chibougamau. Québec. Previously called Camchib Mines, it has been processing copper-gold ores since the 1950s. When gold prices increased while copper prices declined, gold became the dominant economical mineral in the Chibougamau ores. as much as 90% of the total value. A gravity circuit was added to the mill in 1984 to recover as much of the coarse gold as possible ahead of copper flotation. Then. the copper grade in the flotation concentrate was decreased from 23-26% to 17-19%. to achieve a gold recovery increase from 75-82% to 87-90%[120].

Nowadays. the mill treats a low copper grade ore from the Joe Mann mine: 8-10 Au g/t. 0.3%Cu. The ore contains from 3 to 5% sulphides. mostly pyrite. pyrrhotite and chalcopyrite. with traces of sphalerite and galena. The host matrix is made of chlorite. quartz. carbonates (calcite. siderite. ankerite) with minor occurrences of chlorotoid. actinolite and talc. Gold occurs in two generations. The first one (25% of the total gold). very fine (10 μm), is associated with silicates and makes grinding to achieve full gold liberation impracticable; to improve recovery, cyanidation following flotation and gravity concentration must be used. The second generation gold, associated with sulphides and alloyed with silver (20%) as electrum, is coarse, easily liberated and responds well to gravity recovery. In 1991-1992, gold recovery was found to be 88%: 35-40% from gravity, 35-40% from flotation and the rest from cyanidation[1,20].

The gravity and grinding circuit flowsheet of Meston Lake Resources is illustrated in Appendix A, on page 93. After three stages of crushing, ore is ground in a 3.4 x 4.0 m rod mill in open circuit with two 3.1 x 3.7 m ball mills operated in parallel in closed circuit with two 76 cm cyclones. One ball mill discharge is fed to a 6.1 x 1.4 m "double" sluice, and the other to a 6.1 x 0.7 m "single" sluice. Another single sluice is installed on the rod mill discharge. The sluice tails are recycled back to the cyclones, of which the overflow goes to flotation. The sluice concentrates are screened (1.7 mm, 10 mesh) and fed to two 76 cm PKCs operating alternately at a loading cycle of 90 minutes. The Knelson tails are pumped to the cyclones. The Knelson concentrate is pumped to a security area called the gold room where it is screened at 1.7 mm and fed to a 30 cm PKC for a first upgrade. The PKC concentrate is fed to a 2 x 1 m riffleless table yielding a final gold concentrate acid cleaned prior to direct smelting. The table middlings are recycled to the table. The table and the 30 cm PKC tailings are column-cyanided and washed prior to recycling to the grinding circuit[1,20].

3.1.2 Previous Work

Test work by Woodcock[14] determined that the amount of GRG in the ore was 68%, out of which 85% was finer than 200 µm, and 50% finer than 100 µm. In previous test work, Liu[1] evaluated the mill circuit and showed that 40% of the GRG in the ore escaped the gravity circuit via the cyclone overflow. From the gold in their feed, the two 76 cm PKCs recovered between 58 and 71% (between 82 and 93% of the GRG). The highest total gold recovery was achieved with the lowest feed rate (12-15 t/h) and at a wash water pressure of 80 kPa. The lowest recovery occurred at a lower back water pressure (40 kPa) and a feed rate between 18 and 20 t/h. The problem of gold losses was attributed to the pinched sluices which were not judged as efficient (stage recoveries were between 10 and 30%). Different suggestions were proposed to increase the yield and consequently total gold recovery: installing a second sluice to double the single unit and/or doubling the PKCs feed rate from 20 to 40 t/h. Recommendations were to increase the PKCs feed rate by as much as 50% but divide it between the two units run in parallel to increase total gold recovery.

3.1.3 Objectives, Sampling and Test Procedure

The objective of this testwork was to analyse the performance of the two 76 cm PKCs running alternatively at different feed rates (20, 30 and 40 t/h), or running simultaneously at 30 t/h to measure the effect on GRG recovery.

Sampling of these operating conditions was conducted in July 1992 and was labelled T1, T2, T3 and T4, respectively. Samples were processed in the fall of 1992 and the summer of 1993. Samples were prescreened at 850 μm (20 mesh). The undersize was processed in the LKC at a feed rate ranging from 300 to 500 g/min and a water jacket pressure between 21 and 30 kPa (3-5 psi). For each LKC test, timed tailing samples were collected, dried and weighed. Two 300 g samples of LKC tails were wet screened at 25 μm (500 mesh), dry screened from 25 to 600 μm (500 to 28 mesh). Each size fraction of both samples was recombined. All size fractions above 150 μm (100 mesh) weighing more than 20 grams were pulverised. The same dry screening procedure was used for all the LKC concentrates (without the wet screening because of their very low -25 μm content); no pulverisation was done, as each size class was completely assayed. All size fractions were sent to Meston to be fire-assayed. The head grade of the original LKC feed was then back-calculated from concentrate and tails assays. Details of the LKC tests are shown in Appendix B (pages 97-99).

3.1.4 Results and discussion

Table 3.1 shows the percent solids of each sample. The individual PKCs feed samples were taken at the discharge of the sluices. The similarity of the results showed the consistency and the reproducibility of the sampling procedure. Only the tail sample of T4 (the PKCs running together) had a lower percent solid, of about 10%, compared to the others. This would be expected, as twice as much fluidization water was then used.

Table 3.1 Percent solids of the Meston PKC samples

Sample	Wet weight	Dry weight (kg)	%Solids
Feed T1	50.0	27.9	55.0
T2	48.6	27.8	57.3
T3	21.5	12.7	59.1
T4	15.6	8.2	52.5
Tails T1	79.9	36.0	44.6
T2	30.8	13.1	41.9
T3	20.6	17.1	45.7
T4	32.1	11.4	34.8

The overall performance of the two PKCs is shown in Table 3.2. The back-calculated head grades of the PKCs feed samples were similar and average 0.50 oz/st. As the ore itself graded 0.24 oz/st, the upgrading of the circulating load and the sluices was remarkably low: this indicated that the gravity circuit was removing GRG quite efficiently (although part of this was due to the dilution from the concentrate of the rod mill sluice). Grades of the PKCs tails varied between 0.29 to 0.48 oz/st. T2, with a gold content of 0.48 oz/st for the LKC tail sample and a head grade of 0.49 oz/st for its LKC feed sample. This represented either a contamination problem or a mis-identified sample, as it effectively showed no significant PKC recovery (impossible at a feed rate of 20 t/h). As the high GRG content of the T2 tail sample further confirmed its anomalous character, it was excluded from further analysis.

Table 3.2 LKC metallurgical performance of the Meston PKC samples

	T1, 30 t/h		T2, 20 t/h		T3, 40 t/h		T4, 30 t/h in //	
	Feed	Tail	Feed	Tail	Feed	Tail	Feed	Tail
Feed grade, oz/st	0.49	0.29	0.49	0.48	0.49	0.36	0.52	0.30
GRG content, %	55.1	23.6	59.4	47.9	52.4	28.8	53.4	32.2
LKC conc. grade, oz/st	10.1	11.1	34.9	22.6	30.5	12.6	21.9	5.29
non-GRG grade, oz/st	0.23	0.22	0.20	0.25	0.24	0.21	0.24	0.20
Total gold recovery, %	41.6		-		26.5		42.3	
GRG recovery, %	74.6		-		59.6		65.2	

As the LKC recovered virtually no locked gold and 95% of the GRG available, the locked gold content in the LKC tails of the plant feed and tails should be identical. This was effectively the case, as the LKC tail of the feed samples varied between 0.20 and 0.24 oz/st, and that of the tails samples between 0.20 and 0.25 oz/st. To assess the performance of the PKCs. total gold recovery must be first determined. For T1. the overall gold loss. the grade ratio of the PKC feed and tails. was 0.584 or 58%. Thus PKC gold recovery can be determined by

$$R_{GRG} = 100\% * (1 - \frac{G_{tail}}{G_{feed}}) \tag{3.1}$$

It was found to be 41.6%. Gold recovery remained the same when the PKCs were run together at the same feedrate (T4). However. as the feedrate increased. total gold recovery decreased down to 26.5% (T3).

Total gold recovery is a misleading estimate of the PKCs performance since only 52 to 59% of the gold in the PKC feed is gravity recoverable. mostly as a result of the gravity circuit having efficiently brought the gold circulating load down. The Knelson's loss of GRG was the ratio of the GRG gold grades and GRG content can be determined according to

$$R_{GRG} = 100\% * (1 - \frac{GRG_{tail}}{GRG_{feed}}) \tag{3.2}$$

Thus. at the actual plant feedrate of 30 t/h. GRG recovery was a respectable 75%. below that measured by Liu[1] (82-93%) at lower feed rates. GRG recovery dropped as the feed rate increased. T3. at a higher feed rate of 40 t/h. showed an even lower GRG recovery of 60%. T4. which in principle should display a high recovery. as the feed rate to each Knelson was only 15 t/h. showed a GRG recovery of only 65%.

Figure 3.1 and Figure 3.2 show the size-by-size gold distribution and GRG content for the PKC feed and tails. In both samples. 70 to 75% of the total gold was finer than 150 μm. The GRG content increased as particle size decreased. The low GRG content in the coarse size classes (>300 μm) indicated that gold may not be fully liberated at those sizes. or that it may be recovered so efficiently that it does not build up in the circulating load.

Figure 3.3 shows the size-by-size GRG recovery of the PKCs run at the three different conditions. Only the data obtained when the PKCs were run in series at 30 t/h was not noisy. The recoveries of the three tests were averaged but some noise still remained. However, the ability of the Knelson to recover GRG over the full size range

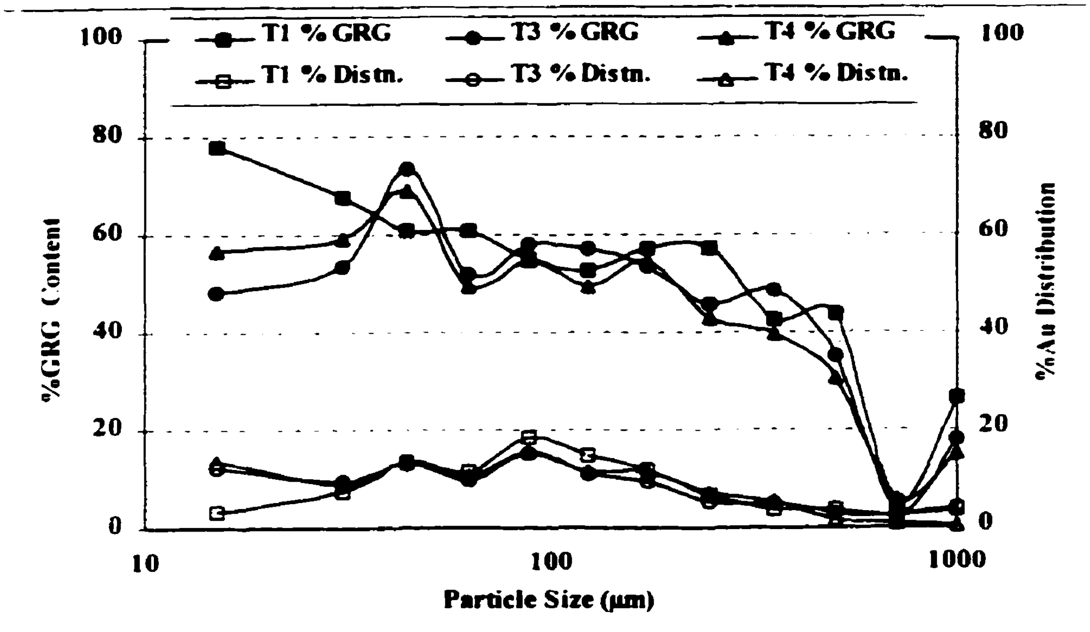


Figure 3.1 Size-by-size total gold distribution and GRG content in the Meston PKC feed samples

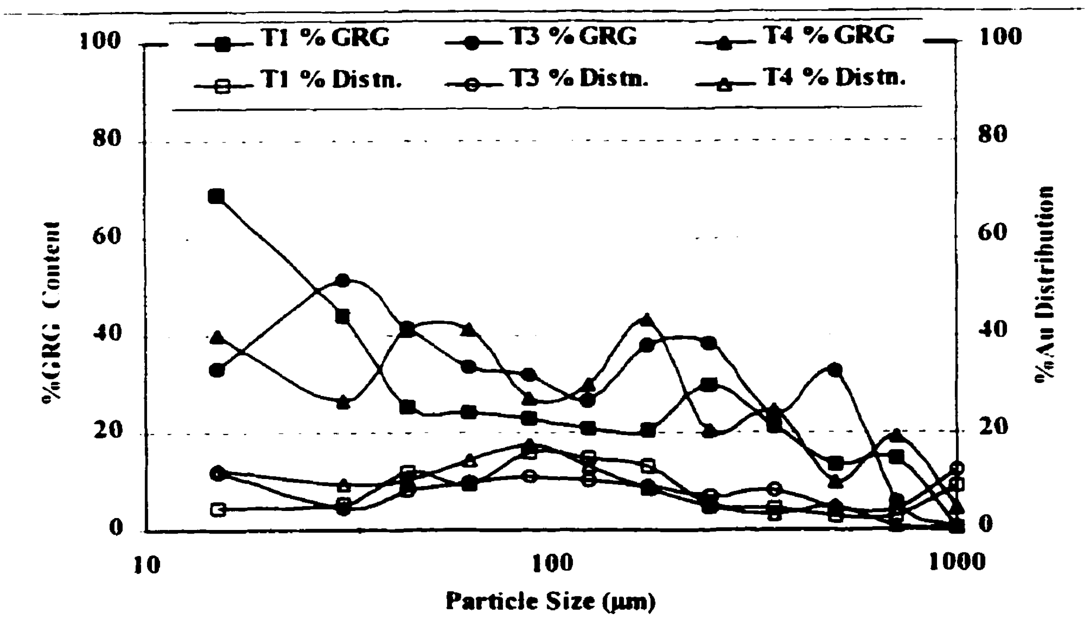


Figure 3.2 Size-by-size total gold distribution and GRG content in the Meston PKC tails samples

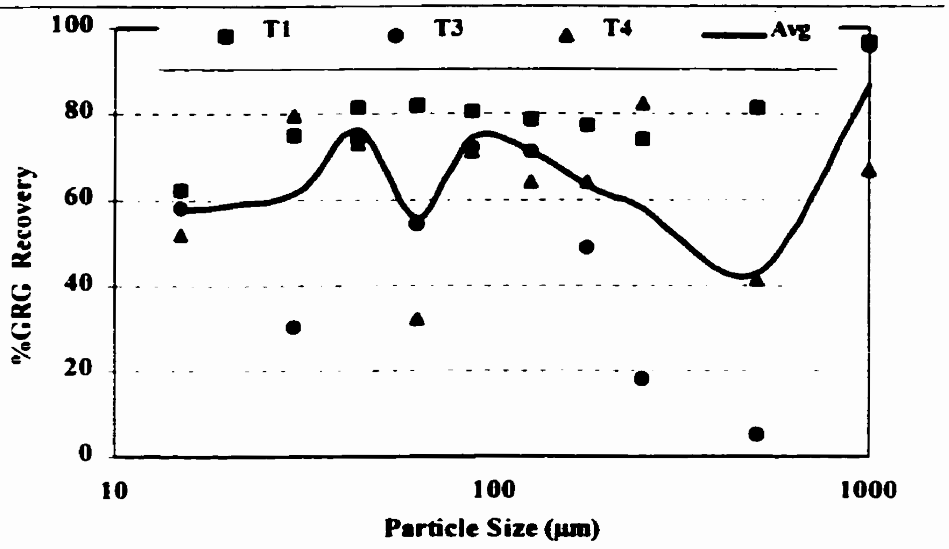


Figure 3.3 Size-by-size PKC GRG recoveries for Tests 1, 3 and 4

clearly comes out. PKC total gold recovery actually increased with decreasing particle size, as GRG recovery remained constant, but GRG content increased (Figure 3.1). Below 37 μm, the unit showed its limitations when recovery slightly dropped.

Tests 1 and 3 rightly suggested that increasing feed rate would result in a decrease of both total gold and GRG recoveries. The results of T4 are more difficult to assess, as the use of the two Knelsons should have increase recovery significantly. Nevertheless, all three tests concurred: over the full size range, even below 25 μm, GRG recovery averaged 50 to 75%.

3.2 Barrick Gold-Est Malartic Division

3.2.1 Introduction and Mill Description

The Est Malartic mill is located near the town of Malartic in the Abitibi region of north-western Quebec. The mill has been in production since the mid-1930's, processing mainly a clean, free milling gold ore. Late 1989, the mill was modified to treat a massive sulphide copper/gold ore from the Bousquet 2 mine located 40 km west of Malartic. Modifications included the addition of gravity and flotation circuits to recover free coarse gold and copper, respectively. The existing leach circuit was used to process the copper flotation tailings with some minor modifications to the solution handling system and the addition of an SO_2/air cyanide destruction circuit. The throughput was raised from 1500 to 2500 tons per day producing gold gravity and copper flotation concentrates and a gold precipitate[13].

The Bousquet 2 orebody is a massive sulphide zone consisting mainly of pyrite, chalcopyrite and bornite. Parallel zones of brecciated and disseminated sulphides carry lesser amounts of pyrite, copper mineralization and gold. The current ore grade is 7-10 g/t (0.2-0.3 oz/st) gold and 0.7% Cu. The ore from the high grade massive sulphide core has run more than 2.5% Cu and carried an ounce per ton of gold[13].

The flowsheet of the mill is shown in Appendix A, on page 93. The ore is ground in a crushing plant with a conventional three-stage circuit consisting of a jaw and two cone crushers. The -12 mm (1/2") ore is further ground in an open circuit rod mill, then in a ball mill in closed circuit with classifying cyclones to produce an 80% passing 70 µm flotation feed. A portion of the cyclone underflow is screened at 1.7 mm and fed to a gravity concentration circuit consisting of two 76 cm CD Knelson Concentrators operating continuously and two Gemeni tables for final upgrading. The gravity circuit tailings are partially dewatered using cyclones before they are returned to the ball mill circuit[13]. The main objective of this testwork was to analyse the performance of a PKC when processing a high gangue density feed.

3.2.2 Sampling and Test Procedure

The mill personnel sent a 25 kg sample of PKC feed and 18 kg of PKC tails to McGill. The +1.7 mm was removed from both samples. Four LKC tests were performed: with the original feed sample, T1; with the -300 µm (50 mesh) fraction of the feed, T2; with the + and -300 µm fractions of the tails sample, T3 and T4, respectively. Processing a -300 µm sample was meant to maximise fine gold recovery by minimising erosion of the LKC concentrate bed by coarse dense gangue particles. All samples were processed at a feed rate between 380 and 440 g/min. The pressure of the water jacket on the LKC ranged from 21 to 30 kPa. LKC samples were extracted, processed and assayed using the standard procedure described in the previous section. Assaying was performed at the Est-Malartic laboratory. Details of the LKC tests are shown in Appendix B (pages 99-101).

3.2.3 Results and Discussion

Table 3.3 summarises the metallurgical test results of the four samples processed, detailed in Appendix B. The original feed contained 57% GRG and the LKC upgraded a feed of 1.18 oz/st into a concentrate of 60.8 oz/st in T1; when only the 300-850 μm fraction of T1 is considered, recovery increases to 73% in a 27.8 oz/st concentrate, from a 0.80 oz/st feed; T2, the -300 μm feed, had a slightly higher grade, 1.23 oz/st, as was the LKC recovery, 66%, in a concentrate assaying 64.8 oz/st. Removal of the +300 μm clearly increased recovery of the -300 μm fraction from slightly less than 57% in T1 (57% is the weighted average of the recovery of the +300 and -300 μ fractions) to 66% in T2. The PKC tail sample yielded very different results: whereas T3, the -300 μm fraction, contained virtually as much GRG as the feed (60% vs. 66%), the +300 μm tails contained much less GRG, 31% vs. 73% (the +300 μm of T1). This showed that the plant could recover GRG much more effectively from a +300 μm feed; unfortunately only 9.4% of the gold fed to the Knelson was in this range.

Table 3.3 LKC metallurgical performance of the Est-Malartic PKC samples

	T1 Feed -1.77 mm	T1 Feed +300 μm*	T2 Feed -300 μm	T3 Tails -300 μm	T4 tails +300 μm
Feed, oz/st	1.18	0.80	1.24	1.22	0.42
GRG content, %	56.7	72.7	65.4	60.4	31.3
LKC conc. grade, oz/st	60.9	27.8	64. 8	91.0	3.07
non-GRG grade, oz/st	0.52	0.22	0.44	0.49	0.31

*Excerpt from the results of T1 Feed test

Figure 3.4 presents the GRG of the various tests as a function of particle size. Results were typical of a high density gangue material in the fine size range[22]: gold accumulated in the circulating load between 37 and 150 μm, the finer gold was rejected to the cyclone overflow (T1 and T2 distributions) and the coarser gold was either absent in the ore or ground too rapidly to accumulate in the circulating load.

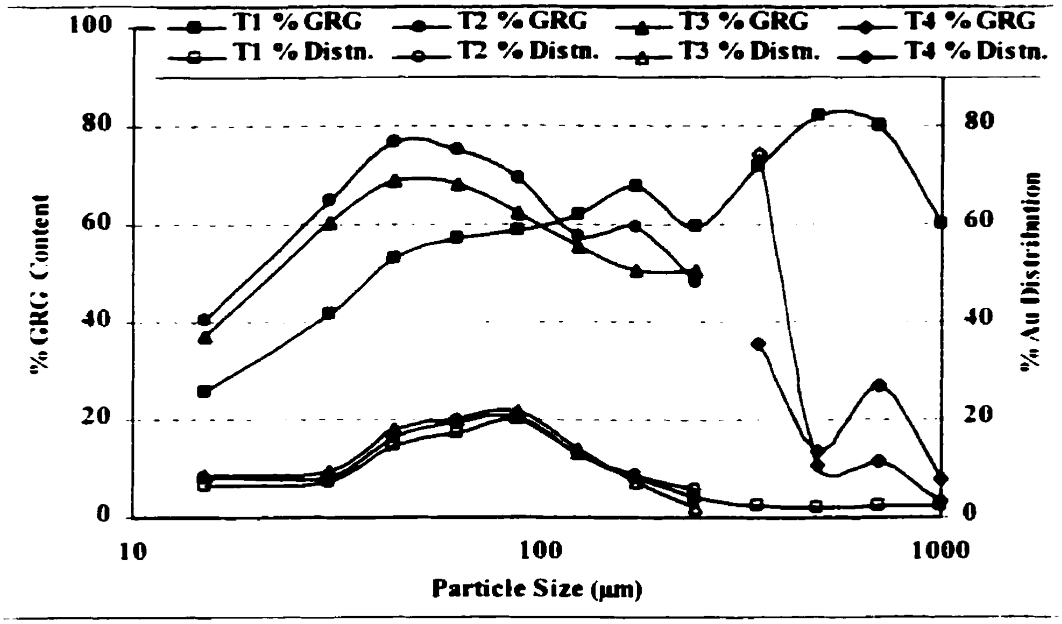


Figure 3.4 Size-by-size GRG content in the Est Malartic PKC samples

The -300 μm feed curve (T2) validates the results obtained for T1 inasmuch as the two size distributions of gold were in good agreement, with most of the gold between 37 and 105 μm. Removing the +300 μm resulted in a very significant recovery increase for the -300 μm, from 52% to 69%: most of the recovery increase was below 100 μm. The +300 μm fraction of the feed contained a high proportion of GRG, which identified very coarse gold grains. The -300 μm of the tails showed very similar results to the PKC feed, with only slightly less GRG, and a similar gold distribution.

If the PKC was recovering all the GRG, clearly none would be left in its tail. However, if no recovery took place, the GRG content of the feed and tails of the PKC would be identical. Thus the difference between the two GRG contents (i.e. LKC recovery) is a good measure of the effectiveness of the PKC. Its increase from 3 to 10% as particle size increased from -25 μm to 212-300 μm in Figure 3.5 demonstrated that the Knelson's performance worsened with decreasing particle size. Table 3.3 also showed that there was much less GRG in the +300 μm fraction of the PKC tails than its feed, 31% vs. 73%, another confirmation that Knelson efficiency is size dependent at Est Malartic.

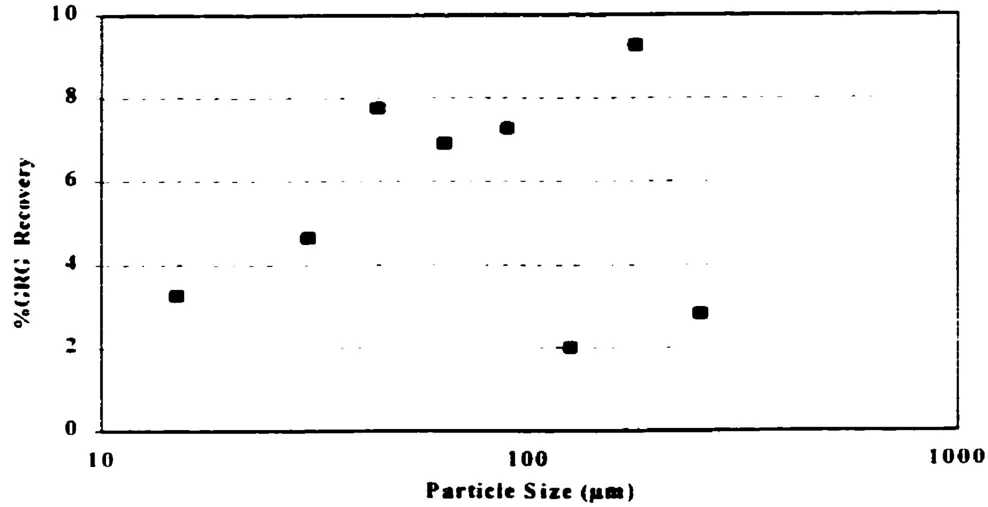


Figure 3.5 Difference between the GRG of the Est Malartic PKC feed and tails as a function of particle size

The above data can also be used to estimate GRG recovery. The GRG content of the feed and tail of the PKC must first be estimated carefully. Of critical importance was the need not to underestimate its content by failing to consider the effect of gangue density in LKC performance. Thus, we considered that the GRG content of the -300 μm fraction was that measured with the -300 μm feed tests (i.e. tests T2 and T3). The +300 μm GRG content was estimated from tests T1 and T4. The GRG content of the full size distribution was the weighted average of the -300 and +300 μm fractions. The grades of T3 and T4 were combined together to obtain an overall grade for the PKC tails: 1.12 oz/st. Using the grade of the PKC feed T1, total gold recovery (Eq. 3.1) was calculated to be 5.1%, with an error of 2.9% (assuming an error for both the feed and tails of 0.02 oz/st and 0.03 oz/st respectively, and using Taylor Series to estimate error propagation).

The same procedure applied to GRG yielded a grade of 0.79 oz/st in the PKC feed and 0.66 oz/st in its tails. With these data, Eq. 3.2 yielded a GRG recovery of 16%, or 0.13 oz/st. It should be noted that 46% of the gold recovered was above 300 μm, although only 9% of the Knelson feed gold reported to the +300 μm. The 0.13 oz/st recovered by the plant Knelson can yield an estimate of gold production. The feed rate, at the time of the testwork, was estimated to be around 40 t/h. Thus 5 ounces of gold was collected per hour, or 122 ounces per day. Based on a daily mine production of approximately 600 ounces (2000 tons at 0.3 ounces/t), this corresponded to a recovery of 20% by gravity, in good agreement with gravity production estimates reported in the literature[13].

Virtually all of the gold targeted for additional recovery was finer than 300 µm and accumulated in the circulating load. Finer screening (-300 µm) of the feed prior to gravity concentration could prove to be beneficial since it was shown that, at laboratory scale, this yielded a substantial increase in Knelson recovery[3,15]. However, should the significant proportion of +300 µm gold that made up the concentrate be confirmed by further testing, it might be questionable not to feed the +300 µm to the Knelson. This +300 µm would not be entirely lost to gravity, as laboratory work showed that when GRG was ground, most of the product (typically 95%) was also GRG, although finer. The small difference of GRG in the PKCs feed and tails in Figure 3.5 below 300 µm suggested that the unit might not operated at optimum conditions. One problem that has been identified was the high feed rate. One compromise between fine recovery with a -300 µm feed and coarse gold recovery would be to operate one Knelson with the present feed for coarse gold recovery, and a second one with finer feed for fines recovery. Water balance considerations would dictate whether or not this approach is feasible.

Feed and tails samples alone were inadequate to calculate a size-by-size recovery curve or a mass balance because their gold content were too similar. A sample of the Knelson concentrate and a good estimate of its weight are needed.

3.3 Hemlo Gold Mines

3.3.1 Introduction and Description of the Hemlo Mill

The Hemlo Golden Giant Mines (now Battle Mountain of Canada Ltd.) mill is located about 25 km east of Marathon on the north shore of Lake Superior, Ontario. Ore reserves total 20.8 millions tons with a grade of about 8.7 g/t (0.25 oz/st). Milling started in 1985. In 1989, the mine processed 3000 t/day at an average grade of 13 g/t. Grade is variable and monthly average can range from a low 8 g/t to a high of 20 g/t[12,15,17].

The gangue contains predominantly quartz, feldspar, sericite, pyrite, molybdenite and vanadium bearing mica. Visible gold occurs within quartz pods or along fractures.

The orebody is typically massive, up to 40 meters wide (unlike typical gold mine that are silica veins). The gold has been described as finely disseminated with visible gold only rarely seen in the ore[12,15,17].

The mill flowsheet is illustrated in Appendix A, on page 94. Prior to the installation of the gravity circuit, gold was recovered using cyanidation and a CIP circuit. The ore is reduced to -12 mm in a crushing plant consisting of a 2.5 x 4.9 m double deck vibrating scalping screen (19 mm), an open circuit 2.13 m standard crusher, a closed circuit 2.5 x 6.1 m double deck vibrating secondary screen (13 mm) and a 2.13 m short head crusher. The grinding circuit, which processes most of the ore, consists of three 3.7 x 4.3 m ball mills and two 25.4 mm cyclopaks. The underflow of one of the four primary cyclones is fed to a 76 cm PKC to remove coarse gold while the final ground product, 80% passing 75 mm, reports to a 41 m thickener and is then pumped to the cyanidation circuit[12,15,17].

3.3.2 Objectives

In 1991, Banisi[15] investigated the behaviour of gold in the Hemlo grinding circuit and found that gold was ground 6 to 20 times slower than the gangue and reported to the cyclone underflows at a cut-size finer than that of the gangue resulting in high gold circulating loads. Recommendations were to implement a gravity circuit to take advantage of the circulating load at the PCU or at the secondary mill discharge treating only a portion of the stream. A jig had been installed at the discharge of the primary mill but its use was discontinued due to operational difficulties and the perception of lack of economic impact. Later on, plant test work showed gold nuggets as large as 184 grams were found in the crushing plant, and a significant amount of gold flakes with a diameter of up to 2.5 cm was found in pump boxes and behind liners. When leach tails were processed with a LKC, a concentrate with a content of 60 to 100 g/t of free gold was obtained, which contained gold that had not dissolved in the leach circuit. Leaching time was then extended and cyanide concentration was increased to recover that gold but no recovery increase was detected[12,15,17].

As a result of the above findings, a PKC was installed at the discharge of the PCU where the gold circulating load was found to be above 6000%, with a fair amount of the ore coarser than 300 µm. During the first four months of the PKC operation, the rougher stage was first optimised: a two-hour cycle was chosen which resulted in a recovery around 50% at a rougher grade of 3% gold, upgraded on the table to 75-80% gold for an overall recovery between 30-35%. The product was sent directly to the refinery. Results showed that roughly 45% of the gold was in the +*100 µm size range (large free gold* grains). Cost savings were in the order of 250 000$/yr since removing 30% of the gold by gravity made it possible to reduce the number of the strip stages to one per week instead of two[12,15,17]. The objective of this test work was to characterise the performance of the PKC after more than a year of operation.

3.3.3 Sampling and Test Procedures

Two sampling tests (T1 and T2) were performed to characterise the PKC performance, which was done by taking samples of the feed and tails over a full cycle of operation. Sample weight was increased from 20 kg for the feed and 8 kg for the tails (T1) to 75 kg and 50 kg (T2), respectively, to minimise the fundamental error of sampling. All samples were screened at 850 µm and the undersize, about 95% of the material, was processed with the LKC at a feed rate between 396 and 739 g/min, and a water jacket pressure ranging from 38 to 33 kPa (4-4.6 psi). The LKC tails of T2 plant samples were reprocessed (T3) at a feed rate of 950 g/t and a water jacket pressure between 28 and 29.4 kPa (4-4.2 psi). LKC samples were extracted, processed and assayed using the standard procedure. Assays were made at the Hemlo laboratory. Details of the LKC tests are shown in Appendix B (pages 101-103).

3.3.4 Results and Discussion

Table 3.4 shows the overall results of the processing tests; size-by-size results are in Appendix B. For T1, the samples responded well to the LKC with total and size-by-size recoveries all above 62%. The PKC feed assayed 1.06 oz/st of gold and the PKC tails 0.94 oz/st. The GRG contents were 72.8% and 66.8%, respectively. The LKC tail grades of the feed and tails of the plant unit were in good agreement, 0.29 vs. 0.32 oz/st. Calculations showed that only 0.12 oz/st of gold from the feed was actually recovered by the plant unit, while the lab unit indicated that at least 0.77 oz/st, out of the 1.06 oz/st, was gravity recoverable. This yielded a total gold recovery of 11%, and a GRG recovery of 19%. These recoveries were closer to those of Est Malartic than Meston, although the Hemlo ore has a relatively low density, with a sulphide content of 5 to 10%, and is not particularly coarse. The low plant recovery can probably explained by the high feed rate to the plant Knelson, 70-80 t/h (well in excess of the recommended 40 t/h).

Table 3.4 LKC metallurgical performance of the Hemlo PKC samples

	T1		T2 and T3	
	Feed	Tail	Feed	Tail
Feed grade, oz/st	1.06	0.94	5.43	4.32
GRG content, %	72.8	66.8	86.7	82.8
LKC conc. grade, oz/st	93.4	38.5	412	312
non-GRG grade, oz/st	0.29	0.32	0.73	0.75
Total gold recovery, %	11.3		20.4	
GRG recovery, %	18.6		24.0	

For T2, the large weight and exceptional gold grade of the samples processed resulted in a LKC overload, especially for the feed sample, whose LKC concentrate assayed 2122 oz/st. As a result, not all GRG was recovered, and the GRG grade was underestimated, especially for the PKC feed. LKC tails assays were also noisy, an indication of the presence of GRG. To remedy this problem, part of the LKC tails of both samples were reprocessed with the LKC (T3). Combined results of the two tests are shown in Table 3.4. Again, the ability of the LKC to recover all the GRG was illustrated by the similarity of the PKC feed and tails non-GRG content (i.e. the LKC tails gold assay, 0.73 vs. 0.75 oz/st). The amount of GRG was high in both samples, both in actual grade and % of the total gold. The PKC recovered 20.4% of the total gold, and 24.0% of the GRG; the two recoveries were similar because so much of the gold is GRG. Despite the large difference in gold content of tests T1 and T2/T3, Knelson performance, when expressed in terms of GRG (rather than total gold) content, was similar, 19 vs. 24%.

