TESTING A NEW GOLD CENTRIFUGAL CONCENTRATOR

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ABSTRACT

A new gold centrifugal concentrator, the Falcon SuperBowl, was tested both at plant and laboratory scales to assess its ability to recover gold from grinding circuits.

The performances of a 21-in SuperBowl (SB21) at Mineral Hill and New Britannia Mines showed that it could recover gravity recoverable gold (GRG) of all sizes, especially below 25 μm at Mineral Hill Mine. At Mineral Hill, stage recovery was higher, 41-66% gold and 56-82% GRG, but the extremely low feed rate to the SB21 limited plant recovery by gravity to 11-27%. At New Britannia, the SB21 was able to achieve over a one-week trial a stage recovery of 42% gold, whilst recovering 40% of the gold in the ore.

Three types (to test the effect of gangue density and size distribution) of synthetic feeds were used to characterize a laboratory 4-in SuperBowl (SB4) as a function of feed rate and fluidization water flow rate. The SB4 recovered more than 90% tungsten (used to mimic gold) with all the feeds under a wide range of fluidization water flow rate, up to a feed rate of 5 kg/min, the highest feed rate tested. Concentrate bed observation suggested that the SuperBowl operates mainly under non-fluidized conditions.

The grinding circuit surveys performed at the Mineral Hill and New Britannia Mines determined that the best stream for the gravity recovery was the primary cyclone underflow. The grades of grinding circuit streams varied widely and the primary cyclone concentrated most of the gold in its underflow with a highest GRG content.

A 20-in Knelson Concentrator and a shaking table were further tested in this program with the samples extracted from Casa Berardi and Mineral Hill Mine, respectively. The 30-in Knelson was tested at two different conditions to assess the impact of fluidization water flow rate and explore the importance of cycle time. Results showed

that a shorter cycle time and higher water flow would improve the Knelson performance. At Mineral Hill, the shaking table treating the SB21 concentrate could not recover gold below 105 μm effectively.

A standardized GRG test for the New Britannia ore indicated that 74.6% of gold is gravity recoverable. The comparison of a 21-in SuperBowl and 20-in Knelson at mine site showed that both recovered more than 50% of the GRG from the grinding circuit. The comparison was not totally decisive, as feed rate in neither unit was pushed to the maximum, at which point their economic impact is maximized.

RÉSUMÉ

Un nouveau séparateur centrifuge, le SuperBowl de Falcon, a été étudié en laboratoire et en usine, pour évaluer sa capacité à récupérer l'or des circuits de broyage.

Le SuperBowl de 21" (SB21) aux mines Mineral Hill et New Britannia pouvait récupérer l'or récupérable par gravimétrie (ORG) de toutes les fractions granulométriques, et tout particulièrement de la fraction -25 μm à Mineral Hill. A New Britannia, le SB21 a pu récupérer, sur une période d'une semaine, 40% de l'or contenu dans le minerai, et 42% de l'or qu'on lui alimenta. A Mineral Hill, la récupération unitaire du SB21 était de 41 à 66% en or et 56 à 82% en ORG, mais le taux d'alimentation au SB21 étant très faible, seulement de 11 à 27% de l'or présent dans le minerai fut récupéré.

Nous avons utilisé trois différentes alimentations synthétiques (pour évaluer l'effet de la densité et de la distribution granulométrique de la gangue) pour étudier l'effet du taux d'alimentation et du débit d'eau de fluidisation sur le fonctionnement d'un SuperBowl de laboratoire de 4" (SB4). Le SB4 a récupéré plus de 90% du tungstène (de densité égale à l'or) de tous les types d'alimentation sur une plage étendue de débit d'eau de fluidisation, jusqu'à un taux d'alimentation de 5 kg/min (le taux maximum utilisé). En observant le lit de concentré, nous avons conclu que le lit de concentré du SB4 était en grande partie non-fluidisé.

Les campagnes d'échantillonnage du circuit de broyage à Mineral et New Britannia ont démontré que le meilleur flot à cibler pour la récupération gravimétrique était la sous-verse primaire des cyclones (SPC). La teneur en or des différents flots variait de façon considérable, et c'était la SPC dont la concentration en or était la plus élevée, avec la proportion d'ORG également la plus élevée.

Nous avons également étudié un concentrateur Knelson de 30" (pour déterminer

l'effet du débit de fluidisation et de la durée du cycle de récupération) à la mine Casa Berardi et une table à secousse à la mine Mineral Hill. A Casa Berardi, nous avons conclu que le cycle de récupération du Knelson, deux heures, était trop long. A Mineral Hill, nous avons conclu que la table à secousse, que traitait le concentré d'un SB21, était incapable de récupérer tout l'or plus fin que 105 µm de façon efficace.

Un essai standard de caractérisation de l'ORG à la mine New Britannia a démontré que 75% de l'or du minerai était récupérable par gravimétrie. Et un concentrateur Knelson de 20", et un SB21 ont pu récupérer plus de 50% de cet ORG. Toutefois, une comparaison complète des deux appareils exigerait qu'on les pousse à leur capacité maximale, pour maximiser leur impact économique.

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B6	6 in batch Falcon Concentrator
B12	12 in batch Falcon Concentrator
BMD	ball mill discharge
C10	10 in continuous Falcon Concentrator
C20	20 in continuous Falcon Concentrator
C40	40 in continuous Falcon Concentrator
CD	centre discharge
Dist'n	distribution of gold
F_{80}	the particle size at which 80% of the mass passes
g/min	grams per minute
g/t	grams per tonne