Figure 3.6 shows the size-by-size plant Knelson feed size distributions and GRG content (calculated using Eq. 3.2) for both tests. The distributions were fundamentally different, as only 25% of the gold was coarser than 150 μm in T1, compared to 56% in T2/T3. The differences reflected the presence of coarse gold in T2/T3, which also increased the grade of the circulating load. GRG recovery was noisy, as it relied on very few assays. Recoveries of some size classes were even negative, as their GRG content in the PKC tails exceeded that of the feed. As a result, no definite trend was observed. This in itself may suggest no strong relationship between GRG recovery and particle size (as was the case at Meston), but only a representative sample of the PKC concentrate would confirm this. Size-by-size data showed variability, despite the large weights processed. This points again to the need to extract a concentrate sample to evaluate its gold size distribution from which size-by-size recoveries can be evaluated reliably. The large sample weight processed did, however, limit the uncertainty as to total gold and GRG recovery. It was observed that a significant increase in GRG recovery, from 19 to 24%, parallels a shift from finer to coarser GRG in the PKC feed from T1 to T2/T3. This provided indirect evidence that there was at least a slight drop in GRG recovery with decreasing particle size.

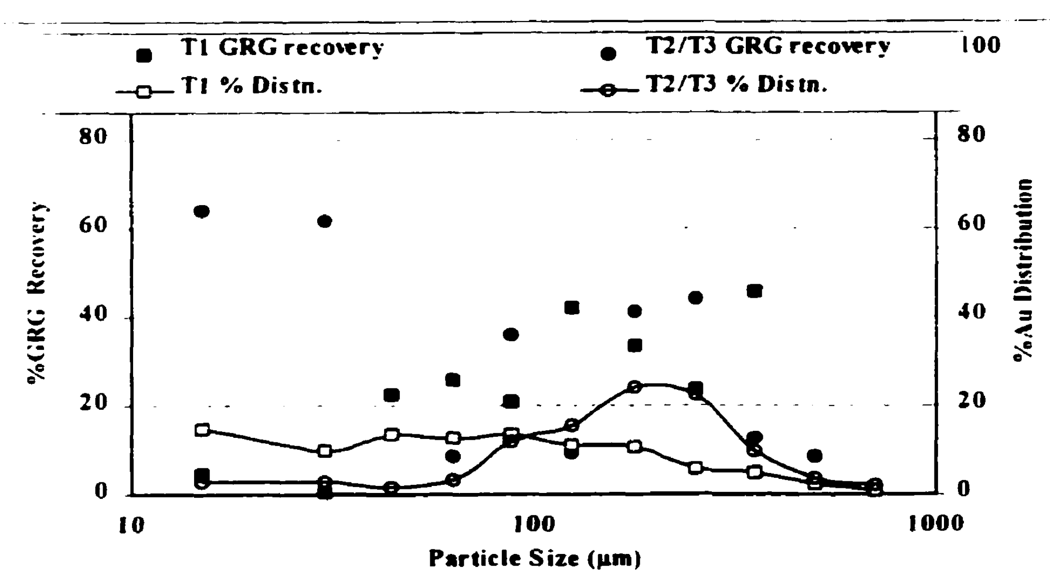


Figure 3.6 Size-by-size total gold distribution in the Knelson feed sample and PKC GRG recoveries of the two Hemlo tests

3.4 Aur Resources- Division Aurbel

3.4.1 Introduction and Description of the Mill

The Aurbel mill treats ores from the Astoria and Aurbel mines by gravity, flotation, regrind and cyanidation of the flotation concentrate. The mill flowsheet is illustrated in Appendix A, on page 94. After the crushing circuit, the ore goes through a rod mill (4 m x 8 m) in open circuit with four ball mills, then the discharge is pumped to four 39 cm (15") cyclones in series whose overflows feed three 30 cm (12") cyclones. The secondary overflows go to flotation. The primary and secondary underflows are fed to a sluice. The sluice tails are recycled to the cyclones while the concentrate is screened at 1.41 mm (12 mesh) and fed to two 51 cm PKCs. The Knelson tails are recycled to the primary cyclones. The Knelson concentrate is passed through a magnetic separator and fed to a shaking table. The table concentrate is melted to bullion. The tailings are combined to the magnetic concentrate and cyanided in a column, then recycled to the grinding circuit[27].

3.4.2 Objectives, Sampling and Test procedure

An evaluation of the PKC performance was performed when gold recovery by gravity decreased after the Aurbel and Astoria ores were processed together (50:50) at the mill. The plant Knelson feed and tails were sampled by Aurbel and McGill personnel[27].

Two sampling tests (T1 and T2) were performed to characterise the gravity circuit. The first was conducted in May 1994 and the second in September 1994. All samples were screened at 850 μm and the undersize was processed with the LKC at a feed rate between 430 and 460 g/min, and a water jacket pressure of 29 kPa (4.1 psi). LKC samples were extracted, processed and assayed using the standard procedure. McGill personnel completed sample processing and the assaying was done at the Aurbel laboratory[27]. Details of the LKC tests are shown in Appendix B (pages 104-105).

3.4.3 Results and Discussion

Table 3.5 shows the LKC performance of the Aurbel PKC samples. For the first test. there was 0.437 oz/st of GRG in the Knelson feed and 0.035 oz/st in the Knelson tails. This corresponded to a plant GRG recovery of 92%. Therefore. the PKC is particularly efficient. but at a relatively low feed rate of 3 t/h. For T2. the feed rate to the PKC was increased to 5 t/h. by opening the concentrate gate on the pinched sluice. Table 3.5 shows that GRG recovery then dropped to 71%. The increase of 60% in feed rate more than compensated for the 20% drop in recovery.

Table 3.5 LKC metallurgical performance of the Aurbel samples

	T1		T2	
	Feed	Tail	Feed	Tail
Feed grade. oz/st	0.95	0.45	0.56	0.39
GRG content. %	45.9	7.9	40.5	17.1
LKC conc. grade. oz/st	60.6	5.58	13.4	7.39
non-GRG grade. oz/st	0.52	0.42	0.34	0.32
Feed rate. t/h	3.00		5.00	
Total gold recovery. %	52.6		30.3	
GRG recovery. %	91.8		70.7	

Figure 3.7 shows that most of the gold. 74%. was finer than 300 μm: this proportion was even greater for the GRG. There was little fine gold (-37 μm) in the Knelson feed. as the grinding circuit produced a coarse flotation feed with coarser cyclone cut-sizes. thereby classifying fine gold to the cyclone overflows. With a more typical grind of 80% -75 μm. a more important charge of fine gold would build up in the grinding circuit and its GRG content would then increase. The PKC feed itself was very coarse. 41-43% +300 μm for T1. and contained 5-10% arsenopyrite for the Astoria ore. Thus. the relatively low feed rate (51 cm PKCs were designed to treat up to 20 t/h) can compensate for the coarseness of the feed. its arsenopyrite content (s.g.: 6.0) and the absence of much coarse GRG.

Figure 3.7 shows the PKC GRG size-by-size recoveries. calculated from the size-by-size GRG content (Appendix C) and Eq. 3.2. As expected. the best results (>80%) were obtained with the initial plant feed rate. As the feed rate was increased (T2). GRG

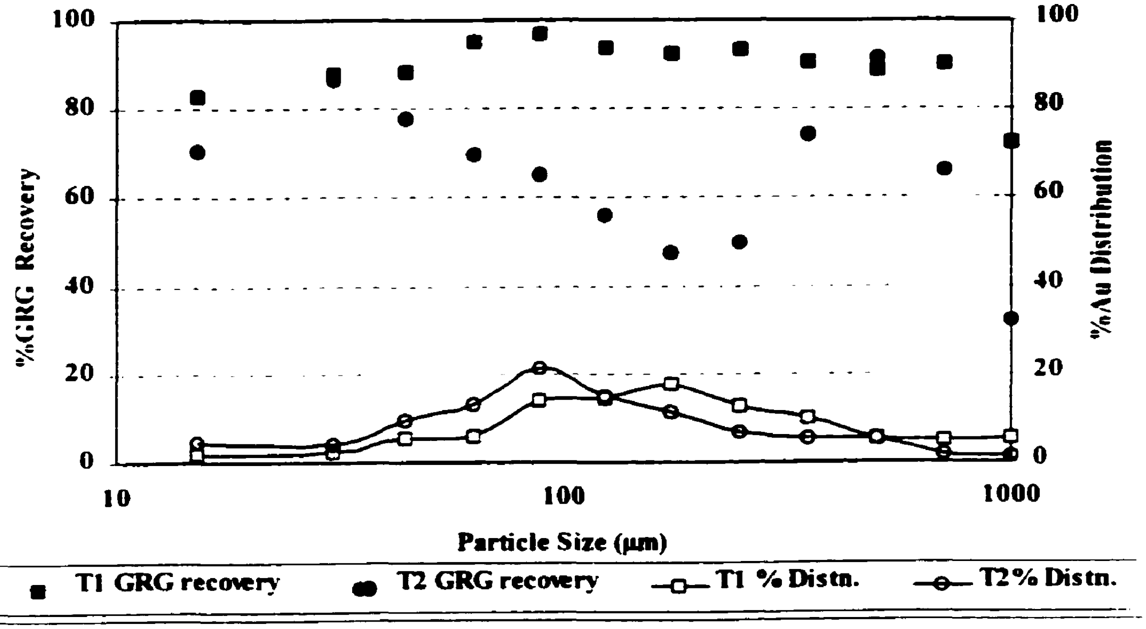


Figure 3.7 Size-by-size total gold distribution in the feed sample and GRG recovery for the Aurbel PKC samples

recovery decreased. The recovery drop appeared to be over the full size range (as opposed to a larger drop at fine size). Coarse GRG data were unreliable for both tests on account of very low GRG content and poor sampling statistics. The drop in recovery at intermediate particle size for T2 has been observed in the LKC before, and could indicate poor percolation due to a fluidization water flow rate below the optimum.

3.5 Agnico-Eagle

This section will present an example of how GRG behaves in a classification unit when treating an ore with a high sulphide content. It also makes a significant contribution to the determination of GRG content in high density samples.

3.5.1 Description of the Agnico-Eagle Mill

The Agnico-Eagle Laronde Division (AELR) mill, located near Cadillac, Québec, treats by flotation and cyanidation 1700 tons of a pyrite-chalcopyrite ore grading 8-10 g/t Au with 50% sulphides.

The AELR grinding circuit is shown in Appendix A, on page 95. After three stages of crushing, ore is fed to two separate primary grinding lines. Two 3.5 x 5.2 m ball mills grind the ore to a P_{80} of 300 µm. A portion of each mill discharge was bled from the circuit via pinched sluices and redirected to a secondary grinding stage which can be operated either as a regrind section for the flotation concentrate when chalcopyrite liberation is more difficult to achieve, or as a secondary mill in the grinding circuit when extra capacity is required. Otherwise it is not used. Buonvino[22] investigated various streams of the circuit with the use of a Falcon and a Knelson (1992). At that time, a secondary grinding circuit was part of the grinding circuit and the pinched sluice concentrates (roughly 10% of the mill discharge flow) were directed to it and were ground to a P_{80} of 150 mm (100 mesh) in a closed circuit cyclone/ball mill circuit. The sluice tails were combined with the secondary cyclone overflows and recirculated to the primary cyclones. The primary cyclone overflows are always combined and pumped to the flotation circuit for further treatment.

Buonvino[22] found a high gold circulating load of 3700% with high gold assays for the regrind mill discharge and the SCU, which clearly indicates that gold is building up in the regrind circuit. Falcon performance was disappointing but limited testing with the Knelson showed obvious promise. Testwork by Woodcock[14] to characterise the ore's potential for gravity recovery had shown that 50% of the gold in the ore is GRG, with very little GRG reporting to the +300 µm fraction. Given the high density of the gangue, removing the virtually barren +300 µm prior to processing if a plant Knelson was installed is the best approach to gravity. More specifically, the ability of the primary cyclones to direct GRG to their underflows had to be quantified if any gravity circuit had to be installed. More testwork was needed to see how much gold could be recovered from the primary cyclone underflow prior to flotation. A reduction in the flotation concentrate gold content, which accounts for roughly to 40 to 50% of the gold in the feed, was targeted.

3.5.2 Sampling and Test Procedure

Samples of the primary cyclone feed (PCF), overflow (PCO) and underflow (PCU) were extracted by AELRD personnel and sent to McGill. Samples were pre-screened at 840 µm (20 mesh). The undersize was processed with the LKC at a feed rate ranging from 377 to 473 g/min and a water jacket pressure between 21 and 35 kPa (3-5 psi). To maximise recovery, generate redundant data and assess the effect of feed top size on Knelson performance, a portion of the samples were treated as is (T1), another was processed with the prior removal of +300 µm (T2), and a third was diluted with 212 µm silica (T3) in a 2:1 dilution ratio. LKC samples were extracted, processed and assayed using the standard procedure. Assays were made at the Agnico-Eagle laboratory. The head grade of the original LKC feed was then back-calculated from concentrate and tails assays. Details of the LKC tests are shown in Appendix B (pages 105-107).

3.5.3 Results and Discussion

Table 3.6 summarises the results of the LKC tests. The calculated head grade of the original feed was 2.09 oz/st: 72% of its gold was gravity recoverable. When the -300 µm fraction was processed, feed grade increased slightly to 2.16 oz/st because the fraction removed (300-840 µm) had a slightly lower grade (0.45 to 1.04 oz/st). GRG content dropped slightly to 70%. With dilution, gold recovery increased to 78%. The back-calculated grade of the diluted feed was 0.55 oz/st, which yields of a grade of 2.20 oz/st prior to dilution, in good agreement with the other samples. The GRG content was higher, even above 300 µm, where less than 3.21% of the gold in the feed reported.

Table 3.6 LKC metallurgical performance of the Agnico-Eagle samples

	T (- 840 µm)	T2 (- 300 µm)			T3 (SiO_2 dilution)		
	Feed	Feed	U/F	O/F	Feed	U/F	O/F
Feed grade, oz/st	2.09	2.16	2.81	0.28	2.20	2.60	0.24
GRG content, %	72.0	69.6	73.9	23.1	77.5	80.0	14.3
LKC conc. grade, oz/st	70.35	49.99	111.0	1.64	241.1	296.4	3.32
non-GRG grade, oz/st	0.60	0.68	0.75	0.22	0.48	0.52	0.20

For the underflow. Table 3.6 shows that the -300 μm material assayed 2.81 oz/st. and had a GRG content of 74%. Silica dilution increased GRG content to 80%: the back-calculated grade, 2.60 oz/st. was slightly lower than that of T2. The overflow. which contained only 2.1% of +212 μm. was first processed as received. Its grade was found to be 0.28 oz/st, with a GRG content of 23%. Diluting it with coarser silica sand lowered the amount of GRG to 14%; the back-calculated head grade was 0.24 oz/st. The content drop was probably due to the size distribution (150-300 μm) of the silica used for the dilution. coarser than the sample itself. Clearly. the correct GRG content was that of the undiluted test (T2). 23%. which was similar to many tests of cyclone overflows (typically 15 to 25% GRG).

The LKC yielded high GRG contents with the PCF and PCU because they are recycled streams in which liberated gold builds up. Gold content was higher with the underflow (74% vs. 70%) because much of the liberated gold in the feed whose shape and size make it non-GRG reports to the overflow.

Cyclone Mass Balance

Mass balancing of the classification circuit (ore. gold) was performed with the NORBAL2[28] software using size-by-size and overall data. The software uses non-linear mass conservation equations to achieve a hierarchical decomposition that separates mass balance problems into smaller elements. Each component of the decomposition is described as a least square problem under constraints and is solved by the Lagrange multipliers method. As an example. if a circuit contains four streams corresponding to one node. the twelve constraints for the mass conservation of twelve size classes (pan included) can be expressed by:

$$W(1,i)C(1,i)-W(2,i)C(2,i)-LW(3,i)C(3,i)=0 \qquad \textbf{(3.3)}$$

where W(j,i) : percent retained on screen i (i=1 to 12) in stream j (j=1 to 3)

C(j,i) : gold content of the size class i of stream j

L : circulating load

In order to adjust the size-by-size assays. a Lagrangian formula is used:

$$\underline{dc} = -VB'(BVB')^{-1}B\underline{c} \qquad \textbf{(3.4)}$$

where $\underline{dc}$: 12x1 column matrix of the grade adjustments

V : 12x12 diagonal matrix of the variances

B : 1x12 matrix expressing the mass balance constraints

$\underline{c}$: 12x1 column matrix of unadjusted grades

The balanced grades will be equal to:

$$\underline{C} = \underline{dc} + \underline{c} \tag{3.5}$$

where $\underline{C}$: 12x1 column matrix of the adjusted grades

After obtaining the adjusted ore size distribution, size-by-size grades, and overall grade, the gold size distribution can be estimated by:

$$\text{Gold\%} = \frac{\text{Weight\%} * \text{Grade}}{\text{Overall grade}} \tag{3.6}$$

The results of mass balancing of the T2 data are shown in Appendix C, on page 122. The size-by-size gold assays showed remarkably few adjustments. The circulating load, 533%, was slightly higher than the results of Buonvino (448%), and may be due to the increased toughness of the ore now processed at the mill, which contained more silicates.

To see how fine gold was recirculated or which gold accumulated in the circuit, the performance of the classifiers was measured in terms of their efficiency at separating smaller and heavier particles from larger lighter ones. The description of the separation can be illustrated by the classifier cyclone performance curve, which is a plot of Y, the mass fraction of feed size d reporting to the underflow, versus d, the mean (geometric) particle size. Y was calculated from:

$$Y = \frac{Uu_w}{Uu_w + Oo_w} \tag{3.7}$$

where U and O are the PCU and PCO mass flowrates; u_w and o_w are the mass fraction of each size class in the PCU and PCO, respectively. With total gold and GRG contents known for each size class, the classifier curve of total and GRG gold can be determined. Y for total gold was calculated by:

$$Y_{gold} = \frac{Uu_w u_m}{Uu_w u_m + Oo_w o_m} \tag{3.8}$$

where u_m and o_m are the gold assays of each size class. As for Y_{GRG}, the top and the bottom terms of Eq. 3.8 are multiplied by the GRG content for each size class. Table 3.7 gives the size distribution and gold content of the PCU and PCO, as well as the classification curves of the ore, total gold and GRG. The performance curves of total ore, total gold and GRG are shown in Figure 3.8.

Table 3.7 Classification curves for the Agnico-Eagle ore, total gold and GRG

Size (μm)	Weight%		Gold Grade, oz/st		Y, %	Y_{gold}, %	Y_{GRG}, %
	PCU	PCO	PCU	PCO			
+212	7.2	2.1	0.03	0.01	95.2	99.2	99.8
+150	16.3	5.7	1.59	0.08	94.3	99.7	100.0
+105	22.3	7.4	1.67	0.08	94.6	99.7	100.0
+75	29.8	10.9	1.85	0.09	94.1	99.7	99.9
+53	13.4	15.0	3.62	0.14	83.8	99.2	99.8
+38	4.6	16.8	11.0	0.29	61.0	98.2	99.4
-38	6.4	42.1	18.0	0.50	44.0	91.0	95.8
Total	100.0	100.0	2.69	0.28	84.2	98.1	99.4

The recovery curve for the ore showed only part of an S-shaped curve. The total solids recovery to the PCU was 84.2% and the cut or separation size, d_{50}, at which the particles of a given size and density have an equal probability to report to the overflow or to the underflow, laid close to 50 μm. The curve showed the typical hump of high density gangue ores, at 100-300 μm. The hump was due to the fact that most of the cyclone feed above 300 μm is silicate gangue, whereas it was sulphide gangue below 100 μm.

The total gold and GRG classification curves were similar. Recovery of gold to the underflow was high for all classes and slowly started to decrease below 38 μm. The d_{50} of gold was far below 38 μm and in fact cannot be determined without sub-sieve data. Gold was classified at a considerably smaller cut-size (d_{50}<25 μm) than the ore: 98.2% of the total gold and 99.4% of the GRG was recycled to the cyclone underflow.

The circulating load of GRG can be calculated as the product of the circulating load (533%) by the fraction of GRG in the PCU (73.9%) and the ratio of PCU to PCO grades (2.69/0.28). This yielded a very high value, 3784%, which was based on the -300 μm fraction only; the circulating load would be slightly lower for the full size distribution, as the +300 μm contained less GRG. Its high value was due to a combination of classification to the cyclone underflow and slow grinding. The behaviour of GRG was such that 99.4% of it reported to the PCUF, compared to 98.1% of the total gold and 84.2% for the ore itself.

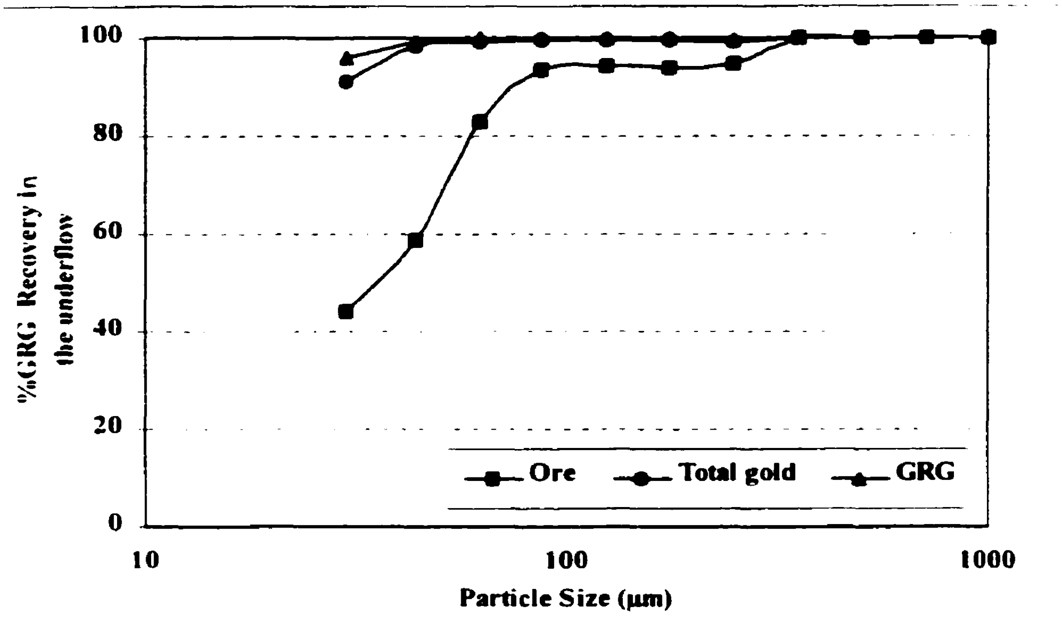


Figure 3.8 Classification curves for Agnico Eagle

3.6 Conclusions

Figure 3.9 compares the size-by-size GRG recovery of three industrial Knelson installations discussed in this chapter. Overall recoveries by gravity were 35-40% for Meston, 25% for Aurbel and 30% for Hemlo. All these plants have handicaps that limit gold recovery, except possibly Meston. Aurbel has two stages grinding with coarse classification in the first stage where gold is recovered; Est-Malartic treats a massive sulphide ore; Hemlo has also two grinding stages and uses a single 30" PKC to treat over 2000 t/d. All these have a significant impact on the gravity recovery.

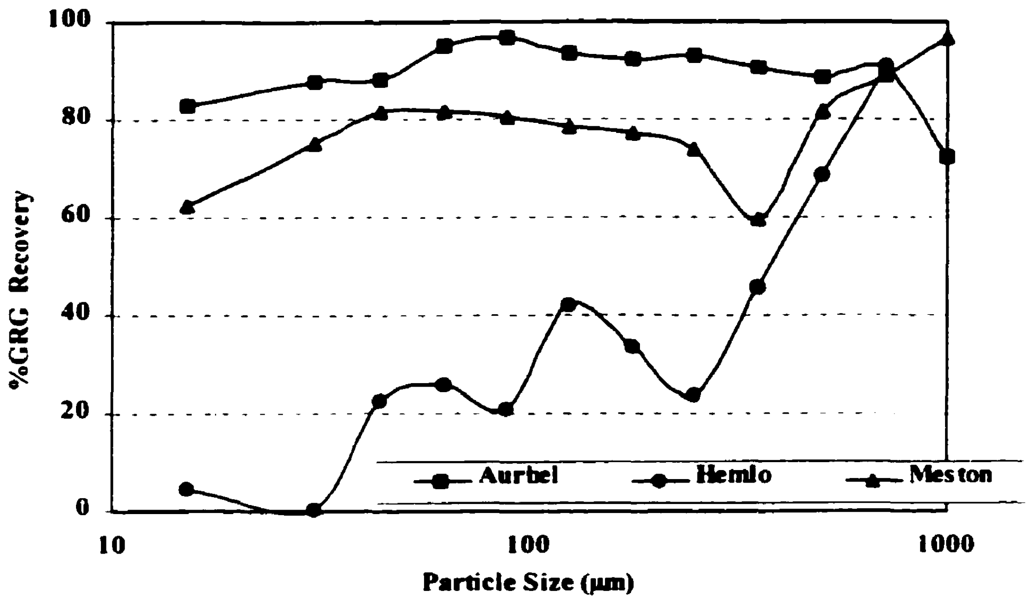


Figure 3.9 Size-by-size GRG recovery at three different mills

Stage recovery was not necessarily correlated to total gold recovery, as the highest stage recovery, that of Aurbel, corresponded here to the lowest gravity recovery, 25%. Gravity recovery was also a function of the GRG content of the ore, the magnitude of the recovery effort (i.e. the fraction of circulating load treated), and the recovery flowsheet. Gravity recovery was only 25% at Aurbel because the ore treated (particularly Astoria's) at the time of the sampling had a lower GRG content, the fraction of the circulating load treated was low, and grinding relatively coarse (65% -75 µm). At Hemlo, recovery could also be higher, as a GRG content of 72% was measured in the ore[25], but the circuit throughput was high for a 76 cm Knelson (2200 t/d), and gold was recovered only from the first of two grinding loops in series (GRG liberated in the second loop can not be recovered by gravity).

The Agnico-Eagle testwork showed that a cyclone which is properly adjusted (apex and vortex diameters) and operated can yield a very high GRG (and total gold, in this case) recovery to its underflow, even with a high density gangue.

Chapter Four

Test Work for Snip Operation

4.1 Description of the Snip Operation Mill

Snip, a 60:40 venture of Cominco and Prime Resources Group (at the time of the study), operated by the former, lies in the narrow mountain valley on the Iskut river, about 98 km from Stewart, British Columbia, and about 80 km east of the town of Wrangell, Alaska. Exploration of the area dates back as early as 1929 but the exploration and feasibility work that resulted in the present mine was carried out in the 1980s and production at Snip began in January 1991[14,29,30]. The main ore reserve, the Twin Zone, is a sheared gold-bearing quartz-carbonate-sulphide vein which contains three distinct ore types: streaky quartz ore consisting of quartz, calcite green biotite and sulphide laminae within strongly sheared greywacke; crackle quartz, composed of shattered quartz veins filled with green mica and disseminated sulphides; and massive sulphide veins containing mostly pyrite and pyrrhotite. In 1991, mining reserves were estimated at 936 000 tonnes grading 28.5 g/t for gold[14,29,30].

The flowsheet of the mill is shown in Appendix A, on page 95. The mill averaged 361 t/day throughput[14,29,30], and at the time of the sampling 459 t/day. Ore entering the mill is crushed to 100% -7.6 cm in a 7.3 x 11 m jaw crusher, followed by secondary crushing to 100% minus 0.95 cm in a 1.3 m shorthead cone crusher. Ore from the fine ore bin is fed to a 2.4 x 3.6 m ball mill operated in closed circuit with two 25.4 cm cyclones. The ball mill discharge is screened and fed to a double 5.6 x 5 m hutch Yuba-Richards mineral jig for gold recovery. Concentrate from the jig, continuously pulled from both hutches, proceeds to a No. 6 Deister table where gold is further upgraded. Coarse gold is extracted for smelting into bullion. The tailings from the jig are then classified. Table tailings and the cyclone underflow report to the ball mill. The cyclone overflow flows by gravity to a battery of flotation cells where a gold bearing bulk sulphide concentrate is extracted. The concentrate is filtered and bagged for sale. About 5% of the total volume of mined ore is extracted as concentrate[14,29,30].

In 1991, the gravity circuit captured 20% of the mill feed gold and produced a doré bullion on site. The flotation section recovered another 70% producing about 5000 t/yr of concentrate.

4.2 Objectives

The objective of this test work was to evaluate the Snip grinding and gravity circuits by characterising their performance in terms of GRG. Test work to characterise the GRG content in the ore was done by Woodcock[14] and Zhang[32] is shown in Figure 4.1. Both showed that the Snip material contained 58-61% GRG. Very little of GRG (<2 %) was coarser than 300 μm. The amount of GRG coarser than 105 μm was equally low, 16%. At the fine end, 12% of the total gold, or 20% of the GRG, was below 25 μm.

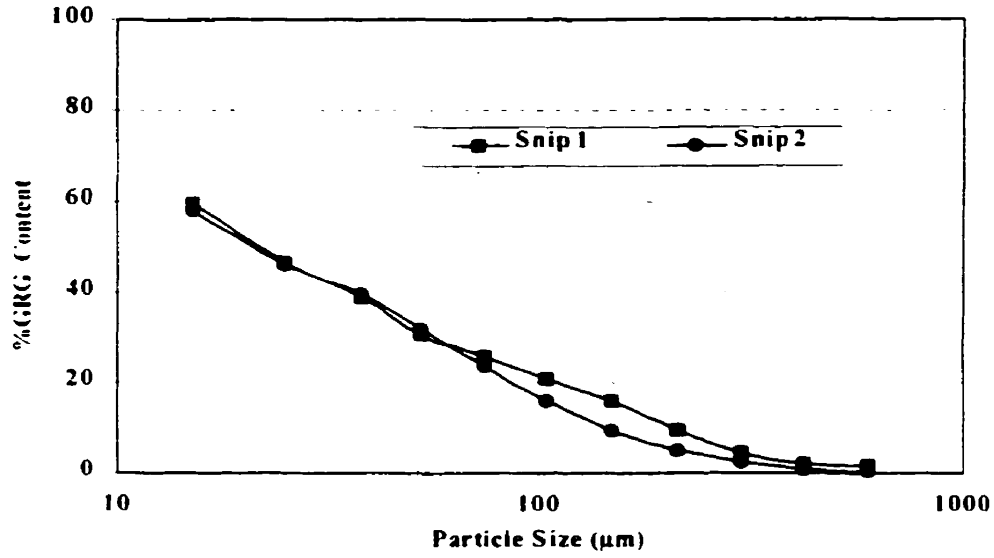


Figure 4.1 Size-by-size GRG content in the Snip ore.

4.3 Sampling and Test Procedure

Woodcock[14] performed a sampling campaign in the summer of 1992. Jig feed, the individual jig concentrates and jig tails were sampled when the jig was running at four different operating conditions (T1 to T4, Table 4.1). The feed samples were taken from the ball mill discharge at the discharge trommel screen, the tail samples at the jig tails stream as it dropped into the primary cyclone box and the timed concentrate samples at the two hutches. Cyclone feed (PCF), underflow (PCU) and overflow (PCO) were sampled by the mill personnel (T5). A second sampling campaign (T6) of the circuit, performed by the mill personnel in the summer of 1994, included cyclone underflow, table tails, jig tails and a combined jig concentrate. All samples were shipped to McGill University for treatment.

Table 4.1 Jig operating conditions

	T1	T2	T3	T4
Stroke frequency, rpm	84	84	117	104
Stroke length, min	9.5	9.5	7.9	9.5
Collection time, min				
Hutch 1	1.5	1.5	1.5	1.5
Hutch 2	2.0	1.5	1.5	1.5
Water flow	low	med. high	med. low	med. low

A 7.5 cm laboratory Knelson Concentrator (LKC) was used to further upgrade all samples and assess their size-by-size total and gravity recoverable gold (GRG) content. The table tails processing was done by L. Huang[33] and will be detailed later on. Samples were pre-screened at 850 μm (20 mesh). The undersize, about 80% of the material for most samples, was processed in the LKC at a feed rate ranging from 140 to 545 g/min. For the jig samples, the pressure of the water jacket on the LKC ranged from 14 to 31.5 kPa (2-4.5 psi), and from 35 to 40.6 kPa (5-5.8 psi) for the cyclone samples, finer feed requiring less pressure. LKC samples were extracted, processed and assayed using the standard procedure described in Chapter 3. Assaying was made at the Snip Operation analytical laboratory. Details of the LKC tests are shown in Appendix B (pages 107-118).

Two additional plant surveys were conducted by mill personnel, May 14 1993 and August 10 1994, and were labelled PS1 and PS2 (Davidson[31]), respectively. Samples of the grinding and gravity circuits were collected and screened, and each size class assayed for gold content. GRG content was not measured. Results are shown in Appendix B (pages 118-120). The plant surveys will be compared to the LKC testwork. T6 and PS2 classification samples should give similar results since they were sampled around the same time.

4.4 Results and Discussion

4.4.1 Plant Cyclones

Table 4.2 summarises how samples of cyclone feed, overflow and underflow responded to the LKC treatment.

Table 4.2 LKC metallurgical results of Snip cyclone samples

	T5			T6
	PCF	PCO	PCU	PCU
Feed grade, oz/st	1.97	0.37	2.77	3.04
GRG content, %	66.6	37.5	70.2	58.1
LKC conc. grade, oz/st	126.3	8.89	181.1	288.4
non-GRG grade, oz/st	0.67	0.24	0.83	1.28

Cyclone feed/Jig tail

The calculated cyclone feed was 1.97 oz/st. This is high for a grinding circuit with gravity, which should decrease the circulating load of gold, hence the cyclone feed grade. Overall GRG content in the cyclone feed was 67%. Size-by-size data can be found in Appendix B, on page 115. Most of the gold (77%) was finer than 75 μm, where GRG content (i.e. LKC recovery) was also the highest. This indicated a significant amount of fine liberated gold. Recovery was poor above 425 μm (less than 30%-i.e. there was little coarse GRG). This was in large part because there was little coarse GRG in the ore (only 2% coarser than 300 μm). Recovery between 300 and 75 μm slowly increased from 30 to 50% with decreasing particle size.

Cyclone underflow

The calculated cyclone underflow grades were 2.77 and 3.04 oz/st for T5 and T6, respectively, higher than that of the cyclone feed, which is consistent with the unit's ability to recover gold to its underflow in an upgraded product. Size-by-size data are shown in Appendix B, on page 116. For T5, the overall GRG content was very high.

70%. 60% of which below 75 μm. Size-by-size GRG content was high below 150 μm and low above 420 μm. much like the cyclone feed. and for the same reasons. In T6. overall GRG content was lower. 58%. This decrease was presumably related to changes in ore mineralogy between the two campaigns. High size-by-size recoveries started below 75 μm where 56% of the gold was located. As for the first campaign. recoveries were poor above 212 μm. Gold recovery increased with decreasing particle size.

Cyclone overflow

The calculated cyclone overflow grade was 0.37 oz/st. The overall GRG content was low. 37%: it was strongly influenced by the -37 μm size fraction where most of the gold (77%) was located. GRG recovery was high for a PCO. as reported values are typically 15-25%: this could indicate a poor jig performance. The size-by-size performance was typical for a cyclone overflow. as LKC recovery was poor in most size classes. and best for in the 25-37 μm fraction. 50-60%. Recovery below 25 μm was only 32%: since 42% of the gold was located in this size fraction (Appendix B. on page 115). the limited efficiency of the LKC in recovering fine. probably flaky gold which characteristically occurred in cyclone overflow. was again demonstrated.

Cyclone mass balance

The size distribution and size-by-size grades of the cyclone samples of T5. were adjusted and used to estimate circulating loads using NORBAL2[28] software (Appendix C. pages 124-126). The classification curves of the ore. total gold and GRG are shown in Figure 4.2. Recovery started to decrease around 53 μm. Gold recovery to the underflow was very close to 100% up to 75 μm where it started to decrease. Gold was classified at a considerably smaller cut-size than the ore. The total solids recovery to the PCU was 87%. at a cut size. d_{50}. of 53 μm. whereas for total gold and GRG. the d_{50} was far below 38 μm and could not be determined without sub-sieve data. Total gold and GRG reported to the PCU in the proportions of 97.4% and 98.6%. respectively. Their classification curves were very similar.

Results gave a large circulating load of gold of 1757%, and a GRG circulating load of 3289%, compared with 267% for the ore. For PS1, the circulating load for gold was 2060% and 408% for the ore. For PS2, the circulating load for gold is 2081% and 481% for the ore (Appendix C, on page 127).

The relative abundance of fine gold in the PCU of T6 was apparent and represented the target of the gravity system. In the -850 µm of the cyclone feed, only 6% of the gold was coarser than +212 µm. Less than 30% of this coarse gold was GRG, compared to 67% for the overall -850 µm.

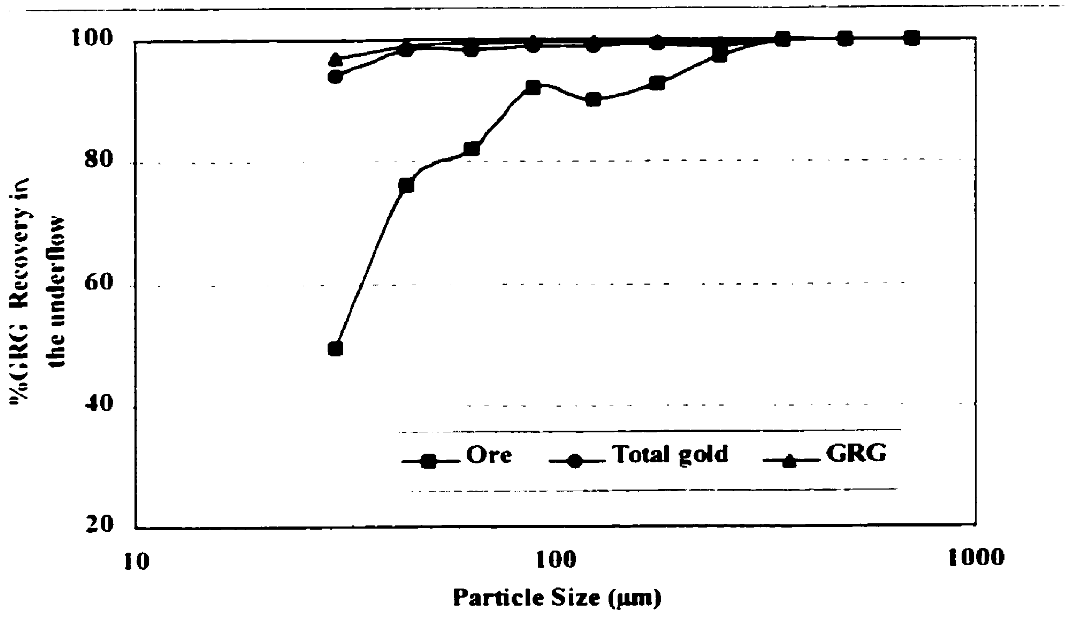


Figure 4.2 Snip classification efficiency curves

4.4.2 Gravity Circuit

Jig feed

The jig feed samples were processed in the LKC at a pressure of 25.2 kPa (3.6 psi), (except for T1, 14 kPa), and at a feed rate between 246 and 340 g/min. The results of LKC tests are summarised in Table 4.3. Back-calculated feed grades varied from 2.26 to 4.33 oz/st. Jig feed grades were 3.93 oz/st and 2.33 oz/st for PS1 and PS2 plant surveys, respectively. Grade fluctuations were imputed to changes in jig efficiency and gold circulating loads. T4, which had a feed grade close to the feed grade of T1, gave a LKC concentrate grade twice as high. This was simply a reflection of the mass treated in the LKC (4.6 kg for T1, 7.6 kg for T4): the LKC gold recovery did not change significantly as the mass treated increased and the mass recovered was virtually constant: it followed that treating more mass increased concentrate grade. The GRG content of T2-T4 was remarkably constant, 69%: that of T1, 58%, may well be due to a lower fluidization water pressure, as the drop in recovery corresponded to the size range were low pressures are known to decrease recovery[3].

Table 4.3 LKC metallurgical results of Snip jig feed

	T1	T2	T3	T4
Feed grade, oz/st	2.26	4.18	4.33	2.49
GRG content, %	57.6	69.0	69.4	69.0
LKC conc. grade, oz/st	64.2	174.3	185.8	130.9
non GRG grade, oz/st	0.98	1.32	1.34	0.78

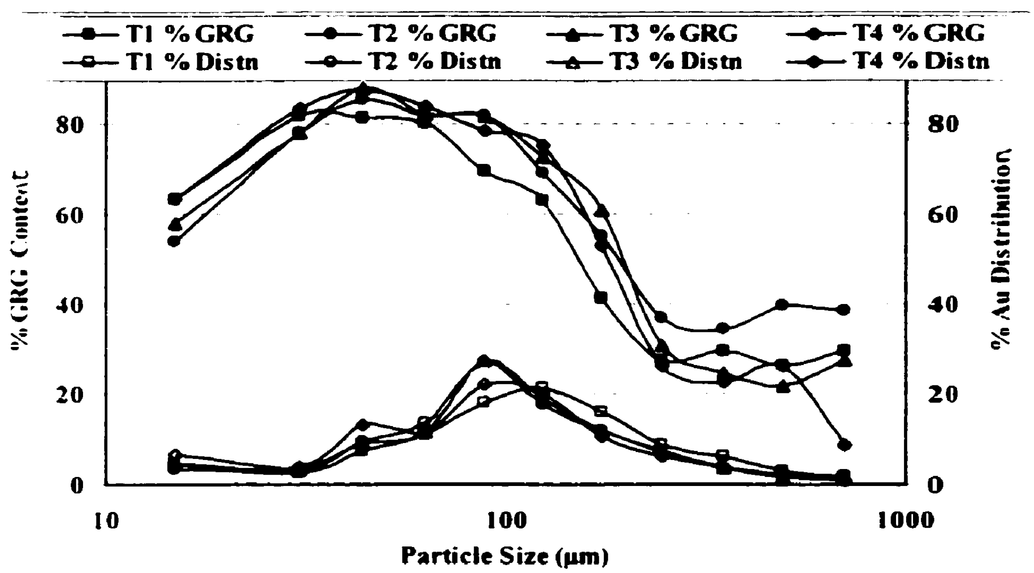


Figure 4.3 Size-by-size total gold distribution and GRG content of Snip jig feeds

Figure 4.3 shows the total gold distribution and the percent of GRG in the jig feed of the four tests. There was a relatively good agreement, with most gold around 75-105 µm, and a relatively low GRG content above 300 µm, highest below 100 µm. Gold distribution varied little from test to test, except that of the jig feed of T4, which was slightly fine.: 35% -75 µm rather than 25-29% for T1 to T3 (see Appendix B, pages 107-108). This was also in reasonable agreement with surveys PS1 and PS2. However, in both plant surveys, the highest gold fraction in the jig feed was in the -75 µm (>46%) while the +75-212 µm range accounted for less than 39% of the material, instead of more than 50% in T1 to T4.

Jig Tails

Jig tails samples were processed in the LKC at a pressure of 21 kPa (3 psi) and at a feed rate of 210 to 325 g/min. The results of the LKC tests are summarised in Table 4.4. The calculated head grades fluctuated between 1.99 and 4.06 oz/st and the GRG content, between 60 to 74%: both were very similar to the jig feed data. This suggested that very little GRG was recovered by the jig, a hypothesis that will be verified later on.

Figure 4.4 shows that the distribution of total gold was similar for all five tests and close to that of the four jig feeds. T4 yielded noisy data in the 105-150 and 150-210 µm classes. This was traced down to a broken 150 µm screen used to process the LKC tails. GRG content was similar to that of the feed, except above 300 µm, where it was slightly lower. GRG content dropped significantly from 100 to 300 µm.

Table 4.4 LKC metallurgical results of Snip jig tails

	T1	T2	T3	T4	T6
Feed grade, oz/st	1.99	4.06	4.03	2.43	2.53
GRG content, %	61.8	68.2	74.2	71.3	60.2
LKC conc. grade,	148.2	221.9	196.1	150.9	99.1
non GRG grade, oz/st	0.77	1.31	1.05	0.71	1.02

The averaged size-by-size GRG content for jig tails is shown in Figure 4.5, along with that of the jig feeds; there was no significant difference between the two curves except for the slight drop in the tails for the three coarsest size classes. Since the head grades of the feeds and the tails were so similar, the sampling of only those two streams was inadequate to determine the performance and the recovery of the jig. Fortunately, this had been anticipated in the test design, and jig concentrates had been sampled and their flow rate measured.

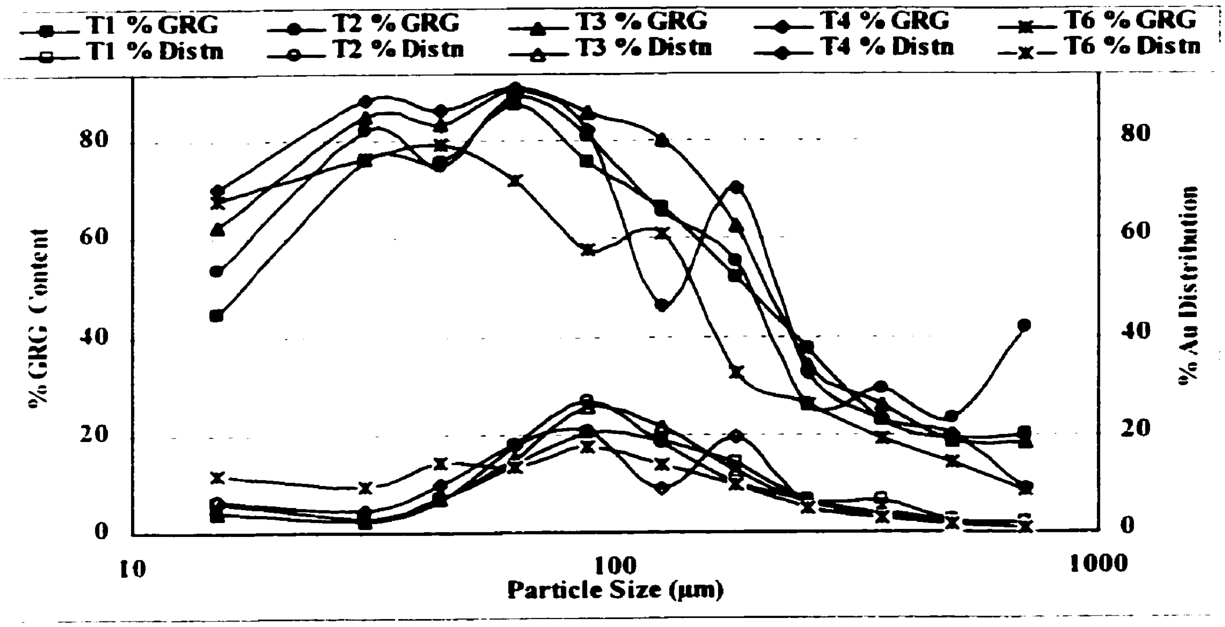


Figure 4.4 Size-by-size total gold distribution and GRG content of Snip jig tails

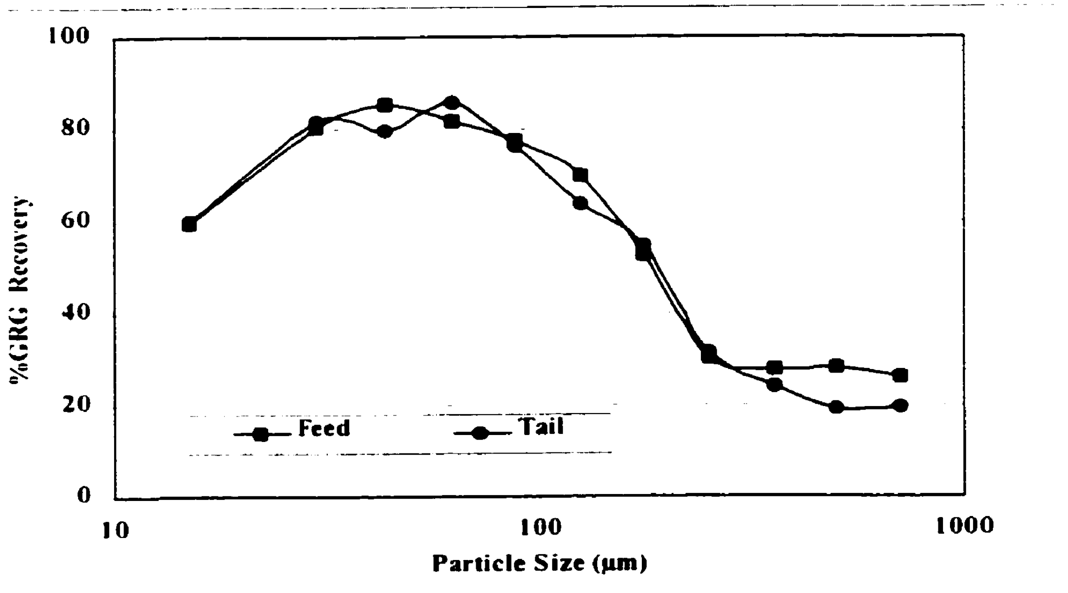


Figure 4.5 Average size-by-size GRG content of Snip jig feed and tails

Jig Concentrates

The jig concentrates were processed with the LKC at a pressure of 22-27 kPa (3.2-3.8 psi). and at a feed rate of 140 to 270 g/min. Table 4.5 shows the results: high recoveries (above 83%) were obtained for all samples except for hutch 1 T1 (73%): the only apparent reason for this was a low liberation. Note how the LKC tails were much lower in gold content than the LKC concentrates (see for example Appendix B page 110). The combined concentrate (T6) gave lower head and concentrate grades with a reasonable gold recovery (69.9%).

Table 4.5 LKC metallurgical results of Snip jig hutches

	T1 Hutch 1	T2 Hutch 1	T3 Hutch 1	T4 Hutch 1	T6 combined. Conc.
Feed grade. oz/st	10.9	16.2	33.7	18.8	5.84
GRG content. %	83.3	77.7	88.9	88.8	69.9
LKC conc. grade. oz/st	208	537	597	334	250
non-GRG grade. oz/st	1.90	3.70	3.92	2.21	1.79
	Hutch 2	Hutch 2	Hutch 2	Hutch 2	
Feed grade. oz/st	55.6	39.8	48.3	33.5	
GRG content. %	86.1	83.7	84.6	88.3	
LKC conc. grade. oz/st	493	683	760	370	
non-GRG grade. oz/st	8.79	6.82	7.86	4.26	

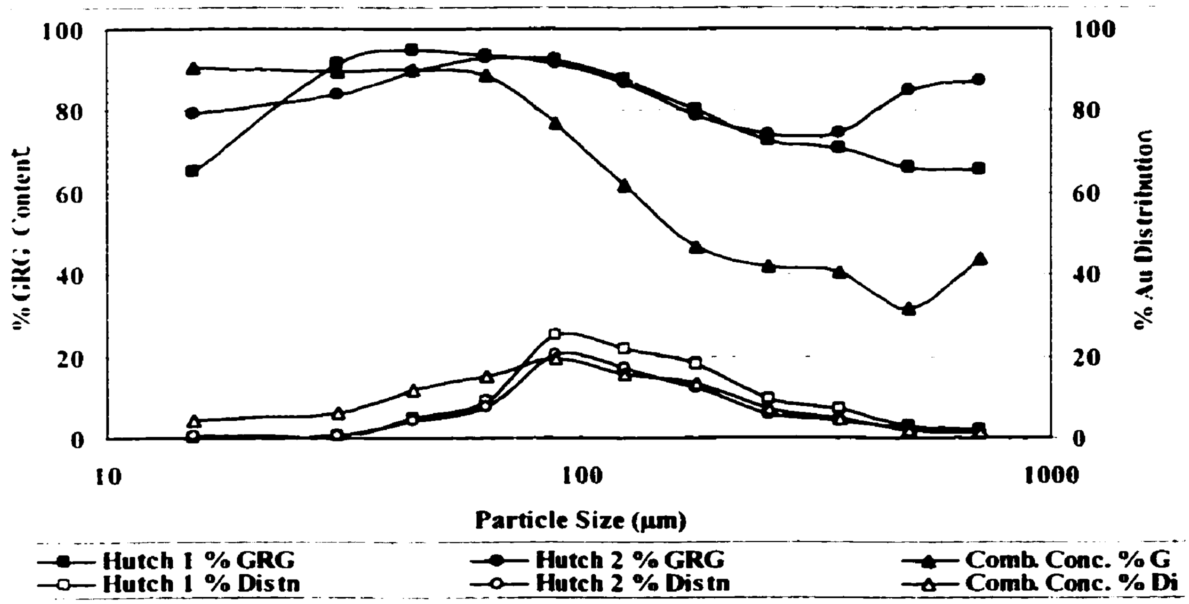


Figure 4.6 Average size-by-size total gold distribution and GRG content of Snip jig hutches

Figure 4.6 shows the average total gold distribution and GRG content of hutch 1 and hutch 2 (tests T1-T4) and the combined concentrate (T6). Again, the highest GRG content was located between 75 and 25 µm. Below 75 µm, there was much more gold than ore and the jig concentrates had a very high gold content. However, the 10 to 22% of the jig concentrates' gold in the -75 µm was much lower than what was achieved with the LKC on the jig feed, 20 to 32%. Coarse gold was also recovered by the first hutch (5 to 30% more of +212 µm), closest to the discharge trunnion.

Figure 4.6 also shows that the GRG content for T6 (comb. conc.) is lower than for T1-T4 above 105 µm and the gold distribution for T6 is higher than for T1-T4 below 75 µm. This reflected a profound change in operating practice, as concentrate rate was increased from 0.4 to 1.8 t/h, to recover more fine gold. The finer gold distribution confirmed that the strategy was effective; however, the jig then recovered coarse gold-bearing pyrite, which lowered the GRG content, especially above 105 µm.

The concentrates were also processed mixed with a 212 µm (70 mesh) silica in a 2:1 dilution ratio. The pressure ranged from 24 to 32 kPa (3.4-4.5 psi) and the feed rate from 220 to 325 g/min. Table 4.6 shows that the calculated head grade of each sample was in good agreement once the effect of silica dilution was taken into account. Dilution increased the GRG content in all cases, but only moderately, by 0.2 to 5.5%.

Table 4.6 LKC metallurgical results of the Snip hutches diluted with silica

	T1 Hutch 1	T2 Hutch 1	T3 Hutch 1	T4 Hutch 1
Feed grade, oz/st	10.9	17.0	30.3	17.7
GRG content, %	88.0	82.8	89.0	92.7
LKC conc. grade, oz/st	695	1041	858	673
non-GRG grade, oz/st	1.32	2.79	3.45	1.32
	Hutch 2	Hutch 2	Hutch 2	Hutch 2
Feed grade, oz/st	35.4	34.4	33.9	29.4
GRG content, %	91.6	86.6	87.4	89.2
LKC conc. grade, oz/st	648	1162	1140	804
non-GRG grade, oz/st	3.12	4.71	4.38	3.30

Figure 4.7 compares the GRG content of diluted and undiluted material for both concentrates. Dilution has increased the GRG content in the coarse end much more than the fine end. This could mean that the fine gold recovered by the jig can be easily recovered by the LKC and dilution of the feed achieves nothing: however, some coarse gold, probably very flaky or in part unliberated, is not recovered well with the LKC, unless the gangue density is lowered. Percolation of these particles is then improved, as is their recovery. It can be postulated that dilution has little impact because the jig recovers only highly gravity-recoverable gold, which the LKC can easily recover, even without dilution.

Mass balancing of the jig circuit, running at four different operating conditions, was performed with NORBAL2[28] using the measured rate of concentrate removal (Table 4.1) and the back-calculated head assays and weight percent from the LKC tests. The jig feed rate was estimated as 125 t/h of solids based on a circulating load of 400% (close to the value of the circulating load of PS1 and PS2 on page 127). Then the results were combined with the assays of the size fractions (+600 μm down to -25 μm) to complete the mass balance. Standard deviations were estimated from the accuracy of the samples and the assays. Results are found in Appendix C on pages 128-139.

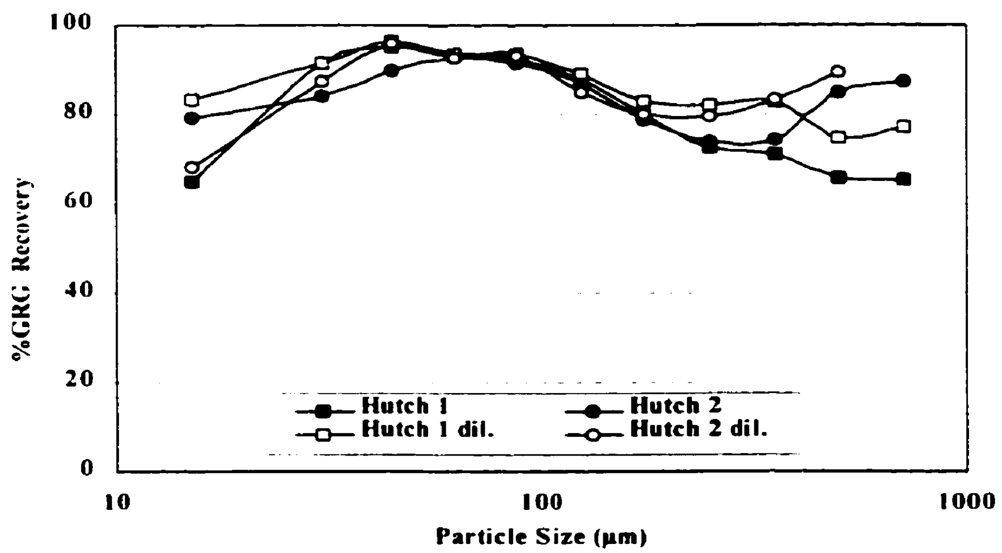


Figure 4.7 Size-by-size GRG content in the Snip hutches undiluted and diluted

Jig stage recoveries were calculated with the ratio of concentrate and feed. They are shown in Table 4.7 and are between 2.61% and 3.12%. Very little can be concluded from these recoveries. except that yields probably affected recovery as much as jig operating conditions (e.g. test T4. with the lowest yield. also exhibited the lowest recovery). The jig stage recovery for PS1and PS2 were calculated from their circuit mass balances (on page 127) and are 3.71% and 2.78%. respectively.

Table 4.7 Jig recovery based on gold to grinding circuit

	T1	T2	T3	T4	PS1	PS2
Ball mill feed. t/h	19.1*	19.1	19.1	19.1	22.5	20.3
Ball mill discharge/Jig feedrate. t/h	125.0	125.0	125.0	125.0	112.0	113.8
Jig yield rate. t/h	0.44	0.67	0.33	0.26	1.00	1.80
Jig feed grade. oz/st	2.15	4.17	4.23	2.48	4.27	2.39
Jig (combined) conc. grade. oz/st	16.1	22.9	42.7	24.4	17.9	4.18
Unit jig recovery. %	2.61	3.12	2.62	2.06	3.71	2.78
Jig circuit recovery. % (30 g/t)	43.6	94.5	86.7	39.1	93.5	43.6
Jig circuit recovery. % (45 g/t)	29.1	63.0	57.8	26.0	60.6	29.1

*: based on a mill feed rate of 459 t/day at the time of the sampling

Size-by-size total gold and GRG recoveries. as estimated with NORBAL2[28]. are shown in Figure 4.8a-d for T1 to T4. GRG recovery was calculated by multiplying total gold recovery with the ratio of GRG content of the concentrate and feed. All tests but T3 show similar results: recoveries for T3 are much lower. although overall recovery (Table 4.7) was not. Because the raw data was internally consistent (overall gold assays were calculated from the weighted size fraction assays). it is concluded that NORBAL2[28]. which does not seek to adjust overall and size-by-size assays in a congruent manner. caused this discrepancy. Of more importance than test-to-test variation are the general trends. which show a gold recovery relatively constant above 100 μm. but rapidly dropping below 100 μm. to a virtually zero recovery below 25 μm. This is consistent with known jig performance. Clearly there is a mismatch between the terminal velocity of finer gold and the average upward flow of hutch water. which is highly deleterious to fines recovery. No tweaking of jig operating conditions can atone for this fundamental problem.

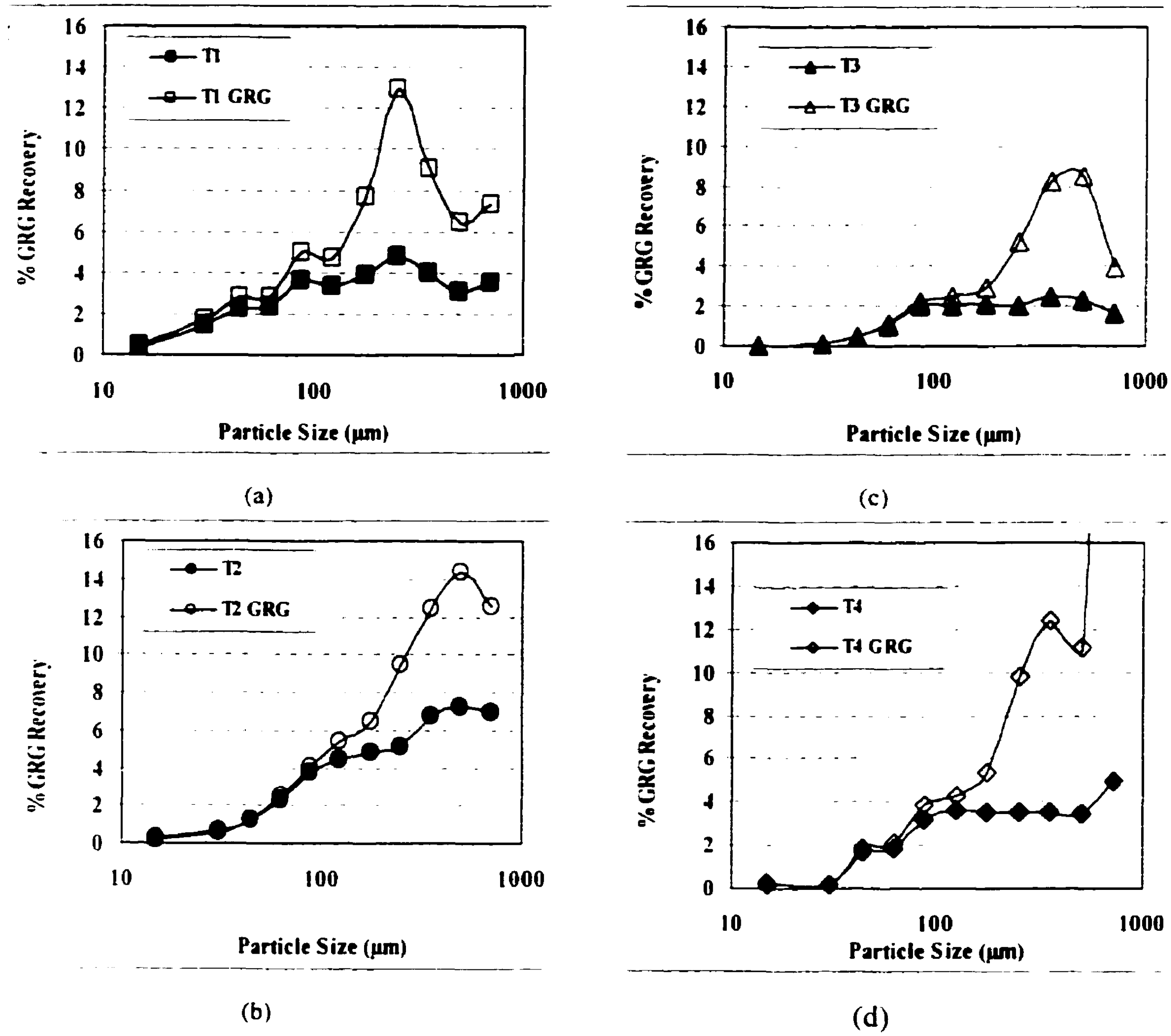


Figure 4.8 Size-by-size total gold and GRG recoveries for the jig

Figure 4.9 averages the size-by-size jig total gold and GRG recoveries for the four tests. Since the GRG content of the jig concentrate is was higher than the feed, GRG recovery was higher than that of total gold, especially in the coarse size classes. GRG recovery increased as particle size increased over the full size range. Total gold content recovery started to plateau at 105 µm because the coarser gold was increasingly unliberated. Figure 4.9 also shows that test PS2, with its much higher yield (concentrate withdrawing rate) achieved much higher recoveries below 100 µm than the average of T1 to T4. It is significant that finer gold should be recovered, as the GRG characterisation tests clearly showed that very little GRG reported to the +105 μm size fractions. Clearly the increased yield is recovering gold in the correct size range, where it is liberated. The twofold increase in recovery was obtained with a fourfold increased in yield – i.e. a significantly lower concentrate grade.

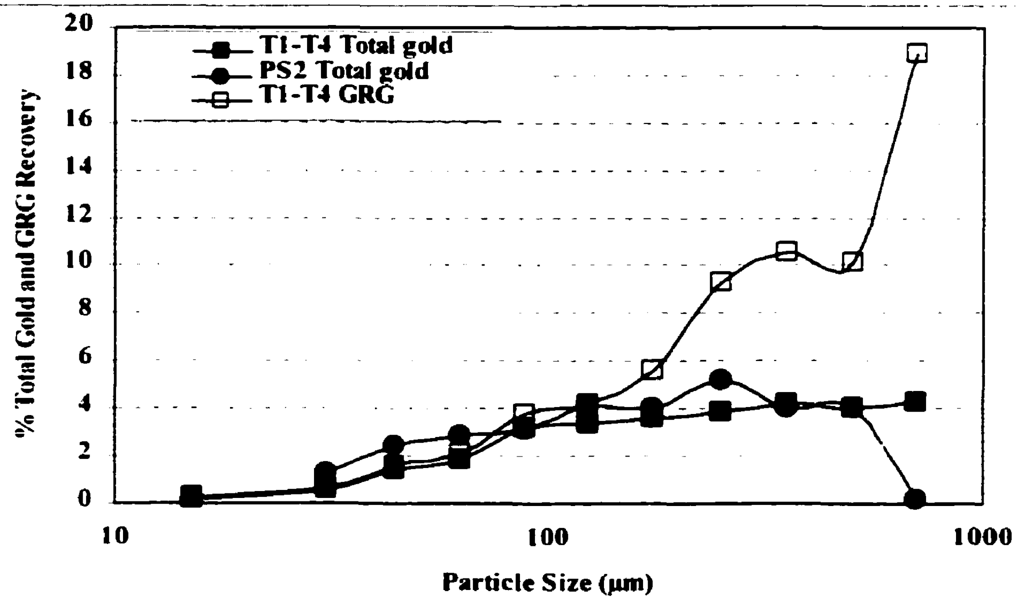


Figure 4.9 Average size-by-size total gold and GRG recoveries for the jig

Overall recovery can only be calculated if the ore grade and the ball mill feed rate are known (Table 4.7). Due to the high circulating load of gold, estimated overall recoveries are much higher than stage recoveries, 26 to 94%.

These estimated values should be corrected (i.e. lowered) to account for the losses of the shaking table, since the actual plant recovery (jig and table) was 24.9% in 1993 and 36.8% in 1994. The range reflected the uncertainty of translating stage recoveries in overall recoveries, when circulating loads were very high. Ideally, fresh feed, i.e. ball mill feed, and cyclone overflow should be extensively sampled cocurrently (to the jig tests), and total recovery based on the difference between the two, but this represented a major sampling, sample processing and assaying effort, especially for the fresh feed (on account of its coarseness).

Table Tails

At Snip, gold from the table is obtained by treating a combined jig concentrate of the two hutches that averages a head grade of 2.63 oz/st. It is much lower than other gold mines' table feed which is usually obtained from Knelson Concentrators.

The jig concentrate was processed on a Deister table. Davidson[30,31] measured table total gold recovery (Figure 4.11, see Appendix B on page 120). Overall recovery was 39%, and size-by-size recovery was very low above 100 μm, as the jig recovered mostly coarse pyrite with gold, which was then rejected, as it should be, by the table. The table also lost some fine gold: whether it could be recovered with a Knelson concentrator should be ascertained.

As part of the McGill Research effort on gold recovery by gravity, Huang[33] has investigated the recovery of gold from table tails using a LKC. He thought that recovery could be maximised by removing the coarse (+212 μm), generally lower grade fraction. Feeding 200 g of magnetite ahead of the actual sample could significantly increase concentrate grade. The magnetite would then be removed magnetically at the end of the test from the concentrate.

A 32.5 kg sample of table tails was pre-screened at 850 μm (20 mesh): the undersize was then screened at 212 μm (65 mesh). This yielded 13.1 kg of 212-850 μm and 19.3 kg of -212 μm. A 10 kg sample of the +212 μm fraction and 2 kg of the -212 μm fraction were then combined to create a coarse sample which was processed at a pressure of 22.4 kPa (3.2 psi) and a feed rate of 450 g/min. The rest of the fine material, 16 kg of -212 μm, was divided in two and was processed with or without magnetite pre-feed at a pressure between 23.4 and 25.9 kPa (3.4-3.7 psi) and at a feed rate of 340 g/min. For the magnetite test, the concentrate was then separated into magnetic and non-magnetic fractions with a hand magnet. Details are in Huang[33]. LKC results are shown in Table 4.8.

For the coarse sample, the results are disappointing: the head grade was found to be 2.65 oz/st, with a gold recovery of only 24.9%. Processing the finer fraction (-212 μm) yielded a 68% recovery into a 238 oz/st concentrate; with a magnetite pre-feed, both concentrate grade and recovery decreased slightly, to 216 oz/st and 60%, respectively. A hand magnet was then used to recover the magnetite, which contained only 3% of the gold, yielding a non-magnetic fraction assaying 1788 oz/st.

Table 4.8 LKC metallurgical results of Snip table tails

	Coarse sample	Fine (-212 μm) w/ magnetite	Fine (-212 μm) w/o magnetite
Feed grade, oz/st	2.65	5.97	5.97
GRG content, %	24.9	68.4	60.3
GRG grade, oz/st	72.4	237.6	1788
non-GRG grade, oz/st	2.01	1.92	2.28

Figure 4.10 shows the GRG content of the table tails, coarse and fine. The medium size range, 150-425 μm, contained the most gold but exhibited the lowest recoveries. This might be due to incomplete liberation: it was known that very little GRG (<15%) was coarser than 150 μm in the grinding feed and the jig did recover significant gold in this size range but this was essentially unliberated gold that followed pyrite. It was found in abundance in the table tails because the table rejected unliberated gold.

The calculated head grades for the table tails for PS1 and PS2 were 8.23 oz/st and 2.58 oz/st, respectively. Figure 4.10 also compares the total gold size distribution of the LKC tests with the total gold size distribution of the PS2 jig concentrate. Meanwhile, the table recovery curve shows that recovery really started below 150 μm. Whatever coarse gold, +150 μm, was in the jig concentrate, was lost in the table tails, because it was not liberated.

Figure 4.11 shows that the amount of GRG in the table concentrate can be determined by dividing the size-by-size total gold content in the concentrate (Davidson[30]) by the amount of size-by-size GRG in the table tails (coarse %GRG, Figure 4.10). The best recoveries (>60%) were in the size range 53-150 μm.

4.5 Conclusions

The Snip gravity circuit was repeatedly sampled and the amount of GRG in various streams measured. It was found that the Yuba-Richards jig failed to recover fine GRG effectively. As there was virtually no coarse gold in the Snip ore, the overall jig

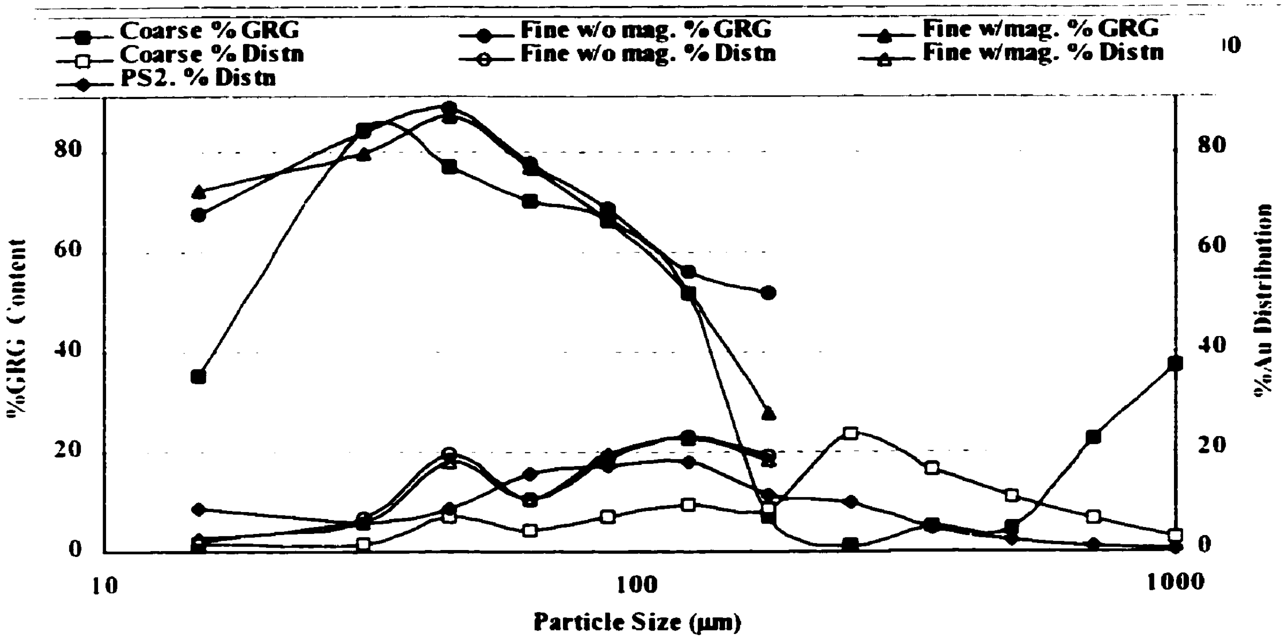


Figure 4.10 Size-by-size total gold and GRG content of Snip table tails

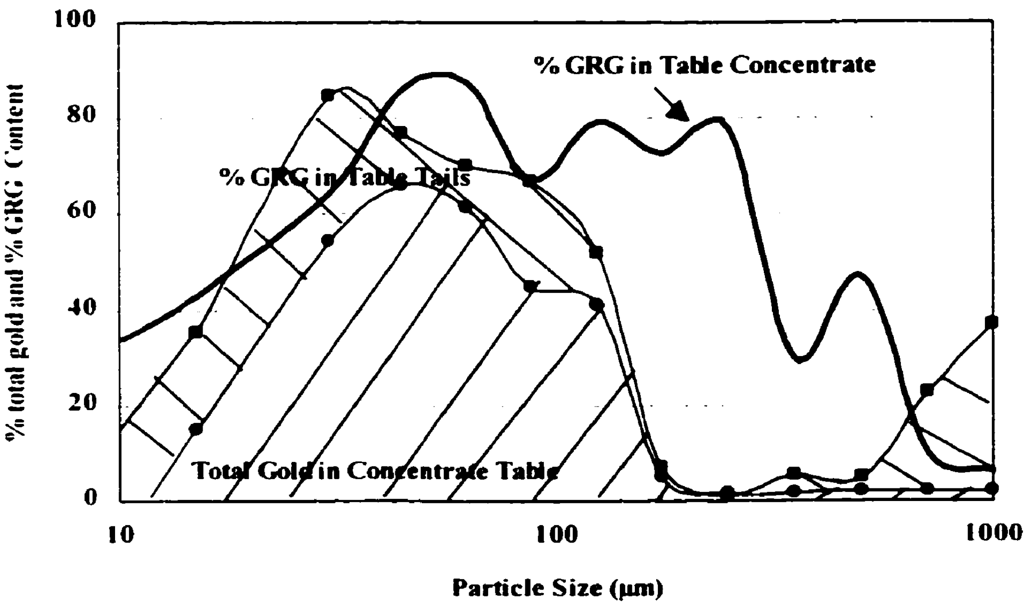


Figure 4.11 Total gold and GRG contents in the table concentrate and tails.

performance was low. Gold recovery was acceptable only because of the high circulating load of gold, 3300%-i.e. each GRG particle was fed to the jig an average 33 times before it was either recovered or ground/classified out of the grinding circuit. The best jig overall recovery (probably linked to particle size) was calculated to be 3.1% in a first series of four tests, but increased about twofold in a fifth test, when the concentrate withdrawal rate was increased fourfold. It reached maximum between 100 and 600 µm, but virtually all gold recovered by the jig above 150 µm was unliberated and rejected by the table. Jig recovery below 75 µm, where much GRG can be recovered, was poor. Virtually all gold finer than 25 µm was not recovered. As for the table, it rejected the coarse unliberated (non-GRG) gold, but it also failed to recover some fine GRG, which could be easily scavenged with a Knelson concentrator. The best option to maximise recovery will be determined in the next chapter, by simulation.

Chapter Five

Simulating Gold Recovery by Gravity

5.1 Introduction

The information presented in Chapters 3 and 4 will now be used to answer some of the questions raised in the first chapter. First, the existing Snip circuit will be simulated to validate the data generated in Chapter 4. An algorithm developed at McGill University, and based on GRG determination, will be used. The methodology makes use of a population balance model that includes gold liberation, breakage and classification behaviour, and applies recovery performance curves to gravity recoverable gold (GRG). Second, replacing the jig with a PKC (54 cm), or scavenging gold from the table tails with a smaller PKC (30 cm), will be simulated, using some of the industrial data presented and analysed in Chapter 3. Third, the relationship between the size distribution of GRG, the fraction of the circulating load treated (with the Knelson), and gold recovery, will be explored.

5.2 Circuit Simulation

5.2.1 Model theory

Consider a grinding circuit made of the block diagram shown in Figure 5.1. As fresh material is ground at the discharge of the mill, GRG is generated as a column matrix $\underline{f}$. Each f_i represents the amount of GRG in size class i in the ore, as determined in the GRG test. A pre-concentration unit (e.g. jig, sluice, PKC) concentrates a proportion p_i of the GRG in each size class (forming the diagonal matrix P) that is fed to the final gravity separator, which will produce the bullion concentrate. From each size class, a GRG recovery of r_i (forming the diagonal matrix R) is achieved. Material not recovered from all gravity units is then classified and a fraction c_i (forming diagonal matrix C) is returned to the mill. In the mill, a fraction h_{ii} of GRG in size class i remains in size class i in the mill discharge, but fractions h_{ji} (j=i+1 to n) report to finer size class i as GRG. Given the above description, it can be shown, with basic linear algebra, that

$$\underline{d} = PR^*[I - HC(I - PR)]^{-1*}\underline{f} \tag{5.1}$$

where $\underline{d}$ is a column matrix of the GRG content in the concentrate. Each d_i corresponds to the amount of gold (fractional, %, of grade) recovered in size class i. The sum of the d_i amounts to the total gold recovery[11].

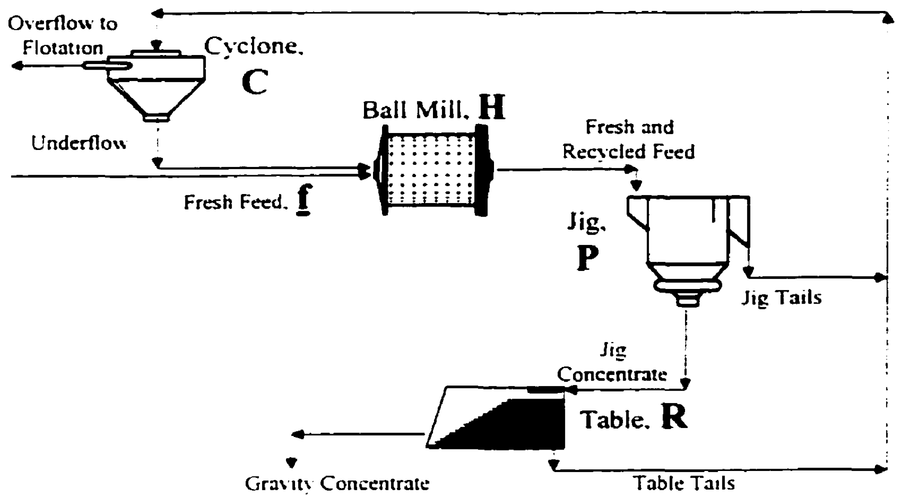


Figure 5.1 Representation of the circuit

5.2.2 Model parameters

Characterising GRG in the ore

The information generated by Woodcock[14] and Zhang[32], and presented in Figure 4.1, will be used. It is presented in Table 5.1. The procedure consists of a three-step recovery, steps 2 and 3 with the tails of the previous one at a finer grind. The first step is at 100% -850 µm, the second at 50-60% -75 µm and the third at 80%-75 µm. After each incremental size reduction, the sample is processed with a LKC to recover its GRG content. The rationale for this approach is to minimise overgrinding of GRG, which would occur by going directly to final size.

Gravity recovery

P and R are the diagonal matrices expressing the probability that GRG in size class i will first be screened or pre-concentrated in a main gravity separator (e.g. jig, PKC, sluice) and then recovered or upgraded by a second gravity unit (e.g. smaller PKC, table). Both are set when designing a gravity circuit by the selection and size of the concentration equipment. An interesting feature of the model is the ability to calculate the GRG circulating load when not using a gravity circuit by setting PR equal to 0, and this can be compared to actual circulating loads to verify at least part of the model[11].

Table 5.1 GRG content in the Snip Ore

Size (μm)	Snip 1	Snip 2
+840	0.0	0.0
+600	1.7	0.4
+420	0.5	0.4
+300	2.4	1.4
+212	4.8	2.6
+150	6.3	4.6
+105	5.0	6.4
+75	5.0	7.8
+53	4.7	8.3
+38	8.3	7.7
+25	7.8	6.3
-25	12.8	11.8
Total	59.4	57.8

Classification

Classification efficiency is characterised by the percent of each size class reporting to the underflow or coarse fraction of the cyclone. Its curve can be described with Plitt's model:

$$R_{u/f} = R_w = (100\% - R_w) \times [1 - e^{-0.693(\frac{d}{d_{50}})^m}] \quad \textbf{(5.2)}$$

where R_w is water recovery to the cyclone underflow, d_{50} the cut size of the classifier and m a measure of classification sharpness. The recovery makes up the diagonal of the classification matrix[11].

It will become apparent that C is probably the most significant data defining the circulating load of gold in a gravity-less grinding circuit, and 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 and is predominantly removed from the circuit either to the cyclone overflow or in the gold gravity concentrate.

Grinding

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, $\underline{m}_f$, with H as the breakage matrix of the material. The model can be resolved into a simple equation:

$$\underline{M}_d = \begin{bmatrix} h_{11} & 0 & 0 & \cdots & 0 \\ h_{21} & h_{22} & 0 & \cdots & 0 \\ h_{31} & h_{32} & h_{33} & \cdots & 0 \\ \cdots & \cdots & \cdots & \cdots & 0 \\ h_{n1} & h_{n2} & h_{n3} & \cdots & h_{nn} \end{bmatrix} \times \underline{M}_f \tag{5.3}$$

Since material can only exit the mill in the same or a finer size class than the one it entered in, h_{ji} is zero for $j<i$, the upper triangle of the matrix is null. For total gold, the sum of each column should be equal to 1 due to mass conservation; unlike the traditional grinding model, the "pan", the finest size class, is included since GRG in the pan can be recovered[11,15].

The grinding matrix, H, is computed with the breakage (B) and the selection (S) functions of the material in a ball mill simulation, using two programs called BALLDATA and BALLMILL, two BASIC programs created at McGill.

The procedure to estimate the breakage function of GRG is explained in detail in Banisi[15] and in Noaparast[34]. Losses from GRG becoming non-GRG due to smearing, overgrinding or excessive flaking, are taken into account and the columns of the H matrix for GRG sum up to less than unity.

The selection function used is based on the work of Banisi[15], who measured for gold a selection function six times smaller than the ore's at 50-100 μm and twenty times smaller at 500-1000 μm because its malleability lowered its grinding rate. The selection function of the ore at Snip was first estimated from a mass balance of the circuit. The selection function of GRG was then estimated from that of the ore, using the ratios of Banisi[15].

The classical population model was then used to calculate the lower diagonal matrix of Eq. 5.3. for GRG.

5.3 Case study: Snip Operation

At Snip, the gravity circuit consists a jig whose is fed the entire discharge of a first ball mill which is in closed circuit with one cyclone.

The matrix P is the size-by-size jig GRG recovery determined in Chapter 4. Results are the average of the four tests. Table 5.2 shows recovery whereas Figure 5.2 shows the GRG content, distribution and grade versus particle size. In Figure 5.2, jig feed grade increased dramatically with decreasing particle size. Both increases created a synergy that pointed at the importance of recovering fine GRG, which was far more abundant than coarse GRG.

R is the size-by-size GRG table recovery determined in Chapter 4 and is shown in Table 5.3. The matrix C is shown in Table 5.4 and Figure 4.2. The last term of the diagonal is the fractional recovery of the -25 μm (500 mesh), 0.916 (91.6%). Recovery climbed to 97.2% for the 25-37 μm, and was greater than 99% for other coarser size classes. For all size classes above 300 μm, recovery was set to 99.9% rather than 100%, to account for very occasional short-circuiting of GRG to the cyclone overflow. The matrix showed excellent potential for gravity recovery, as gold was massively recycled to the cyclone underflow. This indicated that GRG would exit the grinding circuit either as gravity concentrate or as non-GRG below 25 μm.

The resulting H is shown in Table 5.5. The residence time distribution used to calculate H was that of two small perfect mixers (PM, 20% of the total mean residence time, τ) in series with one large PM (70% of τ) and one plug flow unit (10% of τ).

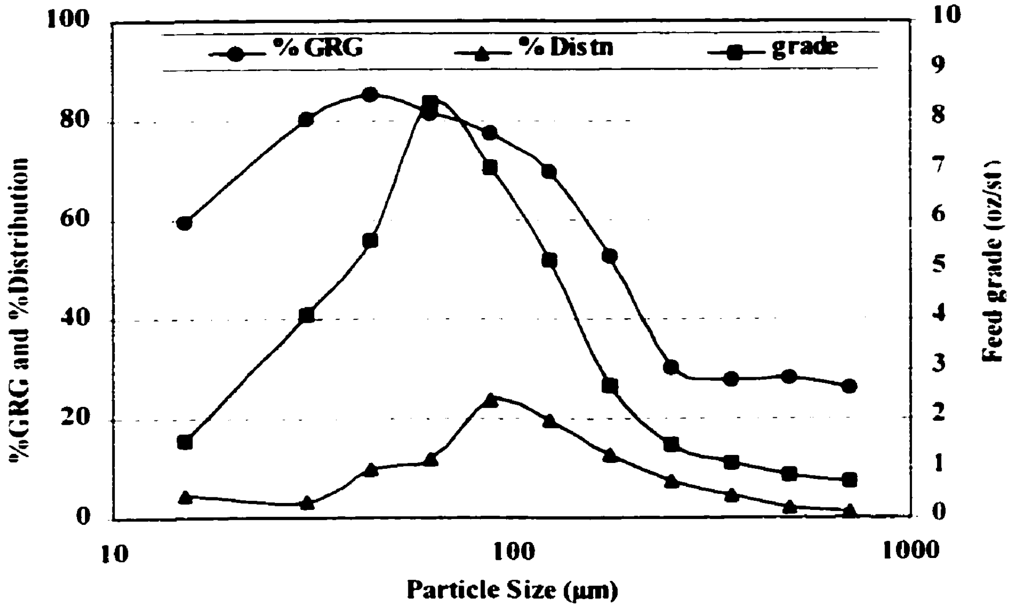


Figure 5.2 The size by-size grade. the gold distribution and %GRG content in the Snip 2 jig feed

Table 5.2 Jig recovery matrix

Size (µm)												
+840	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+600	0.0	0.189	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+420	0.0	0.0	0.101	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+300	0.0	0.0	0.0	0.105	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+212	0.0	0.0	0.0	0.0	0.093	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+150	0.0	0.0	0.0	0.0	0.0	0.056	0.0	0.0	0.0	0.0	0.0	0.0
+105	0.0	0.0	0.0	0.0	0.0	0.0	0.042	0.0	0.0	0.0	0.0	0.0
+75	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.038	0.0	0.0	0.0	0.0
+53	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.021	0.0	0.0	0.0
+38	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.016	0.0	0.0
+25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.007	0.0
-25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.003

Table 5.3 Table recovery matrix

Size (μm)												
+840	0.098	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+600	0.0	0.469	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+420	0.0	0.0	0.298	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+300	0.0	0.0	0.0	0.779	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+212	0.0	0.0	0.0	0.0	0.728	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+150	0.0	0.0	0.0	0.0	0.0	0.722	0.0	0.0	0.0	0.0	0.0	0.0
+105	0.0	0.0	0.0	0.0	0.0	0.0	0.792	0.0	0.0	0.0	0.0	0.0
+75	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.672	0.0	0.0	0.0	0.0
+53	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.876	0.0	0.0	0.0
+38	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.860	0.0	0.0
+25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.645	0.0
-25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.426

Table 5.4 Classification matrix

Size (μm)												
+840	0.999	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+600	0.0	0.999	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+420	0.0	0.0	0.999	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+300	0.0	0.0	0.0	0.999	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+212	0.0	0.0	0.0	0.0	0.997	0.0	0.0	0.0	0.0	0.0	0.0	0.0
+150	0.0	0.0	0.0	0.0	0.0	0.999	0.0	0.0	0.0	0.0	0.0	0.0
+105	0.0	0.0	0.0	0.0	0.0	0.0	0.998	0.0	0.0	0.0	0.0	0.0
+75	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.998	0.0	0.0	0.0	0.0
+53	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.996	0.0	0.0	0.0
+38	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.996	0.0	0.0
+25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.972	0.0
-25	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.916

Table 5.5 Grinding matrix

Size (μm)												
+840	0.936	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
+600	0.042	0.945	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
+420	0.005	0.036	0.954	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
+300	0.003	0.004	0.031	0.960	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
+212	0.002	0.003	0.004	0.027	0.966	0.000	0.000	0.000	0.000	0.000	0.000	0.000
+150	0.002	0.002	0.002	0.003	0.023	0.971	0.000	0.000	0.000	0.000	0.000	0.000
+105	0.001	0.002	0.002	0.002	0.002	0.020	0.975	0.000	0.000	0.000	0.000	0.000
+75	0.001	0.001	0.001	0.001	0.002	0.002	0.017	0.979	0.000	0.000	0.000	0.000
+53	0.001	0.001	0.001	0.001	0.001	0.001	0.002	0.014	0.972	0.000	0.000	0.000
+38	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.012	0.975	0.000	0.000
+25	0.000	0.000	0.000	0.000	0.000	0.001	0.001	0.001	0.001	0.010	0.980	0.000
-25	0.001	0.001	0.001	0.001	0.002	0.002	0.001	0.002	0.002	0.002	0.007	0.980

Validation of the existing circuit

To validate the model. GRG recovery that can be predicted with the jig and table data (of 1993 and 1994) was first determined. For all simulations. the data of the second GRG test (57% GRG) was used. The first simulation yielded a recovery of 26% for GRG. which corresponds to plant performance in 1993 before the jig yield was increased to 1-2 t/h. To simulate the 1994-95 recovery data. jig recovery in each size class was doubled from the 1993 data (as observed in Chapter 4). a reasonable assumption. since jig yield had quadrupled (from 0.4 to 1.8 t/h). This yielded a total recovery of 31.5%. in good agreement with the 1994 recovery of 36.8%. The difference between the two can be explained by the fluctuations in GRG content and the coarser gold recovery not accounted for in the simulation (i.e. not present in the f matrix). Table 5.6 presents the results of the simulations compared with the recoveries measured in 1993 and 1994-1995.

Table 5.6 Total gold and GRG recoveries measured and simulated

	1993	1994-1995
Total gold recovery measured. %	24.9	36.8
Total GRG recovery simulated. %	25.4	31.5

Simulation of Knelson concentrators

In a first simulation. scavenging of the table tails with a 12" PKC was modelled by setting R (table and Knelson). i.e. total GRG recovery. to 95% for each size class. The recovery of the jig was used as P. It yielded a recovery of 33.3%. an increase of almost 1.8%.

In a second simulation. the jig was replaced by a 20" PKC treating only part of the circulating load. a bleed of cyclone underflow of 8 to 12 t/h. This mimics the case of Placer Dome South Porcupine Mine where a 30" PKC replaced four jigs with increased recovery while treating only 13% of the circulating load. Knelson recoveries measured at Meston (T1) at 25-30 t/h on a 30" PKC were used as P but the recoveries were lowered down of about 20% to take into account the size difference of the device (30" to 20"). Snip table recovery of GRG was used as R (Table 5.3). So PR in Eq. 5.1 is multiplied by 0.13. GRG recovery then increased from 31.5% to 39.1%.

In a third simulation, the jig was replaced by a 20" Knelson like in the second simulation (0.13P) and table tails were scavenged by a smaller Knelson (R_t = 0.95). GRG recovery yielded 42.8%. Results of all simulations are shown in Table 5.7.

Table 5.7 Total GRG recovery by simulating with Knelson Concentrators

	1994-95 simulated	Scavenging tails (95%)	Replacing jig by a PKC	Scavenging & Replacing
Total GRG recovery, %	31.5	33.3	39.1	42.8

5.4 Knelson vs. Jigs

The Snip comparison of Knelson versus jigs needs to be generalised. Assuming constant jig and Knelson performances, this will now be done with two GRG distributions (fine and coarse) at various fraction of the circulating load treated by the Knelson.

The rationale for testing a different GRG distribution is that the one at Snip is extremely fine, which will favour the Knelson, capable of superior finer GRG recovery than the jig. Varying the fraction of the circulating load treated at constant performance is a measure of the gravity recovery effort, which will vary depending on its economic importance.

The finer and coarser GRG distributions were generated by lowering two sizes down, or increasing two sizes up, each size class of the matrix f, the GRG content in the jig feed Snip 2 from Table 5.1 (Table 5.8). The fraction of the CL treated was varied between 6 and 25%. It could be argued that even more could be treated, as is the case in some plants. However, this is always achieved by feeding more to the Knelson, with a corresponding decrease in stage recovery. For simulation purposes, this is very similar to using a maximum bleed of 25% at high Knelson performance.

Table 5.9 compares the performance of jigs to that of the various Knelson options. A first observation is that even at the lowest fraction of the circulating load treated (6%), the Knelson yielded a higher recovery than the jig for both GRG distributions. Second, the difference in the performance between Knelson and jig increases dramatically as the

GRG size distribution becomes finer. 25.3% vs. 15% for the fine GRG compared to 43.9% vs. 41.2% for the coarse GRG.

Table 5.8 Fine and coarse GRG size distributions derived from Snip 2

Size (μm)	Coarse	Fine
+840	0.8	0.0
+600	1.4	0.0
+420	2.6	0.4
+300	4.6	0.4
+212	6.4	1.4
+150	7.8	2.6
+105	8.3	4.6
+75	7.7	6.4
+53	6.3	7.8
+38	11.8	8.3
+25	0.0	7.7
-25	0.0	18.1
Total	57.8	57.8

Table 5.9 Gold Recovery. %. as a function of the GRG size distribution and fraction of the circulating load treated by Knelson.

	Knelson recovery, %			Jig Recovery, %
	6% of CL	13% of CL	25% of CL	100% of CL
Coarse GRG	43.9	51.0	54.4	41.2
Fine GRG	25.3	34.4	41.8	15.0

The simulations show that not all the circulating load needs to be treated just to be able to obtain a reasonable GRG recovery. The Knelson should be capable to recover most of the coarse gold in the first pass. Whatever gold is not recovered builds up in the circuit. is ground finer but still manage to be recovered by the unit. However. with the jig. any gold not recovered in the first pass has less and less chance to be recovered afterwards when it becomes finer and finer. hence the reason why all the load is processed.

The simulations also tell which are the strong and weak points of the Knelson. The Knelson performs better with a coarse size fraction. with less than 25% of the circulating load. However. if the size distribution of the feed becomes finer for any reason. increasing the fraction of the circulating load treated can control losses.

At Snip, the actual GRG size distribution is between the fine and coarse size distributions in Table 5.8. Simulation has shown that by replacing the jig with a 20" PKC, scavenging the table tails with a 12" PKC and treating only 25% of the circulating load, GRG recovery should improved from 31.5% up to a recovery between 42% and 54%.

Chapter Six
Conclusions

6.1 Introduction

Although gold gravity separation cannot compete with flotation or cyanidation. it still plays an important role in many recovery flowsheets. as an inexpensive method to recover much of the mill production at a low coast early in the flowsheet. Modest increases in gold recovery. and in some cases significant increases in the return of the gold recovered. combined with savings in milling costs to justify simple and inexpensive gravity circuits. Because Knelson-based circuits can provide both the design and operating simplicity. it has become the standard choice over the past 15 years. replacing jigs in much of North America and Australia. This was the focus of this thesis. which looked at industrial performance for one jig and several Knelson circuits.

Circuit streams were sampled and each product processed with a laboratory Knelson Concentrator with precision and accuracy using a methodology that was designed at McGill University to minimise "nugget" effects. Data thus generated will be used added to the existing database. to assist in the design and optimisation of gravity circuits. using a simulator for the behaviour and recovery of Gravity Recoverable Gold in grinding and gravity circuits. This work has shown the importance of the data bank. as Knelson performance was found to be very significantly from plant to plant.

6.2 Test Work Results

Meston

Meston Lake Resources was one of the first plants to have its Knelson Concentrators' performance analysed. Three sluices were feeding two Knelsons at 30 t/h alternately. Previous work had shown that 68% of the gold in the ore was gravity recoverable. out of which 40% escaped the gravity circuit via the cyclone overflow. More test work showed that the circulating load of the gold was low. indicating that the gravity circuit was removing GRG quite efficiently. Sampling the feed and tails of one Knelson at three different feed rates (20, 30 and 40 t/h) running alternately and at the same feed rate running simultaneously (30 t/h) was proceed. At a feed rate of 30 t/h, total gold stage recovery was found to be 42% while GRG stage recovery was 75%. Increasing the feed rate to 40 t/h lowered total gold recovery of 26% and GRG recovery to 60%. When fed simultaneously at 30 t/h, the total gold stage recovery of the two KCs was 42% while GRG recovery dropped to 65%. Test work showed that 75% of the gold in the Knelson feed was finer than 150 μm and that the GRG content increased with decreasing particle size. The plant KCs proved capable of recovering GRG over the full size range, although a slight drop in efficiency was measured below 37 μm.

Est-Malartic

At Est Malartic, one 76 cm PKC processed a high gangue density feed. The original Knelson feed contained 57% GRG. Splitting the Knelson feed into two fractions (+ and -300 μm) yielded informative results: the -300 μm feed contained 65% GRG, and less than 10% of the total gold reported to the +300 μm. Test work also showed that the amount of GRG dropped from 73% in the PKC feed to 34% in its tails: the difference between the two products being a measure of significant recovery. On the other hand, the -300 μm fraction had very similar GRG contents for the Knelson feed and tails, 65% vs. 60%, an indication of much lower recovery. The difference also decreased with particle size, which confirmed that Knelson performance was not only lower than at Meston, but also much more size dependent. GRG accumulated in the circulating load below 150 μm, where Knelson performance was the poorest.

Although the difference in GRG content of the Knelson tails and feed is a useful measure of the effectiveness of the device; it is inadequate to determine size-by-size GRG recoveries when performance is as low as that of Est Malartic, because of the similarities between their GRG content. A sample of the concentrate and a measure of yield (i.e. concentrate weight recovery) are also necessary.

Hemlo

At Hemlo Golden Giant Mines, the PKC was installed at the discharge of the PCU, replacing a jig. With a known gold circulating load of 6000% and 45% of the gold coarser than 100 µm, the Knelson Concentrator achieved a 35-40% gold recovery (of the gold in the feed of the grinding circuit). After one year of operation, the PKC performance was analysed on two different occasions. In the first test, total gold and GRG recoveries were 11% and 19% respectively. The low plant recovery can be explained by the high feed rate to the PKC, 70-80 t/h. In the second test, total gold and GRG recoveries were found to be 20% and 24% respectively. The slightly higher recovery of the second test was attributed to a shift to a coarser GRG distribution in the Knelson feed, a shift that coincides with a significant increasing GRG recovery. Despite using large sample weights (i.e. 75 kg rather than, 20 kg for the PKC), size-by-size data showed variability, confirming again that a concentrate sample is needed to determine more reliable size-by-size GRG recoveries, when stage recovery is significantly below 50%

Aurbel

At the Aurbel mill, two different ores were processed together by two 51 cm PKCs. At a feed rate of 3 t/h, recoveries of 53% for total gold and 92% for GRG were achieved. As the feed rate was increased to 5 t/h, total gold and GRG recoveries dropped to 30% and 71%, respectively. However, gold production, which could be hurt by the 20% drop in the recovery, actually went up on account of the increase of 60% in feed rate.

Agnico-Eagle

At Agnico-Eagle's Laronde, high gangue density samples were processed to refine the methodology used to measure GRG and study gold's classification behaviour. Previous test work had determined a gold circulating load of 3700%. The grinding circuit was again analysed to determine how much of the gold building up in the circuit was gravity recoverable. A portion of the samples was treated as is, another without any +300 μm material and a third diluted with silica. It was found that 70-72% of the gold in the cyclone feed (as is and screened) was gravity recoverable. as was 74% of the gold in the -300 μm cyclone underflow is gravity recoverable. but 80% when using silica dilution. The cyclone overflow contains 23% GRG.

Mass balancing the -300 μm fraction showed that the circulating load of the ore was 533%. and that of GRG 3784%. The circulating load would be slightly lower for the full size distribution. as the +300 μm contains less GRG. The behaviour of GRG in the grinding circuit was such that 99.4% of it reported to the cyclone underflow and accumulated in the circulating load. compared to 98.1% for total gold and 84.2% for the ore.

<u>Snip</u>

Snip is one of the few Canadian plants still using a jig for gold recovery. An evaluation of the jig and classification circuits was done in order to estimate how much GRG was actually recovered by the unit. and assess if circuit performance could be improved. The GRG circulating load was estimated at 3300% with a relative abundance of fine gold (<105 μm). too fine for efficient recovery with a jig. The behaviour of GRG was such that 98.6% of it reported to the cyclone underflow. compared to 97.4% for total gold and 87% for the ore itself. The average jig stage recovery. determined by the LKC. was found to vary between 2.1 to 3.1% depending on the operating conditions and particle size. Size-by-size jig recoveries increased from 6% to 18% for particle size coarser than 150 μm. however jig performance decreased with decreasing particle size: almost no gold (<1%) finer than 25 μm was recovered. The table rejected almost all gold recovered between 100 and 600 μm by the jig. as it was not liberated.

Jig stage recovery. determined at the mill. was found to be 3.71% in 1993 and 2.78% in 1994. From 1992 to 1994. the jig yield was increased from 0.4 t/h to 1.8 t/h in an attempt to improve total gold recovery. the overall plant recovery went from 24.9% to 36.8%.

6.3 Simulating Gold Recovery

The evaluation of one jig and four Knelson circuits generated data that were used in the simulation of a modelled gravity circuit that includes gold liberation. breakage and classification behaviour. and GRG recovery performance curves. The replacement of the Snip jig by a 54-cm plant Knelson Concentrator with or without the scavenging of the table tails by a 30-cm PKC was simulated. Simulating the existing Snip Circuit first validated the circuit simulator. A 25.4% GRG simulated recovery was achieved. corresponding to plant performance in 1993. 24.9%. In 1994. the jig yield was increased to 2 t/h and plant recovery increased to 36.8%. A simulated 31.5% gold recovery was achieved by doubling size-by-size jig recovery. to account for the fourfold increase in jig yield. The model fitted well and the slight difference in the actual and simulated recoveries can act as a safety factor. The model is suited for taking into account the fact that plant recovery units (even Knelsons) are not as efficient. or efficiently operated. as the LKC is to determine GRG.

Three simulations were done:

(1) using the jig but scavenging the table tails with a small KC:

(2) replacing the jig by a KC:

(3) replacing the jig with the KC and scavenging the table tails

and the simulated plant recoveries were 33.3%. 39.1% and 42.8%. respectively.

The relationship between the size distribution of GRG. the fraction of the circulating load treated and the gold recovery was analysed: the feed size distribution was made coarse and fine to assess how jig and Knelson performances would be affected. The jig could perform more efficiently with a coarse feed and achieved 41% recovery. Nevertheless. even then the Knelson out performed the jig. even with the lowest fraction of the circulating load treated. 6%. With the finer GRG size distribution. Knelson performance decreased. but nearly as much as that of the jig. It was concluded that unless a Knelson is grossly under-designed for a given application, it would always out-perform a jig. The difference in performance, however, is not as large as one would intuitively assume, because gold not recovered in one pass has such a high probability of being recycled to the unit. As a result, most plant personnel where jigs were changed to Knelsons favoured the change for mechanical and operating reasons, rather than metallurgical performance.

6.4 Recommendations

The efficiency of the existing Snip circuit is clearly limited by its main equipment, the jig. As there is virtually no coarse gold in the feed and there is a significant amount of gold too fine to be recovered by the unit, a remedy would be to replace the jig by a Knelson which would recover a wider size distribution of gold. If the change is not feasible, improvements can be made elsewhere. First, by quadrupling the feed rate from 0.4 t/h to 1.8 t/h, the plant performance went from 24.9% to 36.8%. The optimum yield should be determined. Second, operating conditions could be closely monitored so that fine gold is not carried away to the tailings. Third, using a smaller Knelson for secondary upgrading, such as a scavenger which would recover much of the gold that is returning to the grinding circuit from the table tails. Fourth, a screen could be used to separate the coarse pyrite/gold particles from the fine free gold. The oversize could still be processed by the jig for coarse gold removal while the undersize could be directed to a small KC or be sent directly to the table.

6.5 Future Work

This study has yielded very informative data on the operation of a grinding and gravity circuit, but it also has identified areas where further work would be beneficial.

When sampling a gravity circuit, feed, tails and concentrate should all three be sampled, so that more accurate size-by-size GRG recoveries can be determined. Further, if stage recovery is low (which is the correct way to operate a gravity unit for gold recovery from a grinding circuit circulating load) a measure of yield must also be made. to estimate recovery adequately.

Other types of separators should be evaluated to generate data for the model.

Other circuits still using jigs should be investigated in order to have more accurate data on a jig performance.

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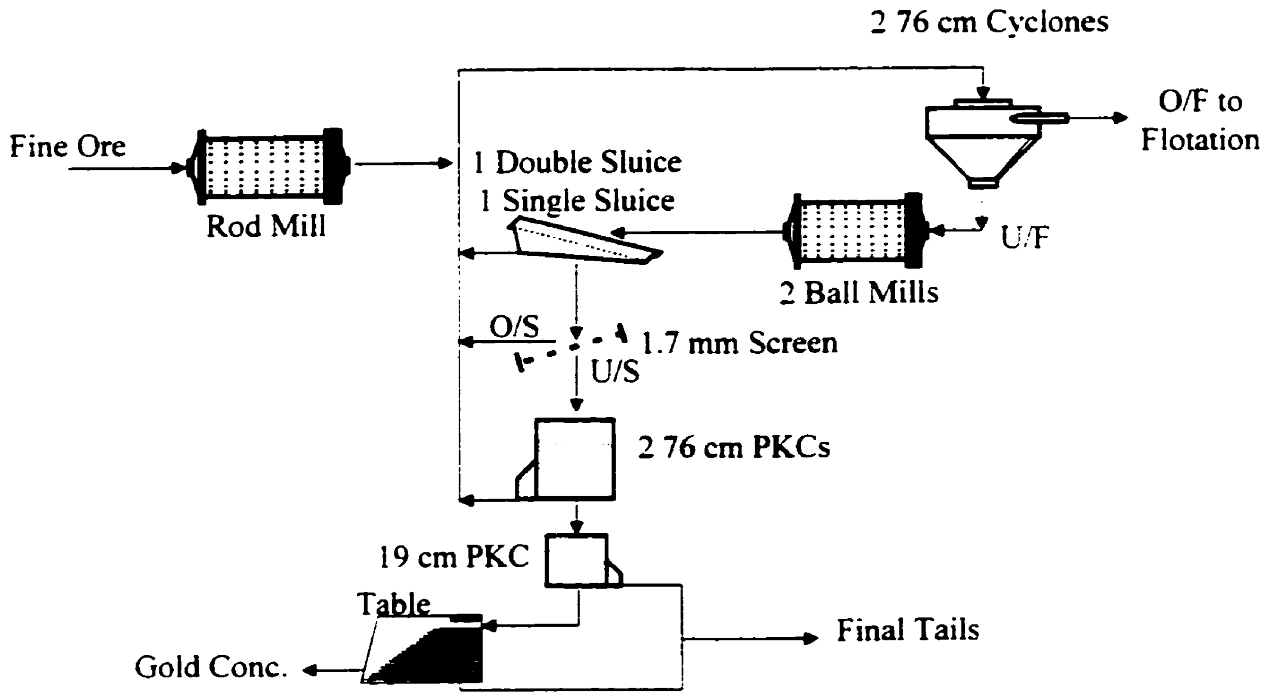
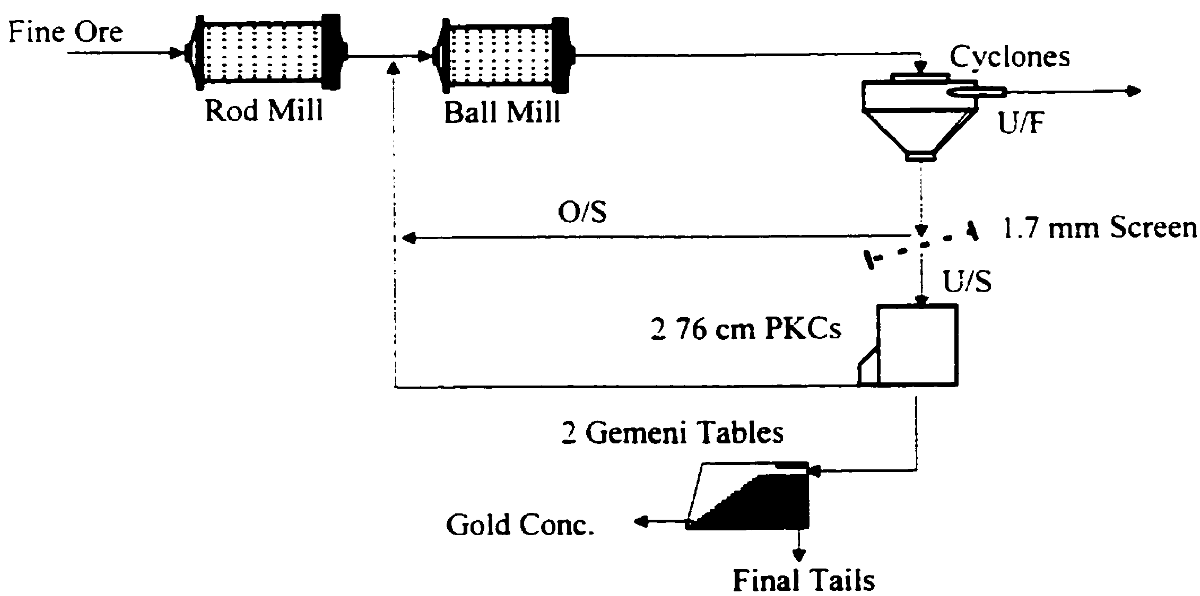
APPENDIX A

GRINDING AND GRAVITY FLOWSHEETS OF GOLD PLANTS

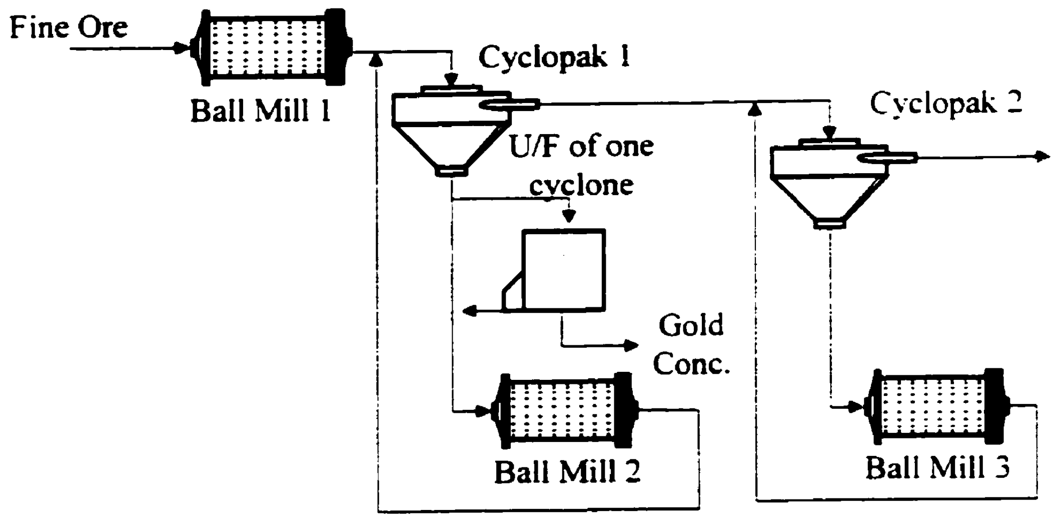
Flowsheet 1 Meston Lake Resources Mill



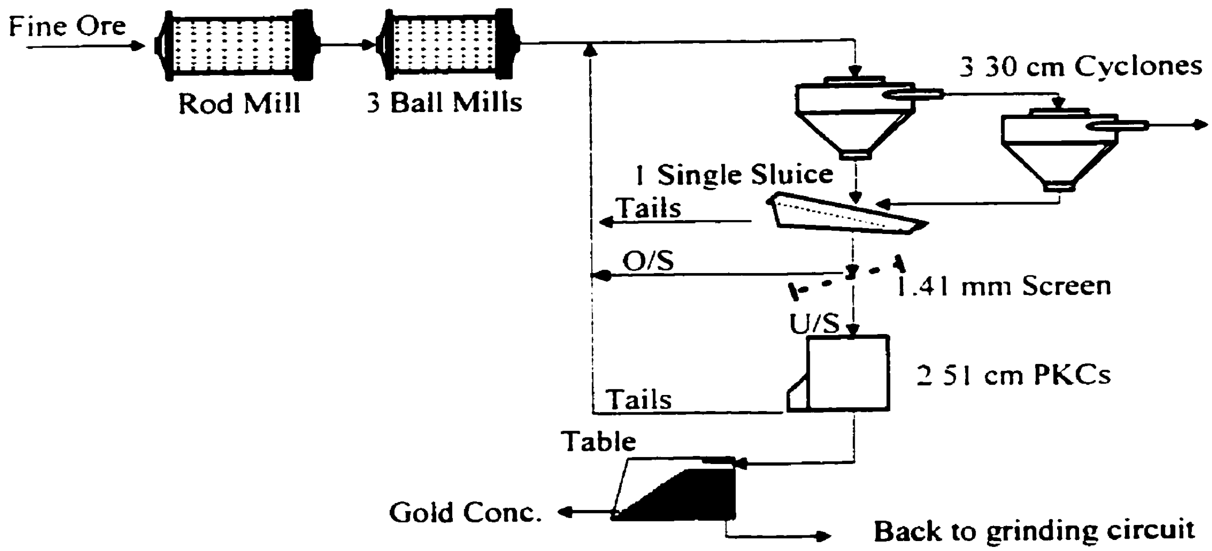
Flowsheet 2 Barrick Gold - Est Malartic Division Mill



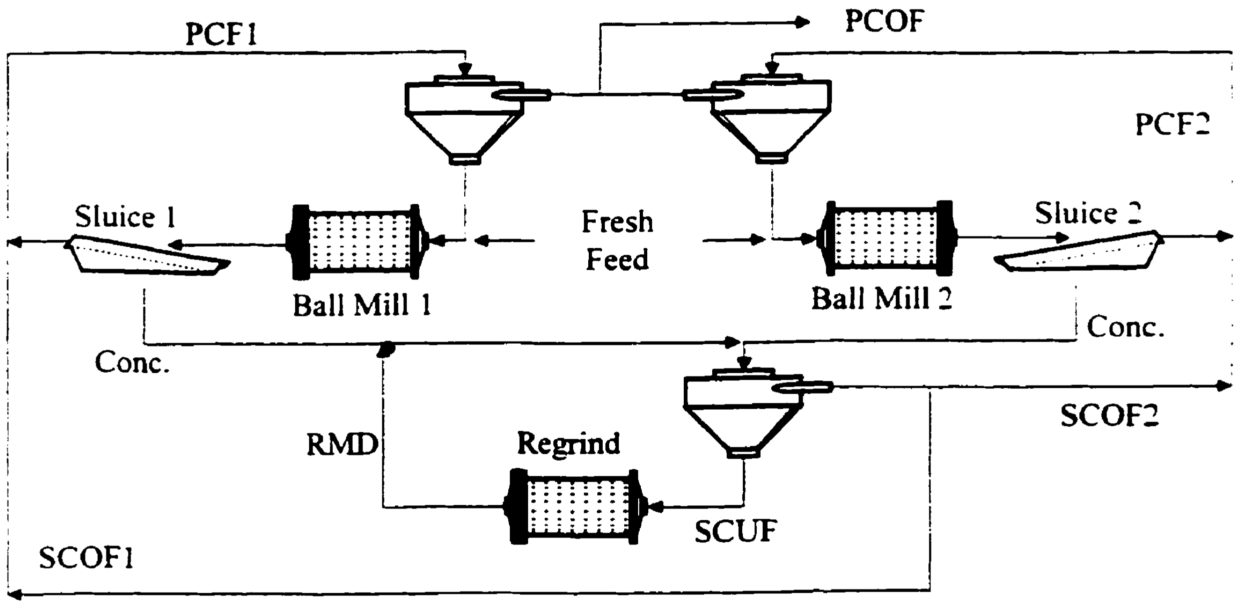
Flowsheet 3 Hemlo Gold Mines Mill



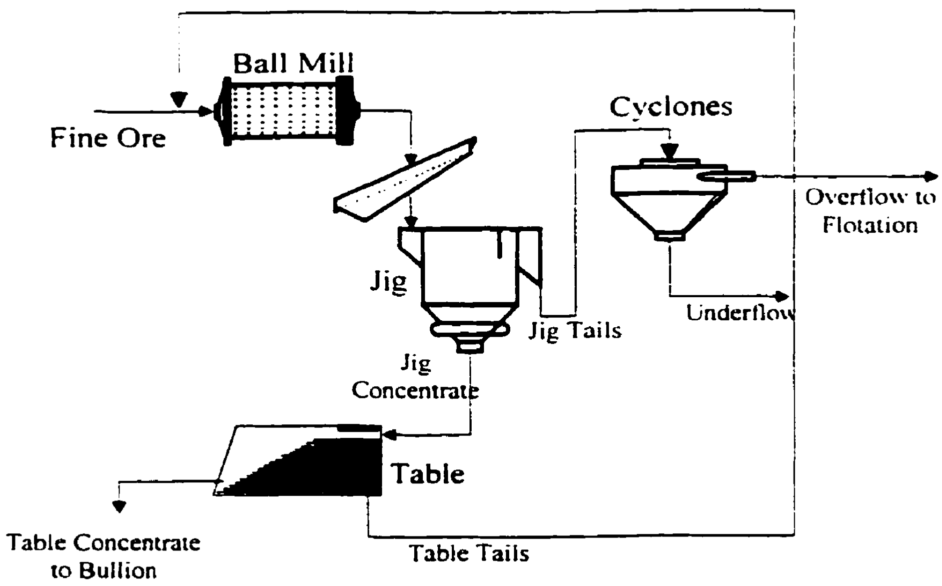
Flowsheet 4 Aurbel Mill



Flowsheet 5 Agnico-Eagle Mill



Flowsheet 6 Snip Operation Mill



APPENDIX B

LKC AND PLANT METALLURGICAL MASS BALANCES

Meston Lake Ressources PKC Feed T1
Feedrate = 300 g/min; Water jacket pressure = 3 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	20.3	7.8	2.19	26.5	472	5.0	0.26	73.5	492	5.1	0.34	3.5
600	20.7	7.9	0.26	4.71	514	5.4	0.21	95.3	534	5.5	0.21	2.4
420	24.5	9.4	3.21	43.6	572	6.0	0.18	56.4	597	6.1	0.30	3.8
300	24.1	9.2	3.12	42.3	625	6.6	0.16	57.7	649	6.7	0.27	3.7
210	25.3	9.7	6.93	57.1	708	7.5	0.19	42.9	733	7.5	0.42	6.4
150	36.6	14.0	8.85	57.2	1150	12.1	0.21	42.8	1187	12.2	0.48	11.8
105	32.1	12.3	11.4	52.5	1272	13.4	0.26	47.5	1304	13.4	0.53	14.6
75	30.6	11.7	16.0	55.1	1328	14.0	0.30	44.9	1358	13.9	0.65	18.6
53	19.0	7.3	17.3	60.6	969	10.2	0.22	39.4	987	10.1	0.55	11.3
37	14.8	5.7	26.5	60.9	1324	14.0	0.19	39.1	1339	13.7	0.48	13.5
25	8.3	3.2	28.7	67.8	491	5.2	0.23	32.2	499	5.1	0.70	7.3
-25	5.2	2.0	22.5	77.9	53	0.6	0.62	22.1	58	0.6	2.56	3.1
Total	261.4	100.0	10.1	55.1	9477	100.0	0.23	44.9	9738	100.0	0.49	100.0

Meston Lake Ressources PKC Tails T1
Feedrate = 300 g/min; Water jacket pressure = 3 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	5.8	6.4	0.22	0.32	394	2.7	0.99	99.7	400	2.7	0.98	9.2
600	7.0	7.6	2.45	14.5	547	3.7	0.18	85.5	554	3.7	0.21	2.8
420	8.7	9.5	2.00	13.4	699	4.7	0.16	86.6	708	4.7	0.18	3.0
300	9.8	10.7	4.27	21.1	879	5.9	0.18	78.9	888	6.0	0.22	4.6
210	10.3	11.3	6.67	29.8	1089	7.4	0.15	70.2	1099	7.4	0.21	5.4
150	14.9	16.4	7.80	20.6	1868	12.6	0.24	79.4	1883	12.6	0.30	13.2
105	13.0	14.2	10.2	20.9	2165	14.6	0.23	79.1	2178	14.6	0.29	14.7
75	10.9	12.0	14.3	22.7	2219	15.0	0.24	77.3	2230	15.0	0.31	16.1
53	5.9	6.4	16.6	24.2	1603	10.8	0.19	75.8	1609	10.8	0.25	9.4
37	3.6	3.9	35.7	25.3	2347	15.8	0.16	74.7	2350	15.8	0.21	11.8
25	1.0	1.1	95.2	44.3	824	5.6	0.15	55.7	825	5.5	0.27	5.2
-25	0.5	0.5	303.6	68.8	182	1.2	0.34	31.2	183	1.2	1.09	4.6
Total	91.2	100.0	11.1	23.6	14816	100.0	0.22	76.4	14907	100.0	0.29	100.0

Meston Lake Ressources PKC Feed T2
Feedrate = 400 g/min; Water jacket pressure = 5 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	4.6	4.9	0.34	2.3	403	3.6	0.17	97.7	408	3.6	0.17	1.3
600	7.9	8.5	3.94	19.2	692	6.2	0.19	80.8	700	6.3	0.23	3.0
420	10.6	11.3	6.44	26.4	807	7.3	0.24	73.6	818	7.3	0.32	4.7
300	14.2	15.2	16.0	60.3	1029	9.3	0.15	39.7	1044	9.3	0.36	6.8
210	13.5	14.4	17.6	53.7	993	8.9	0.21	46.3	1006	9.0	0.44	8.0
150	14.6	15.6	27.5	58.7	1258	11.3	0.23	41.3	1273	11.4	0.54	12.5
105	9.9	10.5	45.0	62.9	902	8.1	0.29	37.1	912	8.1	0.77	12.8
75	7.5	8.0	70.0	69.6	877	7.9	0.26	30.4	885	7.9	0.85	13.6
53	3.3	3.6	85.6	57.5	871	7.8	0.24	42.5	875	7.8	0.57	9.0
37	4.5	4.8	102.6	77.1	863	7.8	0.16	22.9	868	7.7	0.69	11.0
25	1.7	1.8	132.2	52.3	1289	11.6	0.16	47.7	1291	11.5	0.34	7.9
-25	1.3	1.4	272.6	68.8	1121	10.1	0.15	31.2	1122	10.0	0.46	9.5
Total	93.5	100.0	34.9	59.4	11106	100.0	0.20	40.6	11200	100.0	0.49	100.0

Meston Lake Ressources PKC Tails T2
Feedrate = 450 g/min: Water jacket pressure = 4.8 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	5.4	5.6	0.89	5.0	345	3.7	0.26	95.0	350	3.7	0.27	2.1
600	10.1	10.4	0.72	6.6	684	7.3	0.15	93.4	694	7.4	0.16	2.4
420	10.6	11.0	2.29	10.7	769	8.2	0.26	89.3	779	8.3	0.29	5.0
300	14.6	15.1	3.08	19.2	858	9.2	0.22	80.8	873	9.3	0.27	5.1
210	13.8	14.4	7.40	33.4	1020	10.9	0.20	66.6	1034	11.0	0.30	6.7
150	15.3	15.9	8.20	33.0	941	10.1	0.27	67.0	957	10.1	0.40	8.3
105	9.3	9.6	10.7	28.9	788	8.4	0.31	71.1	797	8.5	0.43	7.6
75	7.0	7.2	13.6	27.2	749	8.0	0.34	72.8	756	8.0	0.46	7.7
53	4.0	4.2	16.4	32.9	547	5.9	0.25	67.1	551	5.8	0.36	4.4
37	3.4	3.5	38.9	53.9	561	6.0	0.20	46.1	564	6.0	0.43	5.3
25	1.4	1.5	132.9	50.8	784	8.4	0.24	49.2	786	8.3	0.48	8.3
-25	1.5	1.6	849.9	76.3	1287	13.8	0.31	23.7	1289	13.7	1.31	37.0
Total	96.4	100.0	22.6	47.9	9334	100.0	0.25	52.1	9430	100.0	0.48	100.0

Meston Lake Ressources PKC Feed T3
Feedrate = 500 g/min: Water jacket pressure = 5 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/t)
840	6.5	6.8	6.02	18.1	577	5.2	0.31	81.9	583	5.2	0.37
600	8.7	9.1	0.51	3.1	753	6.7	0.18	96.9	762	6.8	0.19
420	9.2	9.6	6.52	35.2	752	6.7	0.15	64.8	761	6.8	0.22
300	11.3	11.9	12.1	48.6	764	6.8	0.19	51.4	776	6.9	0.36
210	10.6	11.2	12.8	45.7	810	7.2	0.20	54.3	820	7.3	0.36
150	13.6	14.3	20.6	53.4	994	8.9	0.25	46.6	1007	8.9	0.52
105	11.0	11.5	31.7	57.2	1001	9.0	0.26	42.8	1012	9.0	0.60
75	9.9	10.4	49.7	58.1	1037	9.3	0.34	41.9	1047	9.3	0.80
53	4.5	4.8	62.0	52.0	927	8.3	0.28	48.0	931	8.3	0.58
37	5.8	6.1	92.1	73.3	878	7.8	0.22	26.7	884	7.8	0.82
25	2.2	2.4	121.8	53.3	1039	9.3	0.23	46.7	1041	9.2	0.49
-25	1.9	2.0	169.9	48.1	1654	14.8	0.21	51.9	1655	14.7	0.41
Total	95.2	100.0	30.5	52.4	11185	100.0	0.24	47.6	11280	100.0	0.49

Meston Lake Ressources PKC Tails T3
Feedrate = 500 g/min: Water jacket pressure = 4.8 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/t)
840	5.3	5.7	0.40	0.4	727	5.1	0.68	99.6	732	6.4	0.68
600	10.8	11.5	0.90	5.6	784	7.0	0.21	94.4	795	7.0	0.22
420	9.7	10.4	5.94	32.6	768	6.9	0.16	67.4	777	6.8	0.23
300	12.0	12.7	6.66	23.8	781	7.0	0.33	76.2	793	6.9	0.42
210	10.9	11.7	10.1	38.5	805	7.2	0.22	61.5	816	7.1	0.35
150	13.2	14.0	10.6	38.1	971	8.7	0.23	61.9	985	8.6	0.37
105	10.5	11.1	10.6	26.5	1106	9.9	0.28	73.5	1117	9.8	0.37
75	9.1	9.7	15.7	31.9	1086	9.7	0.28	68.1	1095	9.6	0.41
53	5.7	6.0	23.9	33.6	975	8.7	0.27	66.4	981	8.6	0.41
37	3.8	4.1	36.4	41.6	891	8.0	0.22	58.4	895	7.8	0.37
25	1.6	1.7	57.4	51.6	507	4.5	0.17	48.4	508	4.4	0.35
-25	1.3	1.4	125.8	33.0	1938	17.3	0.17	67.0	1939	17.0	0.25
Total	93.9	100.0	12.6	28.8	14186	100.0	0.21	71.2	11432	100.0	0.36

Meston Lake Ressources PKC Feed T4
Feedrate = 380 g/min: Water jacket pressure = 5 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	3.1	3.2	0.98	15.2	109	1.4	0.15	84.8	112	1.5	0.18	0.5
600	5.8	6.1	0.38	5.7	213	2.8	0.17	94.3	219	2.9	0.17	1.0
420	7.7	8.0	2.71	30.5	275	3.6	0.17	69.5	282	3.7	0.24	1.7
300	10.3	10.8	7.76	39.5	355	4.7	0.35	60.5	365	4.8	0.56	5.2
210	10.3	10.8	11.4	43.0	485	6.4	0.32	57.0	496	6.5	0.55	7.0
150	13.9	14.6	17.6	54.6	727	9.6	0.28	45.4	741	9.7	0.61	11.5
105	12.8	13.4	17.3	49.3	853	11.3	0.27	50.7	866	11.3	0.52	11.5
75	13.2	13.8	25.1	54.8	908	12.0	0.30	45.2	921	12.0	0.65	15.4
53	6.4	6.7	31.5	49.5	768	10.1	0.27	50.5	774	10.1	0.53	10.5
37	7.4	7.8	48.9	68.9	712	9.4	0.23	31.1	720	9.4	0.73	13.5
25	2.6	2.7	79.1	59.1	744	9.8	0.19	40.9	747	9.7	0.46	8.8
-25	2.0	2.0	151.4	56.7	1416	18.7	0.16	43.3	1418	18.5	0.37	13.4
Total	95.5	100.0	21.9	53.4	7565	100.0	0.24	46.6	7660	100.0	0.51	100.0

Meston Lake Ressources PKC Tails T4
Feedrate = 400 g/min: Water jacket pressure = 5 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/t)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/t)	Dist. (%)
840	3.0	3.3	0.19	4.4	62	1.2	0.20	95.6	65	1.3	0.20	0.9
600	6.1	6.7	0.44	19.2	132	2.7	0.09	80.8	138	2.7	0.10	0.9
420	6.4	7.0	1.12	9.9	158	3.2	0.41	90.1	165	3.2	0.44	4.8
300	9.1	10.0	1.36	24.3	213	4.3	0.18	75.7	222	4.4	0.23	3.4
210	9.1	10.0	1.60	20.3	338	6.8	0.17	79.7	347	6.8	0.21	4.8
150	13.2	14.5	4.17	43.3	451	9.1	0.16	56.7	464	9.2	0.27	8.5
105	12.1	13.2	4.79	29.9	617	12.4	0.22	70.1	629	12.4	0.31	12.9
75	12.7	14.0	5.54	26.8	665	13.4	0.29	73.2	678	13.4	0.39	17.6
53	9.1	10.0	9.62	41.1	481	9.7	0.26	58.9	490	9.7	0.43	14.2
37	5.8	6.4	10.9	41.1	462	9.3	0.20	58.9	468	9.2	0.33	10.3
25	2.3	2.6	16.3	26.7	676	13.6	0.16	73.3	678	13.4	0.21	9.5
-25	2.3	2.5	31.4	40.1	724	14.5	0.15	59.9	726	14.3	0.25	12.1
Total	91.2	100.0	5.29	32.2	4979	100.0	0.20	67.8	5070	100.0	0.30	100.0

East Malartic PKC Feed T1 (-10 mesh/1.70 mm)
Feedrate = 427 g/min: Water jacket: pressure = 5 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
840	9.1	6.3	26.5	60.0	329	2.6	0.49	40.0	339	2.6	1.19	2.6
600	9.4	6.5	30.4	80.2	414	3.2	0.17	19.8	424	3.3	0.84	2.3
420	8.3	5.7	28.8	82.2	435	3.4	0.12	17.8	443	3.4	0.65	1.9
300	11.2	7.8	26.0	71.8	610	4.7	0.19	28.2	621	4.8	0.65	2.6
210	11.7	8.2	30.3	59.5	815	6.3	0.30	40.5	827	6.3	0.72	3.9
150	18.7	13.0	46.0	67.8	1459	11.3	0.28	32.2	1478	11.3	0.86	8.2
105	22.3	15.6	54.2	61.9	2106	16.3	0.35	38.1	2129	16.3	0.92	12.7
75	25.9	18.1	70.5	58.8	2957	22.9	0.43	41.2	2983	22.9	1.04	20.2
53	15.5	10.8	97.2	57.0	1705	13.2	0.67	43.0	1720	13.2	1.53	17.1
37	7.7	5.3	155.7	53.0	941	7.3	1.12	47.0	949	7.3	2.37	14.6
25	2.1	1.5	220.9	41.5	408	3.2	1.62	58.5	410	3.1	2.76	7.4
-25	1.6	1.1	158.9	25.8	706	5.5	1.06	74.2	707	5.4	1.42	6.5
Total	143.5	100.0	60.9	56.7	12886	100.0	0.52	43.3	13030	100.0	1.18	100.0

East Malartic PKC Feed T2 (-50 mesh/300 µm)
Feedrate = 426 g/min: Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
210	12.5	9.9	26.5	48.0	734	7.4	0.49	52.0	746	7.4	0.92	5.5
150	19.0	15.2	34.1	59.8	1366	13.8	0.32	40.2	1385	13.8	0.78	8.7
105	25.8	20.6	36.3	57.7	1924	19.4	0.36	42.3	1950	19.4	0.83	13.1
75	32.9	26.2	53.8	69.6	2602	26.3	0.30	30.4	2635	26.3	0.96	20.4
53	20.2	16.1	89.8	75.1	1508	15.2	0.40	24.9	1528	15.2	1.58	19.4
37	9.9	7.9	157.0	76.6	817	8.2	0.58	23.4	827	8.2	2.45	16.2
25	2.9	2.3	229.1	65.0	347	3.5	1.03	35.0	350	3.5	2.92	8.2
-25	2.4	1.9	176.4	40.5	607	6.1	1.02	59.5	609	6.1	1.71	8.4
Total	125.6	100.0	64.8	65.4	9904	100.0	0.44	34.6	10030	100.0	1.24	100.0

Est Malartic PKC Feed T1 (-10 mesh)
(excerpt from T1)

Size (µm)	CONCENTRATE Weight (g)	% %Weight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (oz/st)	Distn. (%)
840	9.10	24.0	26.48	60.0	329	18.4	0.49	40.0	339	18.5	1.19	27.7
600	9.39	24.7	30.37	80.2	414	23.2	0.17	19.8	424	23.2	0.84	24.5
420	8.25	21.7	28.79	82.2	435	24.3	0.12	17.8	443	24.2	0.65	19.9
300	11.24	29.6	25.99	71.8	610	34.1	0.19	28.2	621	34.0	0.65	28.0
Total	37.98	100.0	27.80	72.7	1788	100.0	0.22	27.3	1826	100.0	0.80	100.0

East Malartic PKC Tails T3 (-50 mesh/300 µm)
Feedrate = 436 g/min: Water jacket pressure = 3.4 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
210	16.7	13.3	11.8	50.8	859	5.6	0.22	49.2	875	5.6	0.44	2.0
150	22.3	17.7	30.8	50.5	1935	12.5	0.35	49.5	1958	12.6	0.70	7.2
105	25.4	20.2	57.2	55.7	2756	17.8	0.42	44.3	2781	17.8	0.94	13.8
75	28.7	22.8	88.4	62.3	4031	26.1	0.38	37.7	4060	26.1	1.00	21.5
53	17.7	14.1	145.9	68.2	2486	16.1	0.49	31.8	2503	16.1	1.51	20.0
37	9.5	7.5	247.3	68.8	1413	9.1	0.75	31.2	1423	9.1	2.39	17.9
25	2.8	2.2	375.9	60.3	655	4.2	1.06	39.7	658	4.2	2.66	9.2
-25	2.8	2.2	215.5	37.3	1320	8.5	0.76	67.7	1323	8.5	1.21	8.4
Total	125.9	100.0	91.0	60.4	15454	100.0	0.49	39.65	15580	100.0	1.22	100.0

East Malartic PKC Tails T4 (+50 mesh/300 µm)
Feedrate = 380 g/min; Water jacket pressure = 5 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
840	6.8	6.5	0.38	7.7	207	9.39	0.15	92.3	214	9.3	0.16	3.4
600	15.1	14.3	2.04	27.0	353	16.00	0.24	73.0	368	15.9	0.31	11.6
420	22.2	21.1	0.63	13.4	444	20.14	0.20	86.6	466	20.2	0.22	10.6
300	61.3	58.2	4.23	35.6	1201	54.46	0.39	64.4	1262	54.6	0.58	74.3
Total	105.3	100.0	3.07	31.3	2205	100.0	0.31	68.7	2310	100.0	0.42	100.0

Hemlo Gold Mine PKC Feed T1
Feedrate = 422 g/min; Water jacket pressure = 4 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	3.3	2.9	30.7	84.5	160	1.2	0.12	15.5	163	1.2	0.74	0.8
420	5.2	4.5	55.9	76.0	370	2.7	0.25	24.0	375	2.7	1.02	2.6
300	9.8	8.5	55.9	74.0	817	5.9	0.24	26.0	827	5.9	0.90	5.0
210	12.6	10.9	42.2	62.0	1393	10.1	0.23	38.0	1406	10.1	0.61	5.8
150	23.3	20.3	46.7	69.0	2666	19.3	0.18	31.0	2689	19.3	0.59	10.7
105	22.9	19.9	47.5	67.0	2696	19.5	0.20	33.0	2719	19.5	0.60	11.0
75	20.2	17.5	73.9	75.3	2626	19.0	0.19	24.7	2646	19.0	0.75	13.4
53	10.8	9.4	139.0	81.0	1435	10.4	0.25	19.0	1446	10.4	1.28	12.5
37	5.1	4.4	329.4	82.5	890	6.4	0.40	17.5	895	6.4	2.26	13.7
25	1.3	1.1	875.2	74.4	334	2.4	1.13	25.6	335	2.4	4.39	10.0
-25	0.7	0.6	1852	62.6	459	3.3	1.75	37.4	459	3.3	4.68	14.5
Total	115.1	100.0	93.4	72.7	13845	100.0	0.29	27.3	13960	100.0	1.06	100.0

Hemlo Gold Mine PKC Tails T1
Feedrate = 396 g/min; Water jacket pressure = 4.6 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.2	4.4	1.16	9.4	100	1.4	0.57	90.6	106	1.5	0.60	0.9
420	7.0	6.0	51.2	88.9	191	2.7	0.23	11.1	198	2.8	2.04	6.0
300	12.3	10.6	12.3	58.5	408	5.8	0.26	41.5	420	5.9	0.61	3.9
210	14.1	12.1	14.3	49.7	686	9.7	0.30	50.3	700	9.8	0.58	6.0
150	21.6	18.6	16.8	59.1	1326	18.8	0.19	40.9	1348	18.8	0.46	9.2
105	21.3	18.3	15.4	57.0	1393	19.8	0.18	43.0	1414	19.7	0.41	8.6
75	18.2	15.7	33.1	63.9	1332	18.9	0.26	36.1	1350	18.9	0.70	14.1
53	9.9	8.5	58.8	74.1	742	10.5	0.27	25.9	752	10.5	1.04	11.7
37	4.6	4.0	149.0	74.3	471	6.7	0.50	25.7	476	6.6	1.94	13.8
25	1.3	1.1	427.4	76.3	171	2.4	1.02	23.7	173	2.4	4.27	11.0
-25	0.8	0.7	830.0	63.9	225	3.2	1.59	36.1	226	3.1	4.38	14.8
Total	116.3	100.0	38.5	66.8	7046	100.0	0.32	33.2	7162	100.0	0.94	100.0

Hemlo Gold Mine PKC Feed T2
Feedrate = 739 g/min; Water jacket pressure = 4.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.5	4.3	959	77.9	2998	4.1	0.50	22.1	3004	4.1	2.24	1.5
420	7.9	6.3	1102	48.1	4697	6.4	2.01	51.9	4705	6.4	3.86	4.0
300	16.3	12.9	1186	39.3	8365	11.4	3.57	60.7	8381	11.4	5.87	10.9
210	20.2	16.0	3069	58.9	10348	14.2	4.18	41.1	10368	14.2	10.2	23.2
150	28.6	22.6	2092	57.6	15312	20.9	2.88	42.4	15340	21.0	6.77	22.9
105	22.0	17.3	2156	64.8	12990	17.8	1.98	35.2	13012	17.8	5.62	16.1
75	14.1	11.1	2775	77.4	9788	13.4	1.17	22.6	9802	13.4	5.16	11.2
53	6.6	5.2	1374	63.0	4258	5.8	1.25	37.0	4264	5.8	3.36	3.2
37	3.4	2.7	914	41.8	2235	3.1	1.95	58.2	2239	3.1	3.34	1.6
25	1.1	0.9	6589	65.4	908	1.2	4.38	34.6	909	1.2	12.6	2.5
-25	1.0	0.8	7625	56.4	1190	1.6	4.84	43.6	1191	1.6	11.1	2.9
Total	126.7	100.0	2122	59.3	73088	100.0	2.53	40.7	73215	100.0	6.19	100.0

Hemlo Gold Mine PKC Tails T2
Feedrate = 625 g/min; Water jacket pressure = 4 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	6.0	5.1	492	46.8	1955	4.0	1.71	53.2	1961	4.0	3.21	3.3
420	8.4	7.2	641	60.0	3048	6.2	1.18	40.0	3056	6.2	2.95	4.8
300	16.9	14.3	691	56.2	5724	11.6	1.59	43.8	5741	11.6	3.61	11.0
210	19.7	16.7	1023	54.7	7409	15.0	2.25	45.3	7429	15.1	4.95	19.5
150	26.1	22.2	1252	71.2	10595	21.5	1.25	28.8	10621	21.5	4.32	24.3
105	19.0	16.1	1613	88.3	8735	17.7	0.46	11.7	8754	17.7	3.96	18.4
75	12.2	10.3	1406	87.8	6344	12.9	0.37	12.2	6356	12.9	3.06	10.3
53	5.6	4.8	1091	87.5	2830	5.7	0.31	12.5	2835	5.7	2.46	3.7
37	2.5	2.1	922	75.6	1445	2.9	0.52	24.4	1447	2.9	2.11	1.6
25	0.6	0.5	2287	63.3	517	1.1	1.61	36.7	518	1.0	4.39	1.2
-25	0.6	0.5	2658	42.5	631	1.3	3.24	57.5	632	1.3	5.64	1.9
Total	117.5	100.0	1122	69.9	49233	100.0	1.15	30.1	49350	100.0	3.82	100.0

Hemlo Gold Mine PKC Feed T3
Feedrate = 947 g/min; Water jacket pressure = 4.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.4	4.7	56	43.7	760	3.6	0.52	56.3	765	3.6	0.91	1.9
420	8.7	7.6	127	59.5	1229	5.9	0.61	40.5	1238	5.9	1.50	5.0
300	17.0	14.9	212	61.1	2210	10.5	1.03	38.9	2227	10.6	2.63	15.8
210	18.8	16.5	266	62.9	2910	13.9	1.02	37.1	2929	13.9	2.72	21.4
150	25.1	22.0	245	60.4	4310	20.5	0.94	39.6	4335	20.6	2.35	27.4
105	19.1	16.7	140	62.4	3923	18.7	0.41	37.6	3942	18.7	1.08	11.4
75	11.7	10.2	100	51.9	2974	14.2	0.36	48.1	2985	14.2	0.76	6.1
53	5.3	4.6	120	56.6	1351	6.4	0.36	43.4	1357	6.4	0.82	3.0
37	2.3	2.0	217	52.5	705	3.4	0.64	47.5	707	3.4	1.33	2.5
25	0.6	0.5	810	47.7	244	1.2	2.22	52.3	244	1.2	4.24	2.8
-25	0.3	0.3	810	23.7	363	1.7	2.22	76.3	364	1.7	2.91	2.8
Total	114.1	100.0	191	58.7	20979	100.0	0.73	41.3	21093	100.0	1.76	100.0

Hemlo Gold Mine PKC Tails T3
Feedrate = 952.1 g/min; Water jacket pressure = 4.3 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	6.1	5.2	458	83.3	794	3.8	0.70	16.7	800	3.8	4.17	9.8
420	9.8	8.3	115	40.6	1248	6.0	1.32	59.4	1258	6.0	2.20	8.1
300	18.3	15.6	202	58.3	2356	11.2	1.12	41.7	2374	11.3	2.67	18.7
210	20.1	17.1	256	60.9	3067	14.6	1.08	39.1	3087	14.6	2.73	24.8
150	26.3	22.3	17.2	12.4	4421	21.1	0.73	87.6	4447	21.1	0.82	10.8
105	18.1	15.4	89.8	50.4	3808	18.2	0.42	49.6	3826	18.1	0.84	9.5
75	11.3	9.6	56.7	45.0	2881	13.7	0.27	55.0	2892	13.7	0.49	4.2
53	4.9	4.2	54.5	38.2	1287	6.1	0.34	61.8	1292	6.1	0.54	2.1
37	2.1	1.8	1236	89.3	638	3.0	0.49	10.7	640	3.0	4.54	8.6
25	0.6	0.5	391	38.4	198	0.9	1.84	61.6	198	0.9	2.98	1.7
-25	0.1	0.1	391	7.0	282	1.3	1.84	93.0	282	1.3	1.98	1.6
Total	117.5	100.0	158	54.7	20413	100.0	0.75	45.3	20531	100.0	1.65	100.0

Hemlo Gold Mine PKC Feed T2/T3

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	10.9	4.5	167	82.3	755	3.6	0.52	17.7	766	3.6	2.88	1.9
420	16.6	6.9	218	82.9	1222	5.9	0.61	17.1	1238	5.9	3.52	3.8
300	33.3	13.8	275	80.2	2196	10.5	1.03	19.8	2230	10.6	5.13	10.0
210	39.0	16.2	586	88.6	2893	13.9	1.02	11.4	2932	13.9	8.81	22.5
150	53.7	22.3	436	85.4	4284	20.5	0.94	14.6	4337	20.6	6.32	23.9
105	41.0	17.0	397	91.1	3900	18.7	0.41	8.9	3941	18.7	4.54	15.6
75	25.8	10.7	483	92.0	2956	14.2	0.36	8.0	2982	14.1	4.54	11.8
53	11.8	4.9	273	87.1	1343	6.4	0.36	12.9	1355	6.4	2.73	3.2
37	5.7	2.4	245	75.8	700	3.4	0.64	24.2	706	3.3	2.61	1.6
25	1.8	0.7	1519	83.2	242	1.2	2.22	16.8	244	1.2	13.1	2.8
-25	1.3	0.5	1863	75.0	361	1.7	2.22	25.0	362	1.7	8.85	2.8
Total	240.8	100.0	412	86.7	20852	100.0	0.73	13.3	21093	100.0	5.43	100.0

Hemlo Gold Mine PKC Tails T2/T3

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	12.1	5.1	332	87.9	789	3.8	0.70	12.1	801	3.8	5.68	5.1
420	18.2	7.7	185	67.4	1241	6.0	1.32	32.6	1259	6.0	3.98	5.6
300	35.2	15.0	243	76.5	2341	11.2	1.12	23.5	2377	11.3	4.70	12.6
210	39.7	16.9	340	80.4	3048	14.6	1.08	19.6	3088	14.6	5.43	18.9
150	52.4	22.3	268	81.5	4394	21.1	0.73	18.5	4446	21.1	3.88	19.5
105	37.1	15.8	387	90.0	3785	18.2	0.42	10.0	3822	18.1	4.17	18.0
75	23.4	10.0	331	90.9	2864	13.7	0.27	9.1	2887	13.7	2.95	9.6
53	10.5	4.5	268	86.7	1279	6.1	0.34	13.3	1289	6.1	2.51	3.6
37	4.6	2.0	772	92.0	634	3.0	0.49	8.0	639	3.0	6.06	4.4
25	1.2	0.5	683	69.6	196	0.9	1.84	30.4	198	0.9	6.01	1.3
-25	0.7	0.3	999	56.5	280	1.3	1.84	43.5	281	1.3	4.22	1.3
Total	235.0	100.0	312	82.8	20296	100.0	0.75	17.2	20531	100.0	4.32	100.0

Aurbel Mine PKC Feed T1

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
840	14.3	13.2	18.4	33.6	1145	7.7	0.45	66.4	1159	7.7	0.68	5.5
600	9.5	8.8	10.7	14.5	1221	8.2	0.49	85.5	1231	8.2	0.57	4.9
420	10.8	10.0	24.7	33.6	1526	10.3	0.35	66.4	1537	10.2	0.52	5.6
300	16.8	15.6	44.6	53.6	2228	15.0	0.29	46.4	2245	15.0	0.62	9.8
210	17.6	16.3	48.1	47.1	2505	16.8	0.38	52.9	2522	16.8	0.71	12.6
150	19.5	18.0	58.0	44.7	2858	19.2	0.49	55.3	2877	19.2	0.88	17.8
105	11.5	10.6	82.7	45.6	1736	11.7	0.65	54.4	1748	11.7	1.19	14.6
75	5.2	4.8	186.7	48.6	956	6.4	1.06	51.4	961	6.4	2.06	13.9
53	1.8	1.6	312.9	64.4	222	1.5	1.36	35.6	224	1.5	3.80	6.0
37	0.8	0.8	498.6	54.6	283	1.9	1.23	45.4	284	1.9	2.70	5.4
25	0.2	0.2	919.8	57.5	86	0.6	1.73	42.5	87	0.6	4.06	2.5
-25	0.2	0.1	757.6	48.2	125	0.8	0.98	51.8	125	0.8	1.88	1.7
Total	108.2	100.0	60.6	45.9	14892	100.0	0.52	54.1	15000	100.0	0.95	100.0

Aurbel Mine PKC Tails T1

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Distn. (%)
840	25.3	20.1	4.29	17.4	1701	8.6	0.30	82.6	1726	8.7	0.36	6.9
600	10.8	8.6	1.31	3.1	1726	8.7	0.25	96.9	1737	8.7	0.26	5.1
420	11.5	9.1	3.63	7.6	2084	10.5	0.24	92.4	2096	10.5	0.26	6.1
300	17.4	13.8	5.60	10.4	3067	15.5	0.27	89.6	3084	15.5	0.30	10.5
210	17.5	13.9	4.46	6.6	3340	16.9	0.33	93.4	3357	16.9	0.35	13.2
150	20.0	15.8	5.72	6.7	3747	19.0	0.42	93.3	3767	18.9	0.45	19.0
105	12.8	10.1	6.00	4.9	2155	10.9	0.69	95.1	2168	10.9	0.72	17.4
75	6.6	5.2	5.38	4.1	1119	5.7	0.75	95.9	1126	5.7	0.77	9.7
53	2.5	2.0	16.4	8.7	342	1.7	1.27	91.3	345	1.7	1.38	5.3
37	1.2	1.0	36.8	12.9	255	1.3	1.18	87.1	256	1.3	1.35	3.9
25	0.4	0.3	85.6	23.3	109	0.6	0.93	76.7	109	0.5	1.21	1.5
-25	0.2	0.2	85.5	15.7	131	0.7	0.85	84.3	131	0.7	1.00	1.5
Total	126.1	100.0	5.58	7.9	19774	100.0	0.42	92.1	19900	100.0	0.45	100.0

Aurbel Mine PKC Feed T2

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
840	4.3	4.1	1.16	11.9	249	4.1	0.15	88.1	253	4.1	0.17	1.2
600	5.5	5.3	1.23	11.2	276	4.6	0.20	88.8	282	4.6	0.22	1.8
420	7.1	6.8	17.6	65.7	360	6.0	0.18	34.3	367	6.0	0.51	5.5
300	13.0	12.5	6.60	45.9	556	9.2	0.18	54.1	569	9.3	0.33	5.4
210	15.2	14.6	3.68	23.4	753	12.5	0.24	76.6	769	12.5	0.31	6.9
150	21.7	20.8	5.57	30.5	1138	18.8	0.24	69.5	1159	18.9	0.34	11.5
105	19.0	18.2	8.88	32.4	987	16.3	0.36	67.6	1006	16.4	0.52	15.1
75	10.9	10.4	26.0	38.7	715	11.8	0.63	61.3	726	11.8	1.01	21.1
53	4.1	3.9	51.7	46.2	308	5.1	0.80	53.8	312	5.1	1.46	13.2
37	2.0	1.9	93.8	56.6	185	3.1	0.77	43.4	187	3.1	1.77	9.6
25	0.8	0.8	117.9	64.9	102	1.7	0.51	35.1	103	1.7	1.43	4.3
-25	0.8	0.7	69.9	35.3	412	6.8	0.24	64.7	413	6.7	0.37	4.5
Total	104.2	100.0	13.4	40.5	6042	100.0	0.34	59.5	6146	100.0	0.56	100.0

Aurbel Mine PKC Tails T2

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Distn. (%)
840	8.4	7.1	1.31	8.3	816	6.2	0.15	91.7	824	6.2	0.16	2.6
600	7.8	6.6	0.86	4.0	816	6.2	0.20	96.0	823	6.2	0.20	3.2
420	8.9	7.5	3.18	14.0	965	7.3	0.18	86.0	974	7.3	0.21	3.9
300	14.3	12.0	3.80	17.7	1386	10.5	0.18	82.3	1401	10.5	0.22	6.0
210	15.8	13.3	4.06	13.2	1743	13.2	0.24	86.8	1758	13.2	0.28	9.4
150	22.0	18.5	6.04	18.6	2406	18.2	0.24	81.4	2428	18.2	0.29	13.9
105	18.5	15.6	7.79	17.2	1944	14.7	0.36	82.8	1962	14.7	0.43	16.3
75	11.9	10.0	15.7	17.8	1380	10.4	0.63	82.2	1392	10.4	0.76	20.4
53	6.0	5.1	20.0	20.5	587	4.4	0.80	79.5	593	4.4	0.99	11.4
37	3.4	2.9	23.7	22.4	361	2.7	0.77	77.6	365	2.7	0.99	7.0
25	1.2	1.0	21.7	19.9	199	1.5	0.51	80.1	200	1.5	0.63	2.4
-25	0.8	0.7	30.9	13.9	638	4.8	0.24	86.1	639	4.8	0.28	3.5
Total	119.1	100.0	7.39	17.1	13241	100.0	0.32	82.9	13360	100.0	0.39	100.0

Agnico-Eagle Division Laronde Cyclone Feed T1 (-840 µm)
Feedrate = 377 g/min: Water jacket pressure = 4 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	3.3	2.7	5.73	62.3	64	1.1	0.18	37.7	67	1.2	0.45	0.3
420	5.3	4.3	12.7	76.8	104	1.9	0.20	23.2	109	1.9	0.80	0.7
300	10.8	8.8	20.0	81.5	243	4.3	0.20	18.5	254	4.4	1.04	2.2
210	14.1	11.5	25.9	81.4	421	7.5	0.20	18.6	435	7.6	1.04	3.8
150	22.1	18.1	39.5	76.4	747	13.3	0.36	23.6	769	13.4	1.49	9.6
105	20.9	17.0	58.1	77.8	1001	17.9	0.35	22.2	1022	17.8	1.52	13.0
75	24.6	20.1	77.9	76.9	1313	23.4	0.44	23.1	1338	23.3	1.86	20.8
53	12.6	10.3	115.7	74.9	689	12.3	0.71	25.1	701	12.2	2.77	16.2
37	5.3	4.3	285.2	69.8	333	5.9	1.96	30.2	338	5.9	6.39	18.1
25	2.0	1.6	351.6	59.5	186	3.3	2.51	40.5	188	3.3	6.12	9.6
-25	1.6	1.3	186.8	43.4	507	9.0	0.76	56.6	508	8.9	1.34	5.7
Total	122.5	100.0	70.4	72.0	5608	100.0	0.60	28.0	5730	100.0	2.09	100.0

Agnico-Eagle Division Laronde Cyclone Feed T2 (-300 µm)
Feedrate = 436 g/min: Water jacket pressure = 4 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
210	28.3	20.6	0.25	83.1	271	6.1	0.01	16.9	299	6.6	0.03	0.1
150	31.1	22.6	26.4	82.5	627	14.2	0.28	17.5	658	14.4	1.51	10.1
105	24.7	18.0	44.2	77.4	874	19.8	0.36	22.6	898	19.7	1.57	14.3
75	27.7	20.1	53.6	74.3	1176	26.6	0.44	25.7	1203	26.4	1.66	20.2
53	15.4	11.2	86.8	72.8	608	13.7	0.82	27.2	623	13.7	2.94	18.5
37	6.0	4.3	213.7	66.0	286	6.5	2.29	34.0	292	6.4	6.61	19.5
25	2.4	1.7	252.9	54.7	170	3.8	2.94	45.3	173	3.8	6.41	11.2
-25	2.1	1.5	125.9	43.3	413	9.3	0.83	56.7	415	9.1	1.45	6.1
Total	137.5	100.0	50.0	69.6	4424	100.0	0.68	30.4	4561	100.0	2.16	100.0

Agnico-Eagle Division Laronde Cyclone Overflow T2 (-840 µm)
Feedrate = 393 g/min; Water jacket pressure = 3.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
210	9.9	6.1	0.10	23.5	77	1.9	0.04	76.5	87	2.1	0.05	0.4
150	11.6	7.1	0.21	12.4	227	5.7	0.08	87.6	238	5.7	0.08	1.7
105	12.5	7.7	0.19	10.2	298	7.5	0.07	89.8	311	7.5	0.08	2.0
75	20.9	12.8	0.32	15.8	439	11.0	0.08	84.2	459	11.0	0.09	3.7
53	35.5	21.8	0.38	15.6	588	14.7	0.12	84.4	623	15.0	0.14	7.5
37	31.2	19.2	1.53	23.3	669	16.7	0.24	76.7	700	16.8	0.29	17.8
25	32.1	19.7	4.64	31.1	932	23.3	0.35	68.9	964	23.2	0.50	41.4
-25	8.9	5.5	5.02	15.2	768	19.2	0.33	84.8	777	18.7	0.38	25.6
Total	162.7	100.0	1.64	23.1	3997	100.0	0.22	76.9	4160	100.0	0.28	100.0

Agnico-Eagle Division Laronde Cyclone Feed T3 (-840 µm, 1:3 silica dilution)
Feedrate = 431 g/min; Water jacket pressure = 4.4 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	3.2	2.7	0.29	24.4	53	0.3	0.05	75.6	56	0.3	0.07	0.04
420	4.2	3.6	11.1	51.4	130	0.8	0.34	48.6	134	0.8	0.68	1.0
300	11.6	9.7	15.2	87.2	1890	11.3	0.01	12.8	1902	11.3	0.11	2.2
210	19.7	16.5	15.2	68.5	5823	34.8	0.02	31.5	5843	34.7	0.07	4.7
150	29.8	25.0	22.0	88.2	3869	23.1	0.02	11.8	3899	23.1	0.19	8.0
105	22.2	18.6	44.4	83.3	1910	11.4	0.10	16.7	1932	11.5	0.61	12.8
75	16.8	14.1	87.8	82.4	1454	8.7	0.22	17.6	1470	8.7	1.22	19.4
53	7.6	6.4	174.2	83.4	678	4.1	0.39	16.6	685	4.1	2.31	17.1
37	2.5	2.1	491.0	76.8	341	2.0	1.11	23.2	344	2.0	4.74	17.6
25	0.8	0.7	823.8	66.9	174	1.0	1.88	33.1	174	1.0	5.66	10.7
-25	0.7	0.6	438.0	49.9	410	2.5	0.75	50.1	411	2.4	1.49	6.6
Total	119.1	100.0	60.3	77.5	16733	100.0	0.12	22.5	16852	100.0	0.55	100.0

Agnico-Eagle Division Laronde Cyclone Underflow T3 (-840 µm, 1:3 silica dilution)
Feedrate = 462 g/min; Water jacket pressure = 4.6 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	3.2	2.7	4.38	58.3	61	0.4	0.16	41.7	64	0.4	0.37	0.2
420	3.8	3.3	8.48	68.2	116	0.7	0.13	31.8	120	0.7	0.40	0.4
300	9.0	7.8	20.4	85.1	1293	7.9	0.02	14.9	1302	7.9	0.17	2.0
210	16.7	14.4	25.1	72.4	6100	37.2	0.03	27.6	6117	37.1	0.09	5.4
150	30.5	26.3	30.8	90.4	4011	24.5	0.02	9.6	4041	24.5	0.26	9.7
105	25.3	21.8	45.2	85.4	2094	12.8	0.09	14.6	2119	12.8	0.63	12.4
75	17.6	15.2	89.6	83.0	1539	9.4	0.21	17.0	1556	9.4	1.22	17.7
53	7.3	6.3	218.1	83.5	638	3.9	0.49	16.5	646	3.9	2.94	17.6
37	1.9	1.7	823.7	79.9	232	1.4	1.73	20.1	234	1.4	8.53	18.6
25	0.5	0.4	1649	66.5	102	0.6	3.90	33.5	102	0.6	11.6	11.0
-25	0.3	0.3	1061	60.6	206	1.3	1.01	39.4	206	1.3	2.57	4.9
Total	116.1	100.0	74.1	80.0	16392	100.0	0.13	20.0	16508	100.0	0.65	100.0

Agnico-Eagle Division Laronde Cyclone Overflow T3 (-840 µm, 1:3 silica dilution)
Feedrate = 397 g/min; Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	0.7	0.9	0.29	100.0	1	0.02	0.000	0.0	2	0.0	0.11	0.0
420	0.5	0.6	0.06	100.0	7	0.10	0.000	0.0	7	0.1	0.004	0.0
300	6.9	9.2	0.0003	100.0	385	5.50	0.000	0.0	391	5.5	0.000	0.0
210	17.1	22.7	0.07	26.2	2172	31.1	0.001	73.8	2189	31.0	0.002	1.0
150	18.6	24.6	0.08	24.3	1619	23.1	0.003	75.7	1637	23.2	0.004	1.4
105	7.9	10.5	0.04	4.1	821	11.7	0.01	95.9	829	11.7	0.01	1.7
75	6.4	8.4	0.13	8.4	478	6.8	0.02	91.6	485	6.9	0.02	2.3
53	7.2	9.5	0.22	7.7	295	4.2	0.06	92.3	302	4.3	0.07	4.7
37	5.3	7.0	1.27	14.4	295	4.2	0.14	85.6	300	4.2	0.16	10.8
25	2.0	2.7	10.1	26.8	203	2.9	0.27	73.2	205	2.9	0.37	17.3
-25	2.8	3.7	10.7	11.2	719	10.3	0.33	88.8	722	10.2	0.37	60.8
Total	75.3	100.0	0.83	14.3	6995	100.0	0.05	85.7	7070	100.0	0.06	100.0

Snip Operation Jig Feed T1
Feedrate = 311 g/min; Water jacket pressure = 2 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	7.7	8.3	6.50	29.3	246	5.5	0.49	70.7	254	5.5	0.68	1.7
420	9.0	9.7	8.98	25.9	366	8.2	0.63	74.1	375	8.2	0.83	3.0
300	14.1	15.2	13.1	29.2	601	13.4	0.74	70.8	615	13.4	1.03	6.1
210	14.8	15.9	16.2	27.5	645	14.4	0.98	72.5	660	14.4	1.32	8.4
150	13.9	15.0	49.1	41.1	704	15.7	1.39	58.9	718	15.7	2.32	16.1
105	12.9	13.9	106.6	62.8	480	10.7	1.69	37.2	492	10.8	4.44	21.1
75	9.5	10.2	135.5	69.5	371	8.3	1.52	30.5	380	8.3	4.86	17.9
53	5.2	5.6	176.1	79.8	241	5.4	0.97	20.2	246	5.4	4.69	11.2
37	2.9	3.1	219.7	81.1	237	5.3	0.62	18.9	239	5.2	3.23	7.5
25	1.2	1.3	183.1	81.7	98	2.2	0.49	18.3	99	2.2	2.63	2.5
-25	1.7	1.8	176.9	63.5	500	11.1	0.34	36.5	501	10.9	0.92	4.5
Total	92.7	100.0	64.2	57.6	4487	100.0	0.98	42.4	4580	100.0	2.26	100.0

Snip Operation Jig Feed T2
Feedrate = 300 g/min; Water jacket pressure = 3.6 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	6.0	6.4	20.5	38.5	300	5.4	0.65	61.5	306	5.4	1.04	1.3
420	8.3	8.9	24.5	39.4	423	7.6	0.74	60.6	431	7.6	1.20	2.2
300	14.0	14.9	22.5	34.4	673	12.1	0.89	65.6	687	12.2	1.33	3.9
210	14.8	15.9	41.3	36.7	721	13.0	1.46	63.3	736	13.0	2.27	7.1
150	15.4	16.4	98.8	55.0	840	15.2	1.48	45.0	856	15.2	3.22	11.7
105	12.6	13.4	227.0	68.8	676	12.2	1.91	31.2	689	12.2	6.02	17.6
75	11.1	11.8	469.3	81.5	687	12.4	1.72	18.5	698	12.4	9.12	27.0
53	4.4	4.7	595.3	81.8	343	6.2	1.68	18.2	347	6.2	9.14	13.4
37	4.1	4.3	469.5	85.3	268	4.8	1.23	14.7	272	4.8	8.21	9.5
25	1.5	1.6	373.8	77.9	179	3.2	0.89	22.1	181	3.2	4.00	3.1
-25	1.5	1.6	278.2	53.8	437	7.9	0.83	46.2	438	7.8	1.79	3.3
Total	93.5	100.0	174.3	69.0	5547	100.0	1.32	31.0	5640	100.0	4.18	100.0

Snip Operation Jig Feed T3
Feedrate = 246 g/min: Water jacket pressure = 3.6 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.9	6.4	15.8	27.3	323	5.3	0.76	72.7	329	5.3	1.03	1.3
420	7.9	8.7	11.8	21.5	452	7.4	0.75	78.5	459	7.4	0.94	1.6
300	13.8	15.1	18.3	24.3	690	11.3	1.13	75.7	703	11.4	1.47	3.9
210	14.9	16.4	39.9	30.7	943	15.5	1.43	69.3	957	15.5	2.03	7.3
150	15.9	17.5	120.3	60.9	787	12.9	1.56	39.1	803	13.0	3.91	11.7
105	12.2	13.4	300.2	72.7	717	11.8	1.92	27.3	729	11.8	6.92	18.8
75	10.6	11.6	565.5	81.6	748	12.3	1.80	18.4	758	12.3	9.66	27.4
53	3.6	3.9	707.1	81.9	362	5.9	1.54	18.1	366	5.9	8.45	11.5
37	3.4	3.8	619.9	87.9	252	4.1	1.16	12.1	255	4.1	9.45	9.0
25	1.3	1.5	507.1	78.1	197	3.2	0.96	21.9	198	3.2	4.34	3.2
-25	1.6	1.7	421.2	57.9	630	10.3	0.77	42.1	632	10.2	1.83	4.3
Total	91.0	100.0	185.8	69.4	6099	100.0	1.34	30.6	6190	100.0	4.33	100.0

Snip Operation Jig Feed T4
Feedrate = 340 g/min: Water jacket pressure = 3.6 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.8	6.3	2.16	8.7	367	4.9	0.36	91.3	373	4.9	0.39	0.8
420	8.7	9.4	11.8	26.1	563	7.5	0.51	73.9	572	7.5	0.68	2.1
300	14.6	15.8	10.8	22.3	964	12.8	0.57	77.7	979	12.9	0.72	3.7
210	15.4	16.6	20.0	26.2	1060	14.1	0.82	73.8	1075	14.1	1.09	6.2
150	16.8	18.2	63.2	53.2	1160	15.5	0.80	46.8	1177	15.5	1.69	10.5
105	12.3	13.3	232.9	75.0	908	12.1	1.05	25.0	920	12.1	4.14	20.1
75	9.7	10.5	337.6	78.2	822	11.0	1.11	21.8	832	10.9	5.02	22.0
53	3.2	3.5	562.2	83.5	357	4.8	0.99	16.5	360	4.7	5.96	11.3
37	3.3	3.5	670.5	87.2	340	4.5	0.94	12.8	343	4.5	7.29	13.2
25	1.3	1.4	453.2	83.4	148	2.0	0.78	16.6	149	2.0	4.67	3.7
-25	1.4	1.6	534.9	63.1	819	10.9	0.55	36.9	820	10.8	1.49	6.5
Total	92.3	100.0	130.9	69.0	7508	100.0	0.78	31.0	7600	100.0	2.49	100.0

Snip Operation Jig Tail T1
Feedrate = 254 g/min: Water jacket pressure = 3 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	9.1	9.7	9.63	20.0	641	5.7	0.55	80.0	650	5.7	0.67	1.9
420	11.3	12.0	9.33	19.3	877	7.8	0.50	80.7	888	7.9	0.62	2.4
300	17.5	18.6	19.3	22.6	1508	13.4	0.76	77.4	1526	13.5	0.98	6.6
210	16.1	17.2	34.4	37.6	1571	14.0	0.59	62.4	1587	14.0	0.93	6.6
150	15.6	16.6	105.8	52.0	1701	15.2	0.90	48.0	1717	15.2	1.85	14.1
105	11.1	11.8	249.4	66.2	1205	10.7	1.17	33.8	1216	10.8	3.43	18.5
75	7.0	7.4	491.6	75.7	1057	9.4	1.05	24.3	1064	9.4	4.27	20.2
53	3.2	3.4	847.5	87.4	464	4.1	0.85	12.6	467	4.1	6.67	13.8
37	1.5	1.6	776.3	75.5	618	5.5	0.60	24.5	620	5.5	2.44	6.7
25	0.8	0.8	658.8	75.9	253	2.3	0.62	24.1	253	2.2	2.57	2.9
-25	0.8	0.9	761.3	44.7	1321	11.8	0.58	55.3	1322	11.7	1.06	6.2
Total	93.9	100.0	148.2	61.8	11216	100.0	0.77	38.2	11310	100.0	1.99	100.0

Snip Jig Tail T2
Feedrate = 325 g/min; Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	25.1	19.0	15.9	42.0	468	4.5	1.18	58.0	493	4.7	1.93	2.2
420	13.6	10.3	12.6	23.3	771	7.4	0.73	76.7	784	7.4	0.94	1.7
300	15.7	11.9	31.0	29.3	1273	12.2	0.92	70.7	1289	12.2	1.29	3.9
210	16.3	12.3	43.6	25.9	1460	14.0	1.39	74.1	1476	13.9	1.86	6.4
150	19.0	14.3	126.3	55.4	1641	15.7	1.18	44.6	1660	15.7	2.61	10.1
105	16.2	12.2	320.0	65.6	1272	12.2	2.14	34.4	1288	12.2	6.13	18.3
75	14.4	10.9	650.7	80.9	1292	12.4	1.71	19.1	1307	12.3	8.86	26.9
53	7.2	5.4	947.3	88.9	601	5.7	1.42	11.1	608	5.7	12.6	17.8
37	2.5	1.9	828.9	74.8	620	5.9	1.13	25.2	623	5.9	4.48	6.5
25	1.2	0.9	694.0	82.2	206	2.0	0.86	17.8	207	2.0	4.80	2.3
-25	1.1	0.8	833.4	53.8	853	8.2	0.92	46.2	854	8.1	1.99	4.0
Total	132.3	100.0	221.9	68.2	10458	100.0	1.31	31.8	10590	100.0	4.06	100.0

Snip Operation Jig Tail T3
Feedrate = 210 g/min; Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	14.4	11.6	4.19	18.5	399	5.0	0.67	81.5	413	5.1	0.79	1.0
420	14.6	11.8	6.62	18.6	575	7.2	0.74	81.4	590	7.3	0.88	1.6
300	18.7	15.1	16.0	26.1	950	11.9	0.89	73.9	969	12.0	1.19	3.5
210	17.4	14.1	39.2	34.2	1132	14.2	1.16	65.8	1149	14.2	1.73	6.1
150	18.1	14.7	140.4	62.7	1196	15.0	1.26	37.3	1214	15.0	3.34	12.4
105	15.8	12.8	355.4	80.1	909	11.4	1.53	19.9	925	11.4	7.59	21.5
75	13.3	10.8	535.9	85.8	910	11.4	1.30	14.2	923	11.4	9.02	25.5
53	5.9	4.8	747.1	90.5	445	5.6	1.05	9.5	451	5.6	10.87	15.0
37	2.6	2.1	696.5	83.3	477	6.0	0.77	16.7	480	5.9	4.57	6.7
25	1.2	1.0	532.5	84.8	186	2.3	0.63	15.2	187	2.3	4.12	2.4
-25	1.5	1.2	590.3	62.7	797	10.0	0.64	37.3	798	9.9	1.72	4.2
Total	123.5	100.0	196.1	74.2	7976	100.0	1.05	25.8	8100	100.0	4.03	100.0

Snip Operation Jig Tail T4
Feedrate = 288 g/min; Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	8.6	9.2	2.00	8.7	438	5.4	0.41	91.3	447	5.5	0.44	1.0
420	9.8	10.4	7.77	19.8	634	7.9	0.48	80.2	644	7.9	0.59	1.9
300	15.7	16.8	11.9	23.1	1053	13.1	0.59	76.9	1069	13.1	0.75	4.1
210	15.8	16.9	20.8	32.5	1203	15.0	0.57	67.5	1219	15.0	0.83	5.1
150	16.1	17.3	169.1	70.5	1293	16.1	0.88	29.5	1309	16.1	2.95	19.6
105	12.3	13.2	65.8	46.4	938	11.7	1.00	53.6	950	11.7	1.84	8.9
75	8.5	9.1	398.5	82.3	875	10.9	0.84	17.7	883	10.8	4.67	20.9
53	3.5	3.8	903.0	90.7	377	4.7	0.87	9.3	380	4.7	9.27	17.8
37	1.5	1.6	1128	86.0	406	5.0	0.67	14.0	408	5.0	4.76	9.8
25	0.8	0.8	1043	88.3	157	2.0	0.66	11.7	158	1.9	5.61	4.5
-25	0.8	0.8	1154	70.3	672	8.4	0.57	29.7	673	8.3	1.90	6.5
Total	93.3	100.0	150.9	71.3	8047	100.0	0.71	28.7	8140	100.0	2.43	100.0

Snip Jig Hutch 1 T1
Feedrate = 235 g/min; Water jacket pressure = 3.3 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	15.4	7.3	39.52	51.7	275	6.0	2.07	48.3	291	6.0	4.05	2.2
420	21.5	10.2	36.42	48.4	488	10.6	1.70	51.6	510	10.6	3.17	3.1
300	36.1	17.1	70.94	62.0	849	18.5	1.84	38.0	885	18.4	4.66	7.9
210	40.9	19.5	105.6	71.6	861	18.7	1.99	28.4	901	18.7	6.69	11.5
150	41.3	19.6	185.2	80.0	962	20.9	1.99	20.0	1003	20.9	9.54	18.2
105	32.0	15.2	303.5	88.9	612	13.3	1.98	11.1	644	13.4	16.9	20.8
75	18.4	8.7	619.9	95.1	379	8.2	1.54	4.9	397	8.3	30.1	22.8
53	3.3	1.6	1167	94.9	122	2.7	1.69	5.1	125	2.6	32.3	7.7
37	1.2	0.6	1750	96.8	41	0.9	1.68	3.2	42	0.9	51.5	4.1
25	0.2	0.1	2347	94.9	7	0.1	4.06	5.1	7	0.1	77.6	1.0
-25	0.2	0.1	1789	88.6	5	0.1	8.33	11.4	5	0.1	70.5	0.7
Total	210.3	100.0	207.9	83.3	4600	100.0	1.90	16.7	4810	100.0	10.9	100.0

Snip Jig Hutch 2 T1
(Feedrate = 197 g/min; back-water pressure = 3.6 psi)

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	7.3	6.0	101.6	82.9	27	3.0	5.61	17.1	35	3.4	25.9	1.6
420	13.8	11.3	55.3	69.2	59	6.5	5.73	30.8	73	7.1	15.1	1.9
300	24.2	19.9	138.1	71.1	116	12.8	11.7	28.9	140	13.6	33.5	8.2
210	24.4	20.1	201.9	76.5	151	16.7	10.0	23.5	176	17.1	36.7	11.3
150	27.1	22.3	324.5	80.0	230	25.4	9.55	20.0	258	25.0	42.7	19.2
105	16.6	13.7	624.1	88.9	190	20.9	6.81	11.1	207	20.1	56.5	20.4
75	6.2	5.1	1740	93.4	110	12.1	6.93	6.6	117	11.3	99.3	20.2
53	0.8	0.7	5792	95.0	19	2.0	14.1	5.0	19	1.9	267	9.0
37	0.5	0.4	8094	98.0	3	0.3	26.4	2.0	3	0.3	1134	6.7
25	0.3	0.2	2677	98.4	1	0.1	14.2	1.6	1	0.1	670	1.4
-25	0.4	0.3	189.6	88.7	1	0.1	6.69	11.3	2	0.2	46.3	0.1
Total	121.6	100.0	492.8	86.1	908	100.0	8.79	13.94	1030	100.0	55.6	100.0

Snip Jig Hutch 1 T1 Diluted with Silica (2:1 dilution)
Feedrate = 220 g/min; Water jacket pressure = 3.4 psi
For the concentrate, the assay of the -25 µm was estimated to achieve a distribution of 0.66% (original assay: 8709 oz/st).

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	3.3	3.4	109.7	75.4	142	2.0	0.83	24.6	145	2.1	3.31	1.9
420	4.1	4.3	137.5	60.6	244	3.5	1.51	39.4	248	3.5	3.77	3.7
300	9.5	9.8	139.2	74.2	562	8.1	0.82	25.8	571	8.1	3.13	7.0
210	21.3	22.0	108.7	80.0	1335	19.2	0.43	20.0	1356	19.3	2.14	11.3
150	27.9	28.7	136.5	86.2	2031	29.3	0.30	13.8	2059	29.2	2.14	17.3
105	18.1	18.6	272.7	91.3	1259	18.1	0.37	8.7	1277	18.1	4.23	21.1
75	9.0	9.2	590.9	94.3	846	12.2	0.38	5.7	855	12.1	6.59	22.0
53	2.6	2.6	985.9	97.1	249	3.6	0.30	2.9	251	3.6	10.4	10.2
37	1.1	1.1	895.6	97.5	190	2.7	0.13	2.5	191	2.7	5.12	3.8
25	0.2	0.2	1130	87.3	64	0.9	0.49	12.7	64	0.9	3.88	1.0
-25	0.1	0.1	2158	96.4	23	0.3	0.27	3.6	23	0.3	7.47	0.7
Total	97.1	100.0	231.5	88.0	6943	100.0	0.44	12.0	7040	100.0	3.63	100.0

Snip Jig Hutch 2 T1 Diluted with Silica (2:1 dilution)
Feedrate = 280 g/min; Water jacket pressure = 4.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
420	4.4	4.8	137.5	82.8	55	3.2	2.28	17.2	59	3.2	12.3	3.4
300	9.2	10.0	265.9	93.8	137	7.9	1.18	6.2	146	8.0	17.8	12.0
210	23.5	25.6	130.1	90.2	486	28.0	0.68	9.8	510	27.9	6.63	15.7
150	28.1	30.7	71.0	84.0	545	31.4	0.70	16.0	573	31.3	4.15	11.0
105	14.4	15.7	194.8	88.5	243	14.0	1.50	11.5	257	14.1	12.3	14.6
75	7.9	8.6	698.6	94.9	155	8.9	1.92	5.1	162	8.9	35.7	26.8
53	2.0	2.1	1127	95.5	51	2.9	2.06	4.5	53	2.9	43.9	10.7
37	1.7	1.8	643.8	97.0	33	1.9	1.01	3.0	34	1.9	32.0	5.1
25	0.4	0.4	248.3	92.1	13	0.8	0.59	7.9	14	0.7	7.32	0.5
-25	0.2	0.2	116.5	73.0	21	1.2	0.33	27.0	21	1.1	1.21	0.1
Total	91.6	100.0	216.1	91.6	1738	100.0	1.04	8.4	1830	100.0	11.8	100.0

Snip Jig Hutch 1 T2
Feedrate = 267 g/min; Water jacket pressure = 3.4 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	7.2	7.1	145.3	61.7	226	5.4	2.86	38.3	234	5.4	7.25	2.4
420	10.2	10.1	138.4	64.8	389	9.3	1.97	35.2	399	9.3	5.45	3.1
300	17.7	17.5	166.9	58.4	656	15.6	3.21	41.6	674	15.7	7.51	7.2
210	18.4	18.2	209.3	64.2	670	15.9	3.20	35.8	688	16.0	8.70	8.6
150	19.7	19.5	395.9	74.5	799	19.0	3.34	25.5	819	19.0	12.8	15.0
105	14.1	13.9	813.9	79.8	667	15.9	4.34	20.2	681	15.8	21.1	20.6
75	10.8	10.7	1506	85.0	567	13.5	5.06	15.0	578	13.4	33.1	27.4
53	2.0	2.0	2944	86.0	156	3.7	6.19	14.0	158	3.7	43.8	9.9
37	0.9	0.8	3544	88.8	48	1.1	8.01	11.2	49	1.1	70.2	4.9
25	0.1	0.1	2964	87.8	11	0.3	4.58	12.2	11	0.3	37.1	0.6
-25	0.1	0.1	1436	76.1	9	0.20	5.91	23.9	9	0.2	24.4	0.3
Total	101.2	100.0	536.8	77.7	4199	100.0	3.70	22.3	4300	100.0	16.2	100.0

Snip Jig Hutch 2 T2
Feedrate = 177 g/min; Water jacket pressure = 3.4 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	7.9	6.6	245.6	76.4	77	3.3	7.75	23.6	85	3.5	29.8	2.6
420	11.1	9.3	350.7	88.3	153	6.6	3.35	11.7	164	6.7	26.8	4.5
300	20.2	17.0	244.7	68.8	305	13.1	7.36	31.2	325	13.3	22.1	7.4
210	21.0	17.7	305.7	70.7	372	16.0	7.16	29.3	393	16.1	23.1	9.4
150	24.2	20.3	409.0	72.9	518	22.3	7.07	27.1	542	22.2	25.0	13.9
105	18.4	15.5	1076	86.8	457	19.7	6.60	13.2	475	19.5	48.0	23.5
75	12.9	10.8	1927	91.8	344	14.8	6.45	8.2	356	14.6	75.9	27.9
53	2.2	1.9	3974	93.2	69	3.0	9.35	6.8	72	2.9	133	9.8
37	0.8	0.7	559.1	72.6	15	0.7	11.0	27.4	16	0.7	38.2	0.6
25	0.1	0.1	1852	84.0	6	0.3	6.42	16.0	6	0.2	39.4	0.2
-25	0.2	0.1	349.0	50.1	5	0.2	11.3	49.9	5	0.2	21.9	0.1
Total	118.9	100.0	683.3	83.7	2321	100.0	6.82	16.3	2440	100.0	39.8	100.0

Snip Jig Hutch 1 T2 Diluted with Silica (2:1 dilution)
Feedrate = 325 g/min; Water jacket pressure = 3.5 psi

Size (µm)	CONCENTRATE				TAILS				FEED			
	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	3.1	3.4	153.6	76.2	101	1.6	1.46	23.8	104	1.6	5.96	1.7
420	3.5	3.9	226.3	62.1	188	2.9	2.58	37.9	191	2.9	6.67	3.4
300	9.1	10.1	204.9	85.5	560	8.7	0.56	14.5	569	8.7	3.83	5.9
210	23.3	25.9	101.7	75.5	1534	23.8	0.50	24.5	1557	23.8	2.01	8.4
150	23.6	26.3	158.7	73.4	1927	29.9	0.71	26.6	1951	29.8	2.62	13.7
105	14.3	15.9	441.6	85.6	915	14.2	1.16	14.4	930	14.2	7.92	19.8
75	7.9	8.8	1335	89.7	705	10.9	1.70	10.3	713	10.9	16.42	31.5
53	2.4	2.7	1097	85.1	258	4.0	1.78	14.9	261	4.0	11.81	8.3
37	2.0	2.2	1029	93.7	156	2.4	0.89	6.3	158	2.4	13.98	6.0
25	0.5	0.5	578.7	90.3	57	0.9	0.53	9.7	57	0.9	5.44	0.8
-25	0.2	0.3	616.6	80.2	49	0.8	0.71	19.8	49	0.7	3.58	0.5
Total	89.8	100.0	347.0	83.8	6450	100.0	0.93	16.2	6540	100.0	5.68	100.0

Snip Jig Hutch 2 T2 Diluted with Silica (2:1 dilution)
Feedrate = 250 g/min; Water jacket pressure = 3.5 psi

Size (µm)	CONCENTRATE				TAILS				FEED			
	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
420	4.6	5.0	153.6	80.9	115	3.2	1.46	19.1	119	3.3	7.35	2.1
300	9.4	10.1	232.3	71.0	300	8.5	2.98	29.0	310	8.5	9.96	7.4
210	24.1	25.8	68.9	69.3	818	23.1	0.90	30.7	842	23.1	2.85	5.8
150	25.3	27.1	192.8	83.7	1068	30.1	0.89	16.3	1094	30.0	5.33	14.0
105	16.1	17.2	365.2	85.0	536	15.1	1.94	15.0	552	15.2	12.5	16.7
75	8.9	9.5	1463	92.6	406	11.4	2.56	7.4	415	11.4	33.9	33.9
53	2.4	2.5	2176	93.1	141	4.0	2.69	6.9	144	3.9	38.5	13.3
37	1.9	2.0	1316	96.1	82	2.3	1.22	3.9	84	2.3	30.6	6.2
25	0.4	0.4	390.9	86.1	31	0.9	0.81	13.9	32	0.9	5.73	0.4
-25	0.2	0.2	142.9	32.9	49	1.4	1.29	67.1	50	1.4	1.91	0.2
Total	93.4	100.0	387.4	87.0	3547	100.0	1.52	13.0	3640	100.0	11.4	100.0

Snip Jig Hutch 1 T3
Feedrate = 137 g/min; Water jacket pressure = 3.4 psi

Size (µm)	CONCENTRATE				TAILS				FEED			
	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	5.5	4.9	118.0	61.8	106	5.04	3.75	38.2	112	5.0	9.33	1.4
420	9.4	8.4	151.8	73.5	186	8.80	2.77	26.5	195	8.8	9.93	2.6
300	19.2	17.2	202.8	83.8	340	16.11	2.22	16.2	359	16.2	12.9	6.2
210	22.6	20.3	282.3	81.0	389	18.43	3.85	19.0	411	18.5	19.1	10.5
150	24.8	22.2	547.8	87.5	475	22.51	4.08	12.5	499	22.5	31.0	20.7
105	18.2	16.4	923.0	91.5	353	16.74	4.44	8.5	371	16.7	49.5	24.6
75	10.0	9.0	1654	93.0	213	10.12	5.79	7.0	223	10.1	79.6	23.8
53	1.3	1.1	4065	95.0	35	1.64	7.83	5.0	36	1.6	151.4	7.2
37	0.4	0.3	5188	96.5	7	0.35	9.25	3.5	8	0.4	250.9	2.6
25	0.1	0.1	2152	91.4	3	0.13	4.86	8.6	3	0.1	55.3	0.2
-25	0.1	0.1	367.3	70.0	3	0.13	6.53	30.0	3	0.1	20.9	0.1
Total	111.4	100.0	596.7	88.9	2109	100.0	3.92	11.1	2220	100.0	33.7	100.0

Snip Jig Hutch 2 T3
Feedrate = 233 g/min: Water jacket pressure = 3.8 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	1.8	1.3	579.6	94.4	13	0.5	4.68	5.6	15	0.6	73.7	0.9
420	2.8	2.1	784.6	89.6	47	2.0	5.55	10.4	50	2.0	50.2	2.1
300	8.4	6.2	536.0	80.1	144	6.1	7.79	19.9	152	6.1	36.9	4.6
210	15.2	11.3	392.6	75.8	269	11.4	7.08	24.2	284	11.4	27.7	6.5
150	29.0	21.5	645.7	79.1	548	23.2	9.01	20.9	577	23.1	41.0	19.6
105	35.3	26.3	660.0	83.2	615	26.0	7.64	16.8	651	26.0	43.0	23.2
75	33.0	24.5	869.4	88.2	560	23.7	6.87	11.8	593	23.7	54.8	26.9
53	6.5	4.9	1435	89.8	133	5.6	7.98	10.2	139	5.6	74.9	8.6
37	2.2	1.6	3340	92.4	29	1.2	20.3	7.6	31	1.3	248.1	6.5
25	0.2	0.2	4677	92.7	4	0.2	17.3	7.3	5	0.2	226.2	0.8
-25	0.2	0.1	1869	87.4	4	0.2	11.4	12.6	4	0.2	86.4	0.3
Total	134.5	100.0	760.1	84.6	2366	100.0	7.86	15.4	2500	100.0	48.3	100.0

Snip Jig Hutch 1 T3 Diluted with Silica (2:1 dilution)
Feedrate = 228 g/min: Water jacket pressure = 3.6 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	2.8	3.1	108.1	72.9	42	1.5	2.70	27.1	45	1.5	9.33	1.4
420	3.0	3.2	266.4	79.2	74	2.6	2.79	20.8	77	2.6	12.9	3.3
300	8.3	9.0	242.2	85.2	235	8.3	1.49	14.8	243	8.3	9.74	8.0
210	22.2	24.1	120.9	82.5	678	23.9	0.84	17.5	700	23.9	4.65	11.0
150	25.8	28.0	142.6	84.1	888	31.3	0.78	15.9	914	31.2	4.79	14.8
105	16.2	17.6	337.0	89.9	419	14.8	1.46	10.1	435	14.9	14.0	20.5
75	8.7	9.4	910.2	93.8	293	10.3	1.79	6.2	301	10.3	27.9	28.4
53	2.4	2.6	956.1	94.7	101	3.6	1.27	5.3	103	3.5	23.3	8.1
37	2.1	2.2	531.7	97.3	64	2.3	0.48	2.7	66	2.3	17.0	3.8
25	0.5	0.6	255.9	92.8	24	0.8	0.43	7.2	25	0.8	5.84	0.5
-25	0.2	0.3	172.3	79.2	20	0.7	0.55	20.8	21	0.7	2.62	0.2
Total	92.3	100.0	286.0	89.0	2838	100.0	1.15	11.0	2930	100.0	10.1	100.0

Snip Jig Hutch 2 T3 Diluted with Silica (2:1 dilution)
Feedrate = 237 g/min: Water jacket pressure = 4.5 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
420	2.5	2.5	513.8	96.4	31	0.9	1.51	3.6	34	0.9	38.9	3.1
300	6.9	7.1	242.9	79.7	255	7.0	1.67	20.3	262	7.0	7.99	5.0
210	22.6	23.4	67.99	75.6	851	23.5	0.58	24.4	873	23.5	2.33	4.8
150	26.9	27.9	98.64	70.0	1420	39.2	0.80	30.0	1446	38.9	2.62	9.0
105	19.5	20.2	348.3	80.3	643	17.7	2.60	19.7	662	17.8	12.8	20.2
75	12.3	12.8	1094	94.6	223	6.2	3.45	5.4	236	6.3	60.4	33.9
53	3.2	3.3	1567	91.8	108	3.0	4.09	8.2	112	3.0	48.5	12.9
37	2.1	2.2	1694	96.8	38	1.0	3.10	3.2	40	1.1	93.0	8.8
25	0.4	0.4	1602	80.0	58	1.6	2.56	20.0	58	1.6	12.7	1.8
-25	0.1	0.2	1288	86.7	29	0.8	1.02	13.3	29	0.8	7.67	0.5
Total	96.5	100.0	380.6	87.4	3624	100.9	1.46	12.6	3720	100.8	11.3	100.0

Snip Jig Hutch 1 T4
Feedrate = 229 g/min; Water jacket pressure = 3.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	4.4	4.0	170.2	86.6	82	3.9	1.41	13.4	87	3.9	9.98	2.1
420	8.1	7.3	116.2	76.6	168	8.0	1.70	23.4	176	8.0	6.96	3.0
300	17.8	16.2	110.9	79.5	322	15.3	1.58	20.5	339	15.4	7.32	6.0
210	21.0	19.1	108.8	73.7	402	19.1	2.04	26.3	423	19.1	7.35	7.5
150	26.3	23.9	201.7	78.6	501	23.9	2.89	21.4	527	23.9	12.8	16.3
105	19.4	17.6	412.8	90.4	364	17.3	2.35	9.6	383	17.4	23.1	21.4
75	10.9	9.8	1028	96.5	207	9.9	1.98	3.5	218	9.9	53.1	27.9
53	1.6	1.4	2615	96.9	37	1.8	3.50	3.1	39	1.8	109.7	10.3
37	0.7	0.6	3139	97.8	10	0.5	4.57	2.2	11	0.5	195.2	5.2
25	0.1	0.1	1478	91.3	3	0.1	4.40	8.7	3	0.1	48.9	0.4
-25	0.1	0.1	47.6	25.1	4	0.2	4.20	74.9	4	0.2	5.44	0.05
Total	110.4	100.0	333.7	88.8	2100	100.0	2.21	11.2	2210	100.0	18.8	100.0

Snip Jig Hutch 2 T4
Feedrate = 187 g/min; Water jacket pressure = 3.4 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	3.8	2.9	188.9	95.7	18	1.2	1.73	4.3	22	1.4	33.7	1.4
420	4.9	3.8	246.8	92.3	43	2.9	2.32	7.7	48	3.0	27.1	2.4
300	12.4	9.7	160.5	77.8	126	8.5	4.52	22.2	138	8.6	18.5	4.8
210	19.3	15.1	152.5	72.6	208	14.1	5.36	27.4	227	14.2	17.9	7.6
150	32.8	25.7	211.6	83.7	359	24.4	3.76	16.3	392	24.5	21.1	15.5
105	29.8	23.3	339.0	88.4	360	24.5	3.69	11.6	390	24.4	29.3	21.3
75	19.9	15.5	635.1	92.4	271	18.4	3.84	7.6	291	18.2	46.9	25.5
53	3.2	2.5	1723	92.8	58	4.0	7.29	7.2	62	3.9	96.3	11.1
37	1.4	1.1	3380	95.5	20	1.3	11.6	4.5	21	1.3	241	9.6
25	0.2	0.2	353.7	61.5	4	0.3	11.9	38.5	5	0.3	29.4	0.3
-25	0.2	0.1	1737	91.3	4	0.3	8.03	8.7	4	0.2	88.5	0.6
Total	127.9	100.0	369.9	88.3	1472	100.0	4.26	11.7	1600	100.0	33.5	100.0

Snip Jig Hutch 1 T4 with Silica (2:1 dilution)
Feedrate = 247 g/min; Water jacket pressure = 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	2.3	2.6	99.8	84.3	42	1.2	1.03	15.7	44	1.2	6.22	1.3
420	3.0	3.3	198.7	97.3	88	2.4	0.19	2.7	91	2.5	6.67	2.8
300	9.6	10.8	110.2	85.8	318	8.9	0.55	14.2	327	8.9	3.78	5.7
210	24.6	27.5	84.4	91.1	908	25.4	0.22	8.9	933	25.4	2.44	10.5
150	24.7	27.6	113.1	87.6	1128	31.5	0.35	12.4	1153	31.4	2.76	14.7
105	13.8	15.4	227.0	89.4	508	14.2	0.73	10.6	522	14.2	6.70	16.2
75	7.1	8.0	901.4	96.5	333	9.3	0.70	3.5	340	9.3	19.6	30.8
53	2.0	2.2	1135	96.8	119	3.3	0.64	3.2	121	3.3	19.5	10.9
37	1.8	2.0	732.6	97.5	77	2.2	0.43	2.5	79	2.2	16.8	6.1
25	0.4	0.5	299.6	95.3	30	0.8	0.21	4.7	31	0.8	4.54	0.6
-25	0.2	0.2	313.8	77.0	30	0.8	0.55	23.0	30	0.8	2.38	0.3
Total	89.4	100.0	224.3	92.7	3581	100.0	0.44	7.3	3670	100.0	5.89	100.0

Snip Jig Hutch 2 T4 with Silica (2:1 dilution)
Feedrate = 240 g/min; Water jacket pressure = 4.4 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
420	2.8	3.0	335.7	97.3	32	1.1	0.83	2.7	35	1.2	28.1	3.5
300	6.9	7.4	177.7	89.3	173	6.2	0.85	10.7	180	6.3	7.65	4.9
210	22.9	24.3	34.43	84.5	590	21.2	0.25	15.5	613	21.3	1.52	3.3
150	27.6	29.3	71.11	82.2	923	33.1	0.46	17.8	950	33.0	2.51	8.5
105	18.1	19.3	244.3	87.0	473	17.0	1.39	13.0	491	17.0	10.4	18.0
75	10.4	11.0	785.6	89.9	348	12.5	2.63	10.1	358	12.4	25.3	32.1
53	2.6	2.8	1419	90.0	121	4.3	3.39	10.0	123	4.3	33.1	14.5
37	2.1	2.2	1485	93.9	70	2.5	2.87	6.1	72	2.5	45.3	11.6
25	0.5	0.5	1554	91.2	24	0.9	2.89	8.8	25	0.9	32.1	2.8
-25	0.2	0.2	1085	80.3	33	1.2	1.68	19.7	33	1.1	8.46	1.0
Total	94.1	100.0	267.9	89.2	2786	100.0	1.10	10.8	2880	100.0	9.81	100.0

Snip Operation Cyclone Feed T5
Feedrate = 368 g/min; Water jacket pressure = 5.5 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	6.5	6.4	1.50	19.5	289	3.0	0.14	80.5	296	3.1	0.17	0.3
420	7.9	7.8	1.81	5.3	426	4.5	0.60	94.7	434	4.5	0.62	1.4
300	13.5	13.5	7.20	31.9	793	8.3	0.26	68.1	806	8.3	0.38	1.6
210	16.0	15.9	12.2	35.6	1134	11.9	0.31	64.4	1150	11.9	0.48	2.9
150	17.9	17.8	29.8	39.1	1550	16.2	0.53	60.9	1568	16.2	0.87	7.1
105	14.2	14.1	69.7	53.9	1321	13.8	0.64	46.1	1335	13.8	1.38	9.6
75	12.4	12.3	168.2	56.0	2246	23.5	0.73	44.0	2258	23.4	1.65	19.5
53	4.8	4.8	317.5	78.6	396	4.1	1.04	21.4	401	4.2	4.81	10.1
37	4.2	4.2	866.2	79.2	582	6.1	1.65	20.8	586	6.1	7.90	24.3
25	1.7	1.7	1344	82.2	618	6.5	0.80	17.8	619	6.4	4.46	14.5
-25	1.5	1.5	878.5	79.8	204	2.1	1.63	20.2	206	2.1	8.01	8.6
Total	100.6	100.0	126.3	66.6	9559	100.0	0.67	33.4	9660	100.0	1.97	100.0

Snip Operation Cyclone Overflow T5
Feedrate = 293 g/min; Water jacket pressure = 5 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
210	3.1	4.1	2.12	28.2	121	2.5	0.14	71.8	125	2.6	0.19	1.3
150	6.6	8.7	1.08	17.3	420	8.8	0.08	82.7	426	8.8	0.10	2.3
105	9.5	12.4	1.47	20.5	465	9.7	0.12	79.5	474	9.8	0.14	3.8
75	19.1	25.0	2.06	29.3	524	11.0	0.18	70.7	543	11.2	0.25	7.4
53	13.7	18.0	3.35	29.4	404	8.5	0.27	70.6	417	8.6	0.38	8.6
37	13.9	18.2	9.81	49.7	469	9.8	0.29	50.3	483	10.0	0.57	15.2
25	5.2	6.8	28.5	60.1	339	7.1	0.29	39.9	344	7.1	0.71	13.6
-25	5.3	6.9	53.7	32.5	2032	42.6	0.29	67.5	2037	42.0	0.43	47.9
Total	76.5	100.0	8.89	37.5	4774	100.0	0.24	62.5	4850	100.0	0.37	100.0

Snip Operation Cyclone Underflow T5
Feedrate = 387 g/min: Water jacket pressure = 5.8 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	5.8	5.5	1.22	4.5	371	3.8	0.40	95.5	376	3.8	0.42	0.6
420	7.5	7.1	2.31	7.9	539	5.5	0.38	92.1	547	5.5	0.40	0.8
300	13.9	13.1	7.83	27.8	1031	10.6	0.27	72.2	1045	10.6	0.37	1.4
210	18.0	17.0	17.0	36.2	1402	14.4	0.39	63.8	1420	14.4	0.60	3.1
150	19.0	18.0	54.9	50.6	1665	17.1	0.61	49.4	1684	17.1	1.22	7.6
105	15.5	14.7	101.7	59.8	1281	13.1	0.83	40.2	1297	13.2	2.04	9.7
75	14.7	13.9	193.9	66.2	1593	16.3	0.91	33.8	1608	16.3	2.67	15.7
53	5.5	5.2	395.8	69.5	726	7.4	1.31	30.5	732	7.4	4.26	11.4
37	3.7	3.5	1562	85.1	373	3.8	2.72	14.9	377	3.8	18.1	25.0
25	1.1	1.1	2895	78.4	526	5.4	1.73	21.6	527	5.3	8.02	15.5
-25	1.1	1.0	1872	78.5	245	2.5	2.20	21.5	246	2.5	10.2	9.2
Total	105.7	100.0	181.1	70.2	9754	100.0	0.83	29.8	9860	100.0	2.77	100.0

Snip Operation Cyclone Underflow T6
Feedrate = 545 g/min: Water jacket pressure = 3 psi

Size (µm)	CONCENTRATE Weight (g)	eight	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	eight	Grade (oz/st)	Rec. (%)	FEED Weight (g)	eight	Grade (oz/st)	Dist. (%)
600	5.6	5.6	8.54	12.0	583	3.6	0.60	88.0	588	3.6	0.68	0.8
420	7.3	7.4	6.71	5.7	881	5.5	0.92	94.3	888	5.5	0.97	1.7
300	13.3	13.3	15.5	14.4	1597	9.9	0.76	85.6	1611	9.9	0.89	2.9
210	15.0	15.1	27.3	17.3	2198	13.6	0.89	82.7	2213	13.6	1.07	4.8
150	18.5	18.6	78.1	30.7	2810	17.4	1.16	69.3	2828	17.4	1.66	9.5
105	14.5	14.6	184.8	47.1	2204	13.6	1.37	52.9	2218	13.7	2.57	11.6
75	12.7	12.8	259.5	50.5	2339	14.5	1.38	49.5	2352	14.5	2.77	13.2
53	7.2	7.2	785.0	73.5	1268	7.9	1.61	26.5	1275	7.8	6.03	15.6
37	3.3	3.3	2084	80.8	658	4.1	2.48	19.2	662	4.1	12.9	17.2
25	1.1	1.1	4077	76.7	340	2.1	3.92	23.3	341	2.1	16.8	11.6
-25	1.0	1.0	3729	66.4	1269	7.9	1.45	33.6	1270	7.8	4.30	11.1
Total	99.4	100.0	288.5	58.1	16148	100.0	1.28	41.9	16247	100.0	3.04	100.0

Snip Jig Tail T6
Feedrate = 341 g/min: Water jacket pressure = 3.6 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.8	5.4	1.69	8.7	195	2.9	0.53	91.3	200	2.9	0.56	0.6
420	6.4	6.0	5.64	14.1	292	4.3	0.75	85.9	299	4.3	0.85	1.5
300	11.8	11.1	8.30	18.9	555	8.2	0.75	81.1	567	8.2	0.91	3.0
210	14.2	13.4	15.1	26.1	759	11.2	0.79	73.9	773	11.2	1.06	4.7
150	19.2	18.1	28.4	32.5	1011	14.9	1.12	67.5	1031	14.9	1.63	9.6
105	16.7	15.8	89.3	61.0	886	13.0	1.07	39.0	903	13.1	2.71	14.0
75	15.7	14.8	111	57.6	986	14.5	1.30	42.4	1002	14.5	3.01	17.3
53	8.3	7.8	207	72.0	560	8.2	1.19	28.0	568	8.2	4.17	13.6
37	4.3	4.1	460	79.1	343	5.0	1.53	20.9	347	5.0	7.23	14.4
25	1.6	1.5	800	76.6	226	3.3	1.72	23.4	228	3.3	7.28	9.5
-25	2.2	2.1	637	67.9	984	14.5	0.67	32.1	986	14.3	2.10	11.8
Total	105.9	100.0	99.1	60.2	6798	100.0	1.02	39.8	6904	100.0	2.53	100.0

Snip Jig Concentrate T6
Feedrate = 326 g/min: Water jacket pressure = 3.8 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
600	5.9	4.8	39.9	43.6	221	3.0	1.37	56.4	227	3.0	2.37	1.2
420	6.9	5.7	35.4	31.5	435	5.9	1.22	68.5	442	5.9	1.76	1.8
300	14.4	11.8	58.5	40.4	860	11.7	1.45	59.6	874	11.7	2.39	4.8
210	17.8	14.6	71.0	42.1	1175	16.0	1.48	57.9	1193	15.9	2.52	6.9
150	25.4	20.8	106	46.6	1616	21.9	1.91	53.4	1641	21.9	3.52	13.2
105	23.6	19.3	177	61.7	1410	19.1	1.84	38.3	1433	19.1	4.73	15.5
75	18.8	15.4	347	76.9	1122	15.2	1.75	23.1	1141	15.2	7.45	19.4
53	6.8	5.6	853	88.5	356	4.8	2.14	11.5	363	4.8	18.2	15.1
37	1.8	1.5	2639	90.3	96	1.3	5.27	9.7	98	1.3	53.1	11.9
25	0.4	0.3	6235	89.5	31	0.4	8.71	10.5	32	0.4	81.8	5.9
-25	0.3	0.3	5408	90.5	43	0.6	4.20	9.5	43	0.6	43.8	4.3
Total	122.1	100.0	250	69.9	7365	100.0	1.79	30.1	7487	100.0	5.84	100.0

Snip Operation Table Tails, Coarse Material
Feedrate = 450 g/min; Water jacket pressure = 3.2 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Rec. (%)	Weight (g)	eight	Grade (oz/st)	Dist. (%)
840	6.3	5.8	49.8	37.1	198	1.7	2.66	62.9	205	1.7	4.11	2.7
600	6.2	5.7	77.3	22.9	621	5.3	2.60	77.1	628	5.3	3.34	6.6
420	12.2	11.2	13.7	4.8	1513	12.8	2.19	95.2	1525	12.8	2.28	11.0
300	24.5	22.6	11.1	5.3	2975	25.2	1.65	94.7	2999	25.2	1.73	16.4
210	9.6	8.8	9.58	1.2	4090	34.6	1.78	98.8	4099	34.4	1.80	23.4
150	37.7	34.7	5.08	7.1	1078	9.1	2.34	92.9	1116	9.4	2.43	8.6
105	6.3	5.8	248.6	51.9	614	5.2	2.35	48.1	620	5.2	4.84	9.5
75	3.7	3.4	412.8	66.9	412	3.5	1.83	33.1	416	3.5	5.47	7.2
53	1.4	1.3	701.1	70.3	154	1.3	2.64	29.7	155	1.3	8.82	4.3
37	0.7	0.7	2308	76.7	70	0.6	7.39	23.3	71	0.6	31.41	7.1
25	0.1	0.1	3562	84.5	12	0.1	6.45	15.5	13	0.1	41.14	1.6
-25	0.0	0.0	9028	35.2	67	0.6	4.63	64.8	67	0.6	7.15	1.5
Total	108.7	100.0	72.4	24.9	11805	100.0	2.01	75.1	11914	100.0	2.65	100.0

Snip Operation Table Tails, Fine Material Without Magnetite
Feedrate = 340 g/min; Water jacket pressure = 3.7 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	eight (%)	Grade (oz/st)	Rec. (%)	Weight (g)	eight (%)	Grade (oz/st)	Rec. (%)	Weight (g)	eight (%)	Grade (oz/st)	Dist. (%)
150	59.4	43.2	79.69	51.8	2729	34.7	1.62	48.2	2788	34.9	3.28	19.2
105	41.0	29.9	151.8	56.2	2410	30.7	2.01	43.8	2451	30.6	4.52	23.2
75	23.7	17.3	260.1	68.7	1646	20.9	1.71	31.3	1670	20.9	5.38	18.8
53	8.4	6.1	467.4	77.5	654	8.3	1.75	22.5	662	8.3	7.67	10.6
37	4.1	3.0	2013	88.8	280	3.6	3.73	11.2	284	3.6	32.7	19.5
25	0.6	0.4	4601	84.1	52	0.7	9.60	15.9	52	0.7	59.6	6.5
-25	0.1	0.1	5931	67.6	89	1.1	3.69	32.4	89	1.1	11.4	2.1
Total	137.3	100.0	237.6	68.4	7860	100.0	1.92	31.6	7997	100.0	5.97	100.0

Snip Table Tails. Fine Material with Magnetite
Feedrate = 340 g/min: Water jacket pressure = 3.4 psi

Size (µm)	CONCENTRATE Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	TAILS Weight (g)	Weight%	Grade (oz/st)	Rec. (%)	FEED Weight (g)	Weight%	Grade (oz/st)	Dist. (%)
150	5.5	4.0	455	28.1	2790	35.7	2.28	71.9	2796	34.9	3.17	18.5
105	5.4	3.9	1039	51.8	2376	30.4	2.20	48.2	2382	29.7	4.56	22.7
75	3.6	2.6	1756	66.6	1565	20.0	2.00	33.4	1569	19.6	5.97	19.6
53	0.9	0.6	4382	76.9	606	7.7	1.93	23.1	607	7.6	8.35	10.6
37	0.6	0.5	12073	87.1	278	3.6	4.01	12.9	278	3.5	31.1	18.1
25	0.2	0.1	13005	79.7	54	0.7	10.3	20.3	55	0.7	50.9	5.8
-25	0.0	0.0	18402	72.4	156	2.0	2.16	27.6	156	1.9	7.82	2.6
Mag.	122.2	88.3	8.07	100.0	43		0.00	0.0	165	2.1	5.98	2.1
Total	138.4	100.0	216.0	62.5	7869	100.0	2.28	37.5	8007	100.0	5.97	100.0

Gold distribution inside the concentrate:

	Weight (g)	Weight% (%)	Grade (oz/st)	Rec. (%)	Total Rec. (%)
Gold-concentrate	16.2	11.7	1788	96.7	60.4
Mag-concentrate	122.2	88.3	8.07	3.30	2.06
Total	138.4	100.0	7405	100.0	62.5

Note: Collected 34.89 g from the overflow when feeding the 200 g magnetite. 42.9 g magnetite mixed with the tails
Pure magnetite does not contain gold, and we assumed that the magnetite in the tail does not contain gold either. However, for the reason of balance, when calculated feed gold grade from the concentrate and the tail, it showed some gold content.

Snip Ball Mill Feed and Discharge PS1

Size (µm)	Ball Mill Feed Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)	Ball Mill Discharge/Jig Feed Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)
11800	99	1.7	98.3	0.16	0.3					
9600	684	11.8	86.5	0.57	6.1	10	0.3	99.7	0.13	0.0
6300	1257	21.7	64.8	1.08	21.0	43	1.3	98.4	0.21	0.1
4000	796	13.7	51.1	0.84	10.3	71	2.2	96.2	0.38	0.2
2000	704	12.1	38.9	1.31	14.2	102	3.1	93.1	0.53	0.4
1000	552	9.5	29.4	1.05	8.9	130	4.0	89.1	0.56	0.6
500	387	6.7	22.7	1.31	7.8	221	6.7	82.4	0.80	1.4
212	345	5.9	16.8	1.90	10.1	774	23.6	58.8	2.20	13.0
150	109	1.9	14.9	2.05	3.5	383	11.7	47.1	2.90	8.5
105	109	1.9	13.0	1.81	3.0	393	12.0	35.1	4.85	14.6
75	101	1.7	11.3	2.13	3.3	325	9.9	25.2	6.21	15.5
53	90	1.6	9.7	1.99	2.8	187	5.7	19.5	11.6	16.6
45	48	0.8	8.9	1.86	1.4	87	2.7	16.8	14.6	9.8
37	39	0.7	8.2	1.45	0.9	58	1.8	15.0	12.2	5.4
-37	477	8.2		0.89	6.5	493	15.0		3.68	13.9
Total	5797	100.0		1.12	100.0	3278	100.0		3.98	100.0

Snip Cyclone Underflow and Overflow PS1

Size (µm)	Cyclone Underflow Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)	Cyclone Overflow Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)
9600	17	0.6	99.4	0.42	0.05					
6300	86	3.0	96.4	1.40	0.8					
4000	102	3.6	92.8	0.66	0.5					
2000	107	3.8	89.1	1.53	1.1					
1000	135	4.7	84.3	0.63	0.6					
500	244	8.6	75.8	1.03	1.7					
212	791	27.7	48.1	1.43	7.6	5	2.6	97.4	0.28	1.1
150	359	12.6	35.5	3.57	8.7	17	8.3	89.1	0.22	2.7
105	347	12.2	23.3	5.97	14.0	19	9.7	79.3	0.37	5.3
75	267	9.3	14.0	8.28	14.9	19	9.6	69.8	0.48	6.9
53	125	4.4	9.6	18.8	15.9	18	9.0	60.8	0.69	9.4
45	44	1.5	8.1	34.7	10.3	10	5.2	55.5	0.82	6.4
38	23	0.8	7.3	33.0	5.2	7	3.7	51.8	0.93	5.2
-38	208	7.3		13.3	18.7	104	51.8		0.81	63.0
Total	2856	100.0		5.18	100.0	200	100.0		0.67	100.0

Snip Ball Mill Feed and Discharge PS2

Size (µm)	Ball Mill Feed Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Ball Mill Discharge/Jig Feed Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)
11800	260	7.2	92.8						
9600	816	22.5	70.3		1.4	1.4	98.7	0.06	0.04
6300	984	27.1	43.2		1.2	1.2	97.4	0.22	0.1
4000	487	13.4	29.7		1.3	1.3	96.1	0.25	0.1
2000	339	9.4	20.4		2.1	2.1	94.0	0.34	0.3
1000	170	4.7	15.7		3.0	3.0	91.0	0.51	0.7
500	97	2.7	13.0		6.0	6.0	85.0	0.76	2.0
300	55	1.5	11.5		9.5	9.5	75.5	0.71	0.0
212	36	1.0	10.5		10.5	10.5	65.0	0.77	3.4
150	40	1.1	9.4		13.3	13.3	51.7	1.44	8.2
105	43	1.2	8.2		11.9	11.9	39.8	2.46	12.6
75	42	1.2	7.1		11.4	11.4	28.5	3.35	16.3
53	37	1.0	6.0		7.4	7.4	21.1	4.40	13.9
38	35	1.0	5.1		4.3	4.3	16.8	6.90	12.7
-38	183	5.1			16.8	16.8		3.72	26.8
Total	3626	100.0		0.62	100.0	100.0		2.33	100.0

Snip Cyclone Underflow and Overflow PS2

Size (µm)	Cyclone Underflow Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)	Cyclone Overflow Weight (g)	Weight%	Cum. % Passing	Units	Dist. (%)
9600	12	0.5	99.5	0.15	0.04					
6300	28	1.3	98.2	0.22	0.1					
4000	36	1.6	96.6	0.25	0.2					
2000	59	2.7	93.9	0.53	0.7					
1000	84	3.8	90.1	0.59	1.1					
500	164	7.5	82.5	0.55	2.0					
300	254	11.6	70.9	0.61	3.4	1	0.2	99.8	0.81	0.3
212	264	12.1	58.9	0.76	4.4	10	1.8	98.0	6.68	2.5
150	310	14.2	44.7	1.56	10.7	43	7.7	90.3	8.30	3.1
106	278	12.7	32.0	2.01	12.4	53	9.4	80.9	9.81	3.7
75	260	11.9	20.1	2.10	12.1	54	9.7	71.3	17.4	6.6
53	149	6.8	13.3	3.66	12.1	52	9.4	61.9	24.7	9.4
38	72	3.3	9.9	6.96	11.2	54	9.8	52.1	33.8	12.8
-38	217	9.9		6.13	29.5	291	52.1		162.0	61.5
Total	2186	100.0		2.06	100.0	558	100.0		263.4	100.0

Snip Jig Tails and Concentrate PS2

Size (µm)	Jig Tails Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)	Jig Concentrate Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)
11800	7	0.4	99.6	0.06	0.01					
9600	19	1.0	98.6	0.06	0.03					
6300	24	1.3	97.4	0.22	0.1					
4000	25	1.3	96.0	0.25	0.1					
2000	40	2.1	93.9	0.34	0.3					
1000	57	3.0	90.9	0.51	0.7	2	0.2	99.8	1.83	0.1
500	113	6.0	84.9	0.74	1.9	39	6.3	93.5	1.83	2.8
300	179	9.5	75.4	0.69	2.8	88	14.0	79.5	1.22	4.2
212	197	10.4	65.0	0.74	3.4	103	16.5	63.0	1.62	6.5
150	249	13.1	51.9	1.42	8.1	132	21.1	41.9	2.30	11.9
105	224	11.8	40.1	2.43	12.4	119	19.0	22.9	3.98	18.5
75	215	11.3	28.7	3.31	16.3	84	13.5	9.4	5.58	18.4
53	140	7.4	21.4	4.33	13.9	34	5.5	4.0	10.5	14.1
38	82	4.3	17.0	6.78	12.7	10	1.6	2.3	28.4	11.4
-38	323	17.0		3.68	27.2	15	2.3		21.4	12.2
Total	1894	100.0		2.31	100.0	626	100.0		4.10	100.0

Snip Table Tails and Concentrate PS2

Size (µm)	Table Tails Weight (g)	Weight%	Cum. % Passing	Grade (oz/st)	Dist. (%)	Table Concentrate Grade (oz/st)
1000	2	0.3	99.7	1.79	0.2	2.25
500	48	6.7	93.0	1.79	4.6	2.25
300	99	13.8	79.2	1.84	9.7	1.58
212	112	15.5	63.7	1.90	11.3	1.54
150	146	20.3	43.4	2.32	18.0	5.17
106	130	18.0	25.4	2.48	17.1	41.1
75	100	13.9	11.5	2.98	15.8	45.0
53	46	6.4	5.1	3.47	8.5	61.6
38	15	2.0	3.1	7.83	6.1	66.0
-38	22	3.1		7.30	8.7	54.5
Total	720	100.0		2.61	100.0	36.8

APPENDIX C

CIRCUIT MASS BALANCES

Agnico-Eagle Cyclone Classification

Residual sum of squares: 37.91801

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 CF	100.00	100.0	100.0	0.0	0.0
2 CU	84.22	80.0	84.2	20.0	4.2
3 CO	15.78		15.8		

Stream	Relative Solids Flowrate
1 CF	100.00
2 CU	84.22
3 CO	15.78

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
CF	2.160	2.307	1.000	0.147	100
CU	2.810	2.686	1.000	-0.124	98
CO	0.280	0.280	0.014	-0.000	2

Fractional Size Distribution Data

Size	CF Meas	CF Calc	CF SD.	CF Adj.	CU Meas	CU Calc	CU SD.	CU Adj.
210 micron	6.60	6.43	0.5	-0.2	7.10	7.24	0.5	0.1
150 micron	14.40	14.63	0.5	0.2	16.50	16.31	0.5	-0.2
105 micron	19.70	20.05	0.5	0.3	22.70	22.41	0.5	-0.3
75 microns	26.40	26.94	0.5	0.5*	30.40	29.94	0.5	-0.5
53 microns	13.70	13.67	0.5	-0.0	13.40	13.42	0.5	0.0
37 microns	6.40	6.42	0.5	0.0	4.50	4.48	0.5	-0.0
25 microns	3.80	4.64	0.5	0.8*	1.90	1.19	0.5	-0.7*

Size	CO Meas	CO Calc	CO SD.	CO Adj.
210 micron	2.10	2.13	0.5	0.0
150 micron	5.70	5.66	0.5	-0.0
105 micron	7.50	7.45	0.5	-0.1
75 microns	11.00	10.91	0.5	-0.1
53 microns	15.00	15.00	0.5	0.0
37 microns	16.80	16.80	0.5	-0.0
25 microns	23.20	23.07	0.5	-0.1

Assays of size fractions for CF

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
210 micron	0.030	0.031	0.006	0.001	100
150 micron	1.510	1.499	0.302	-0.011	100
105 micron	1.570	1.583	0.314	0.013	100
75 microns	1.660	1.737	0.332	0.077	100
53 microns	2.940	3.038	0.588	0.098	100
37 microns	6.610	6.742	2.000	0.132	100
25 microns	6.410	4.359	2.000	-2.051*	100
PAN	1.450	1.718	0.290	0.268	100

Assays of size fractions for CU

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
210 micron	1.120	0.032	0.224	-1.088*	99
150 micron	1.580	1.591	0.316	0.011	100
105 micron	1.690	1.676	0.338	-0.014	100
75 microns	1.950	1.850	0.390	-0.100	100
53 microns	3.780	3.646	0.756	-0.134	99
37 microns	11.760	11.275	5.000	-0.485	98
25 microns	15.590	18.358	5.000	2.768	91
PAN	4.540	2.675	1.000	-1.865*	91

Assays of size fractions for CO

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
210 micron	0.005	0.005	0.001	-0.000	1
150 micron	0.080	0.080	0.016	0.000	0
105 micron	0.080	0.080	0.016	-0.000	0
75 microns	0.090	0.090	0.018	-0.000	0
53 microns	0.140	0.140	0.028	-0.000	1
37 microns	0.290	0.290	0.058	-0.000	2
25 microns	0.500	0.504	0.100	0.004	9
PAN	0.380	0.372	0.076	-0.008	9

Snip Cyclone Classification

Residual sum of squares: 195.278

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 PCF	68.53	100.0	100.0	0.0	0.0
2 PCU	64.92	80.0	87.0	20.0	7.0
3 PCO	3.62		13.0		

Stream	Relative Solids Flowrate	Weight Percent Solids Meas	Calc	S.D.	Adjust
1 PCF	100.00	62.0	68.5	5.0	6.5*
2 PCU	94.72	80.0	74.6	5.0	-5.4*
3 PCO	5.28	30.0	27.7	5.0	-2.3

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
PCF	1.970	2.325	1.000	0.355	100
PCU	2.770	2.434	1.000	-0.336	99
PCO	0.370	0.370	0.019	-0.000	1

Fractional Size Distribution Data

Size	PCF Meas	PCF Calc	PCF SD.	PCF Adj.	PCU Meas	PCU Calc	PCU SD.	PCU Adj.
600 micron	3.06	3.35	0.5	0.3	3.82	3.54	0.5	-0.3
420 micron	4.49	4.89	0.5	0.4	5.54	5.16	0.5	-0.4
300 micron	8.35	9.24	0.5	0.9*	10.60	9.76	0.5	-0.8*
212 micron	11.90	12.89	0.5	1.0*	14.40	13.46	0.5	-0.9*
150 micron	16.23	16.45	0.5	0.2	17.08	16.87	0.5	-0.2
105 micron	13.82	13.37	0.5	-0.4	13.15	13.57	0.5	0.4
75 microns	23.33	19.49	0.5	-3.8*	16.31	19.94	0.5	3.6*
53 microns	4.15	5.90	0.5	1.8*	7.42	5.76	0.5	-1.7*
37 microns	6.07	5.06	0.5	-1.0*	3.83	4.79	0.5	1.0*
25 microns	6.41	5.90	0.5	-0.5*	5.35	5.83	0.5	0.5

Size	PCO Meas	Calc	SD.	Adj.
600 micron	0.00	-0.00	0.1	-0.0
420 micron	0.00	-0.00	0.1	-0.0
300 micron	0.00	-0.00	0.1	-0.0
212 micron	2.57	2.52	0.5	-0.1
150 micron	8.79	8.78	0.5	-0.0
105 micron	9.78	9.80	0.5	0.0
75 microns	11.20	11.40	0.5	0.2
53 microns	8.61	8.52	0.5	-0.1
37 microns	9.96	10.01	0.5	0.1
25 microns	7.10	7.13	0.5	0.0

Assays of size fractions for PCF

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.170	0.205	0.034	0.035*	100
420 micron	0.620	0.465	0.124	-0.155*	100
300 micron	0.380	0.375	0.076	-0.005	100
212 micron	0.480	0.526	0.096	0.046	100
150 micron	0.870	0.981	0.174	0.111	100
105 micron	1.380	1.574	0.276	0.194	100
75 microns	1.650	1.923	0.330	0.273	100
53 microns	4.810	4.304	0.962	-0.506	100
37 microns	7.900	9.506	1.580	1.606*	100
25 microns	4.460	5.267	0.892	0.807	100
PAN	8.010	4.653	1.602	-3.357*	100

Assays of size fractions for PCU

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.420	0.205	0.084	-0.215*	100
420 micron	0.400	0.465	0.080	0.065	100
300 micron	0.370	0.375	0.074	0.005	100
212 micron	0.600	0.529	0.120	-0.071	100
150 micron	1.220	1.007	0.244	-0.213	100
105 micron	2.040	1.632	0.408	-0.408	100
75 microns	2.670	1.977	0.534	-0.693*	100
53 microns	4.260	4.627	0.852	0.367	99
37 microns	18.100	10.548	3.620	-7.552*	99
25 microns	8.020	5.577	1.604	-2.443*	99
PAN	10.200	12.158	2.040	1.958	94

Assays of size fractions for PCO

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.000	0.000	0.000	0.000	-0
420 micron	0.000	-0.000	0.000	-0.000	0
300 micron	0.000	-0.000	0.000	-0.000	0
212 micron	0.190	0.190	0.038	-0.000	0
150 micron	0.100	0.100	0.020	-0.000	0
105 micron	0.140	0.140	0.028	-0.000	0
75 microns	0.250	0.250	0.050	-0.000	0
53 microns	0.380	0.380	0.076	0.000	1
37 microns	0.570	0.569	0.114	-0.001	1
25 microns	0.710	0.709	0.142	-0.001	1
PAN	0.430	0.436	0.086	0.006	6

Grinding and Gravity Circuits Mass Balance Summary PS1

Feed rate= 22.5 wet t/h

Table concentrate produced at 10 kg/day

Circulating loads: solids=408.5%: Au=2060%

Stream	%Solids	Solids s.g.	Solids, dry t/r	Au. (oz/st)	Water, usgpm
BMF	97.7	2.91	22	1.07	2
BMD	75.1	2.95	113	4.27	164
JTL	69.2	3.02	112	4.15	219
PCU	78.6	3.19	90	5.01	107
JCO	19.7	3.75	1.0	17.9	18
TTL	12.8	3.35	1.0	8.23	30
PCO	37.2	2.82	22	0.63	163

Grinding and Gravity Circuits Mass Balance Summary PS2

Feed rate= 20.3 wet t/h

Table concentrate produced at 9.44 kg at 44%

Circulating loads: solids=481.6%: Au=2081%

Stream	%Solids	Solids s.g.	Solids, dry t/h	Au. (oz/st)
BMF	97.9	2.91	20	0.62
BMD	-	-	-	2.39
JTL	69.4	3.02	112	2.32
PCU	76.4	3.19	96	2.70
JCO	23.2	3.75	1.8	4.18
TTL	9.49	3.35	1.8	2.58
PCO	39.1	2.82	20	0.48

Snip Operation Jig Circuit T1 Mass Balance

Residual sum of squares: 10.90972

Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1	FEED	125.00	125.0	125.0	12.5	0.0
2	HUTCH1	0.39	0.4	0.4	0.0	0.0
3	HUTCH2	0.05	0.0	0.0	0.0	0.0
4	TAIL	124.56		124.6		

	Stream	Relative Solids Flowrate
1	FEED	100.00
2	HUTCH1	0.31
3	HUTCH2	0.04
4	TAIL	99.65

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
FEED	2.260	2.149	1.000	0.111	100
HUTCH1	10.910	10.944	10.000	0.034	2
HUTCH2	55.610	55.628	20.000	0.018	1
TAIL	1.990	2.100	1.000	0.110	97

Fractional Size Distribution Data

Size	FEED Meas	Calc	SD.	Adj.	HUTCH1 Meas	Calc	SD.	Adj.
840 micron	21.36	20.59	0.5	0.8*	2.60	2.60	0.5	0.0
600 micron	4.36	4.49	0.5	0.1	5.88	5.88	0.5	0.0
420 micron	6.45	6.38	0.5	0.1	10.31	10.32	0.5	0.0
300 micron	10.57	10.70	0.5	0.1	17.92	17.92	0.5	0.0
210 micron	11.34	11.30	0.5	0.0	18.25	18.25	0.5	0.0
150 micron	12.35	12.27	0.5	0.1	20.31	20.31	0.5	0.0
105 micron	8.46	8.55	0.5	0.1	13.04	13.04	0.5	0.0
75 microns	6.53	7.04	0.5	0.5*	8.05	8.04	0.5	0.0
53 microns	4.23	3.77	0.5	0.5	2.54	2.54	0.5	0.0
37 microns	4.12	4.25	0.5	0.1	0.85	0.85	0.4	0.0
25 microns	1.71	1.75	0.5	0.0	0.15	0.15	0.1	0.0

	HUTCH2				TAIL			
Size	Meas	Calc	SD.	Adj.	Meas	Calc	SD.	Adj.
840 micron	1.61	1.61	0.5	0.0	19.88	20.65	0.5	0.8*
600 micron	3.29	3.29	0.5	0.0	4.61	4.48	0.5	0.1
420 micron	6.96	6.96	0.5	0.0	6.29	6.36	0.5	0.1
300 micron	13.36	13.36	0.5	0.0	10.81	10.68	0.5	0.1
210 micron	16.75	16.75	0.5	0.0	11.24	11.28	0.5	0.0
150 micron	24.54	24.54	0.5	0.0	12.17	12.24	0.5	0.1
105 micron	19.68	19.68	0.5	0.0	8.61	8.53	0.5	0.1
75 microns	11.10	11.10	0.5	0.0	7.54	7.04	0.5	0.5*
53 microns	1.85	1.85	0.5	0.0	3.31	3.77	0.5	0.5
37 microns	0.32	0.32	0.2	0.0	4.39	4.26	0.5	0.1
25 microns	0.12	0.12	0.1	0.0	1.79	1.75	0.5	0.0

Assays of size fractions for FEED

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.680	0.686	1.000	0.006	100
420 micron	0.830	0.734	1.000	0.096	100
300 micron	1.030	1.023	1.000	0.007	100
210 micron	1.320	1.149	1.000	0.171	100
150 micron	2.320	2.120	1.000	0.200	100
105 micron	4.440	3.989	1.000	0.451	100
75 microns	4.860	4.640	1.000	0.220	100
53 microns	4.690	5.108	1.000	0.418	100
37 microns	3.230	2.867	1.000	0.363	100
25 microns	2.630	2.619	1.000	0.011	100
PAN	0.920	0.992	1.000	0.072	20

Assays of size fractions for HUTCH1

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	4.050	4.049	5.000	0.001	2
420 micron	3.170	3.182	5.000	0.012	2
300 micron	4.660	4.661	5.000	0.001	2
210 micron	6.690	6.711	5.000	0.021	3
150 micron	9.540	9.566	5.000	0.026	2
105 micron	16.950	17.163	10.000	0.213	2
75 microns	30.090	30.168	10.000	0.078	2
53 microns	32.250	32.162	10.000	0.088	1
37 microns	51.490	51.512	10.000	0.022	1
25 microns	77.590	77.591	20.000	0.001	1
PAN	70.490	70.489	20.000	0.001	0

Assays of size fractions for HUTCH2

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	25.890	25.890	10.000	0.000	1
420 micron	15.110	15.114	10.000	0.004	1
300 micron	33.460	33.460	10.000	0.000	2
210 micron	36.660	36.670	10.000	0.010	2
150 micron	42.710	42.726	10.000	0.016	2
105 micron	56.450	56.491	10.000	0.041	1
75 microns	99.330	99.385	20.000	0.055	1
53 microns	266.600	266.526	30.000	0.074	1
37 microns	1133.500	1133.611	100.000	0.111	1
25 microns	669.700	669.703	100.000	0.003	1
PAN	46.260	46.260	10.000	0.000	0

Assays of size fractions for TAIL

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.670	0.664	1.000	0.006	96
420 micron	0.620	0.715	1.000	0.095	97
300 micron	0.980	0.987	1.000	0.007	96
210 micron	0.930	1.100	1.000	0.170	95
150 micron	1.850	2.049	1.000	0.199	96
105 micron	3.430	3.878	1.000	0.448	97
75 microns	4.270	4.489	1.000	0.219	96
53 microns	6.670	5.000	2.000	1.670	98
37 microns	2.440	2.803	1.000	0.363	98
25 microns	2.570	2.581	1.000	0.011	99
PAN	1.060	0.988	1.000	0.072	20

Snip Operation Jig Circuit T2 Mass Balance

Residual sum of squares: 22.07773

Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1	FEED	125.00	125.0	125.0	12.5	0.0
2	HUTCH1	0.51	0.5	0.5	0.1	0.0
3	HUTCH2	0.19	0.2	0.2	0.0	0.0
4	TAIL	124.30		124.3		

	Stream	Relative Solids Flowrate
1	FEED	100.00
2	HUTCH1	0.41
3	HUTCH2	0.16
4	TAIL	99.44

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
FEED	4.180	4.173	1.000	0.007	100
HUTCH1	16.250	16.251	5.000	0.001	2
HUTCH2	39.780	39.781	10.000	0.001	1
TAIL	4.060	4.067	1.000	0.007	97

Fractional Size Distribution Data

Size	FEED Meas	Calc	SD.	Adj.	HUTCH1 Meas	Calc	SD.	Adj.
840 micron	15.38	15.50	0.5	0.1	1.52	1.52	0.5	0.0
600 micron	4.59	4.26	0.5	0.3	5.35	5.35	0.5	0.0
420 micron	6.63	6.45	0.5	0.2	9.15	9.15	0.5	0.0
300 micron	10.31	10.30	0.5	0.0	15.43	15.43	0.5	0.0
210 micron	11.03	11.41	0.5	0.4	15.76	15.76	0.5	0.0
150 micron	12.84	13.05	0.5	0.2	18.75	18.75	0.5	0.0
105 micron	10.33	10.31	0.5	0.0	15.61	15.61	0.5	0.0
75 microns	10.47	10.44	0.5	0.0	13.25	13.25	0.5	0.0
53 microns	5.21	5.02	0.5	0.2	3.61	3.61	0.5	0.0
37 microns	4.08	4.51	0.5	0.4	1.11	1.11	0.5	0.0
25 microns	2.72	2.18	0.5	0.5*	0.26	0.26	0.2	0.0

Size	HUTCH2 Meas	HUTCH2 Calc	HUTCH2 SD.	HUTCH2 Adj.	TAIL Meas	TAIL Calc	TAIL SD.	TAIL Adj.
840 micron	0.81	0.81	0.4	0.0	15.69	15.58	0.5	0.1
600 micron	3.46	3.46	0.5	0.0	3.92	4.26	0.5	0.3
420 micron	6.68	6.69	0.5	0.0	6.25	6.43	0.5	0.2
300 micron	13.21	13.21	0.5	0.0	10.26	10.28	0.5	0.0
210 micron	15.96	15.96	0.5	0.0	11.75	11.38	0.5	0.4
150 micron	22.05	22.05	0.5	0.0	13.22	13.01	0.5	0.2
105 micron	19.33	19.33	0.5	0.0	10.25	10.27	0.5	0.0
75 microns	14.49	14.49	0.5	0.0	10.40	10.43	0.5	0.0
53 microns	2.91	2.91	0.5	0.0	4.84	5.03	0.5	0.2
37 microns	0.65	0.65	0.3	0.0	4.96	4.53	0.5	0.4
25 microns	0.24	0.24	0.2	0.0	1.65	2.19	0.5	0.5*

Assays of size fractions for FEED

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	1.040	1.080	0.208	0.040	100
420 micron	1.200	1.082	0.240	0.118	100
300 micron	1.330	1.350	0.266	0.020	100
210 micron	2.270	1.984	1.000	0.286	100
150 micron	3.220	2.972	1.000	0.248	100
105 micron	6.020	6.183	1.000	0.163	100
75 microns	9.120	9.125	1.000	0.005	100
53 microns	9.140	10.965	1.000	1.825*	100
37 microns	8.210	6.380	1.000	1.830*	100
25 microns	4.000	4.411	1.000	0.411	100
PAN	1.790	1.892	1.000	0.102	33

Assays of size fractions for HUTCH1

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	7.250	7.132	5.000	0.118	3
420 micron	5.457	5.753	5.000	0.296	3
300 micron	7.510	7.466	5.000	0.044	3
210 micron	8.700	8.740	5.000	0.040	2
150 micron	12.790	12.826	5.000	0.036	3
105 micron	21.090	20.990	10.000	0.100	2
75 microns	33.080	33.078	10.000	0.002	2
53 microns	43.790	43.257	10.000	0.533	1
37 microns	70.160	70.892	20.000	0.732	1
25 microns	37.050	37.030	10.000	0.020	0
PAN	24.410	24.409	10.000	0.001	0

Assays of size fractions for HUTCH2

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	29.760	29.643	10.000	0.117	3
420 micron	26.800	27.132	10.000	0.332	4
300 micron	22.100	22.043	10.000	0.057	3
210 micron	23.140	23.202	10.000	0.062	3
150 micron	24.970	25.035	10.000	0.065	2
105 micron	48.020	47.972	10.000	0.048	2
75 microns	75.870	75.866	20.000	0.004	2
53 microns	132.620	131.138	30.000	1.482	1
37 microns	38.210	38.251	10.000	0.041	0
25 microns	39.400	39.393	10.000	0.007	0
PAN	21.930	21.930	10.000	0.000	0

Assays of size fractions for TAIL

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	1.930	1.013	1.000	0.917	93
420 micron	0.940	1.012	0.188	0.072	93
300 micron	1.290	1.271	0.258	0.019	93
210 micron	1.860	1.899	0.372	0.039	95
150 micron	2.610	2.856	1.000	0.246	95
105 micron	6.130	5.968	1.000	0.162	96
75 microns	8.860	8.855	1.000	0.005	96
53 microns	12.580	10.762	1.000	1.818*	98
37 microns	4.480	6.308	1.000	1.828*	99
25 microns	4.800	4.389	1.000	0.411	99
PAN	1.990	1.888	1.000	0.102	33

Snip Operation Jig Circuit T3 Mass Balance

Residual sum of squares: 42.53944

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 FEED	125.00	125.0	125.0	12.5	0.0
2 HUTCH1	0.13	0.1	0.1	0.0	0.0
3 HUTCH2	0.20	0.2	0.2	0.0	0.0
4 TAIL	124.67		124.7		

Stream	Relative Solids Flowrate
1 FEED	100.00
2 HUTCH1	0.10
3 HUTCH2	0.16
4 TAIL	99.74

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
FEED	4.330	4.230	1.000	0.100	100
HUTCH1	33.660	33.663	5.000	0.003	1
HUTCH2	48.320	48.324	5.000	0.004	2
TAIL	4.030	4.129	1.000	0.099	97

Fractional Size Distribution Data

Size	FEED Meas	FEED Calc	FEED SD.	FEED Adj.	HUTCH1 Meas	HUTCH1 Calc	HUTCH1 SD.	HUTCH1 Adj.
840 micron	17.29	18.44	0.5	1.2*	1.13	1.13	0.5	0.0
600 micron	4.39	4.23	0.5	0.2	4.98	4.98	0.5	0.0
420 micron	6.14	5.97	0.5	0.2	8.68	8.68	0.5	0.0
300 micron	9.40	9.47	0.5	0.1	15.98	15.98	0.5	0.0
210 micron	12.80	12.06	0.5	0.7*	18.31	18.31	0.5	0.0
150 micron	10.73	11.35	0.5	0.6*	22.25	22.24	0.5	0.0
105 micron	9.74	9.44	0.5	0.3	16.53	16.53	0.5	0.0
75 microns	10.13	9.62	0.5	0.5*	9.95	9.95	0.5	0.0
53 microns	4.88	4.65	0.5	0.2	1.59	1.59	0.5	0.0
37 microns	3.41	4.06	0.5	0.7*	0.35	0.35	0.2	0.0
25 microns	2.65	2.24	0.5	0.4	0.13	0.13	0.1	0.0

Size	HUTCH2 Meas	Calc	SD.	Adj.	TAIL Meas	Calc	SD.	Adj.
840 micron	0.19	0.19	0.1	0.0	19.64	18.49	0.5	1.1*
600 micron	0.59	0.59	0.3	0.0	4.07	4.23	0.5	0.2
420 micron	1.98	1.98	0.5	0.0	5.80	5.97	0.5	0.2
300 micron	6.07	6.07	0.5	0.0	9.53	9.46	0.5	0.1
210 micron	11.34	11.34	0.5	0.0	11.31	12.05	0.5	0.7*
150 micron	23.02	23.02	0.5	0.0	11.95	11.32	0.5	0.6*
105 micron	25.97	25.97	0.5	0.0	9.10	9.41	0.5	0.3
75 microns	23.68	23.68	0.5	0.0	9.09	9.60	0.5	0.5*
53 microns	5.57	5.57	0.5	0.0	4.43	4.66	0.5	0.2
37 microns	1.26	1.26	0.5	0.0	4.72	4.07	0.5	0.7*
25 microns	0.18	0.18	0.1	0.0	1.84	2.25	0.5	0.4

Assays of size fractions for FEED

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	1.030	0.808	1.000	0.222	100
420 micron	0.940	0.919	0.188	0.021	100
300 micron	1.470	1.318	0.294	0.152	100
210 micron	2.030	1.790	1.000	0.240	100
150 micron	3.910	3.654	1.000	0.256	100
105 micron	6.920	7.309	1.000	0.389	100
75 microns	9.660	9.508	2.000	0.152	100
53 microns	8.450	9.699	1.000	1.249*	100
37 microns	9.450	7.023	1.000	2.427*	100
25 microns	4.340	4.232	1.000	0.108	100
PAN	1.830	1.810	0.366	0.020	26

Assays of size fractions for HUTCH1

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	9.330	9.337	5.000	0.007	1
420 micron	9.930	9.953	5.000	0.023	2
300 micron	12.930	13.007	5.000	0.077	2
210 micron	19.150	19.159	5.000	0.009	2
150 micron	31.040	31.092	10.000	0.052	2
105 micron	49.510	49.439	10.000	0.071	1
75 microns	79.560	79.576	20.000	0.016	1
53 microns	151.390	150.281	50.000	1.109	1
37 microns	250.850	251.387	50.000	0.537	0
25 microns	55.310	55.311	10.000	0.001	0
PAN	20.910	20.910	10.000	0.000	0

Assays of size fractions for HUTCH2

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	9.330	9.331	5.000	0.001	0
420 micron	12.900	12.908	5.000	0.008	1
300 micron	9.740	9.785	5.000	0.045	1
210 micron	4.650	4.659	5.000	0.009	0
150 micron	4.790	4.811	5.000	0.021	0
105 micron	13.960	13.790	10.000	0.170	1
75 microns	27.910	27.925	10.000	0.015	1
53 microns	23.320	23.082	10.000	0.238	0
37 microns	17.020	17.140	10.000	0.120	0
25 microns	5.840	5.840	1.000	0.000	0
PAN	2.620	2.620	0.524	0.000	0

Assays of size fractions for TAIL

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.790	0.796	0.158	0.006	98
420 micron	0.880	0.899	0.176	0.019	98
300 micron	1.190	1.289	0.238	0.099	98
210 micron	1.730	1.759	0.346	0.029	98
150 micron	3.340	3.594	1.000	0.254	98
105 micron	7.590	7.203	1.000	0.387	98
75 microns	9.020	9.360	3.000	0.340	98
53 microns	10.870	9.624	1.000	1.246*	99
37 microns	4.570	6.996	1.000	2.426*	100
25 microns	4.120	4.228	1.000	0.108	100
PAN	1.790	1.809	0.358	0.019	26

Snip Operation Jig Circuit T4 Mass Balance

Residual sum of squares: 27.46534 Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 FEED	125.00	125.0	125.0	12.5	0.0
2 HUTCH1	0.16	0.2	0.2	0.0	0.0
3 HUTCH2	0.10	0.1	0.1	0.0	0.0
4 TAIL	124.74		124.7		

Stream	Relative Solids Flowrate
1 FEED	100.00
2 HUTCH1	0.13
3 HUTCH2	0.08
4 TAIL	99.79

Assay Data

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
FEED	2.490	2.483	1.000	0.007	100
HUTCH1	18.770	18.770	5.000	0.000	1
HUTCH2	33.490	33.490	5.000	0.000	1
TAIL	2.430	2.437	1.000	0.007	98

Fractional Size Distribution Data

	FEED				HUTCH1			
Size	Meas	Calc	SD.	Adj.	Meas	Calc	SD.	Adj.
840 micron	19.07	20.22	0.5	1.2*	0.85	0.85	0.4	0.0
600 micron	3.97	4.14	0.5	0.2	3.89	3.89	0.5	0.0
420 micron	6.09	6.15	0.5	0.1	7.89	7.89	0.5	0.0
300 micron	10.42	10.37	0.5	0.1	15.23	15.23	0.5	0.0
210 micron	11.45	11.61	0.5	0.2	18.96	18.96	0.5	0.0
150 micron	12.53	12.60	0.5	0.1	23.66	23.66	0.5	0.0
105 micron	9.79	9.49	0.5	0.3	17.20	17.20	0.5	0.0
75 microns	8.86	8.70	0.5	0.2	9.77	9.77	0.5	0.0
53 microns	3.84	3.75	0.5	0.1	1.76	1.76	0.5	0.0
37 microns	3.66	3.79	0.5	0.1	0.50	0.50	0.3	0.0
25 microns	1.59	1.55	0.5	0.0	0.14	0.14	0.1	0.0

Size	HUTCH2 Meas	Calc	SD.	Adj.	TAIL Meas	Calc	SD.	Adj.
840 micron	0.59	0.59	0.3	0.0	21.41	20.26	0.5	1.1*
600 micron	1.36	1.36	0.5	0.0	4.32	4.15	0.5	0.2
420 micron	3.00	3.00	0.5	0.0	6.22	6.15	0.5	0.1
300 micron	8.57	8.57	0.5	0.0	10.32	10.37	0.5	0.1
210 micron	14.12	14.12	0.5	0.0	11.77	11.60	0.5	0.2
150 micron	24.36	24.36	0.5	0.0	12.65	12.57	0.5	0.1
105 micron	24.23	24.23	0.5	0.0	9.17	9.47	0.5	0.3
75 microns	18.08	18.08	0.5	0.0	8.53	8.69	0.5	0.2
53 microns	3.83	3.83	0.5	0.0	3.67	3.75	0.5	0.1
37 microns	1.32	1.32	0.5	0.0	3.94	3.80	0.5	0.1
25 microns	0.29	0.29	0.2	0.0	1.52	1.56	0.5	0.0

Assays of size fractions for FEED

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.390	0.421	0.078	0.031	100
420 micron	0.680	0.641	0.136	0.039	100
300 micron	0.720	0.746	0.144	0.026	100
210 micron	1.090	0.944	0.218	0.146	100
150 micron	1.690	1.826	0.338	0.136	100
105 micron	4.140	3.244	0.828	0.896*	100
75 microns	5.020	4.915	1.000	0.105	100
53 microns	5.960	7.683	1.000	1.723*	100
37 microns	7.290	6.074	1.000	1.216*	100
25 microns	4.670	5.144	1.000	0.474	100
PAN	1.490	1.524	0.298	0.034	20

Assays of size fractions for HUTCH1

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	9.980	9.830	5.000	0.150	3
420 micron	6.960	6.963	1.000	0.003	2
300 micron	7.320	7.318	1.000	0.002	2
210 micron	7.350	7.356	1.000	0.006	2
150 micron	12.820	12.817	1.000	0.003	2
105 micron	23.120	23.419	10.000	0.299	2
75 microns	53.140	53.155	10.000	0.015	2
53 microns	109.710	108.794	30.000	0.916	1
37 microns	195.220	195.401	30.000	0.181	1
25 microns	48.950	48.945	10.000	0.005	0
PAN	5.440	5.440	1.000	0.000	0

Assays of size fractions for HUTCH2

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	33.730	33.593	10.000	0.137	2
420 micron	27.130	27.216	10.000	0.086	2
300 micron	18.540	18.519	5.000	0.021	2
210 micron	17.860	17.937	5.000	0.077	2
150 micron	21.150	20.961	10.000	0.189	2
105 micron	29.310	29.585	10.000	0.275	2
75 microns	46.910	46.928	10.000	0.018	2
53 microns	96.270	95.690	20.000	0.580	1
37 microns	240.810	241.683	50.000	0.873	1
25 microns	29.420	29.413	10.000	0.007	0
PAN	88.510	88.506	20.000	0.004	0

Assays of size fractions for TAIL

Au (oz/st)	Meas.	Calc.	Std. Dev.	Adjustment	%Rec
600 micron	0.440	0.401	0.088	0.039	95
420 micron	0.590	0.620	0.118	0.030	97
300 micron	0.750	0.722	0.150	0.028	96
210 micron	0.830	0.914	0.166	0.084	96
150 micron	2.950	1.769	1.000	1.181*	97
105 micron	1.840	3.142	1.000	1.302*	96
75 microns	4.670	4.774	1.000	0.104	97
53 microns	9.270	7.549	1.000	1.721*	98
37 microns	4.760	5.975	1.000	1.215*	98
25 microns	5.610	5.136	1.000	0.474	100
PAN	1.900	1.521	1.000	0.379	20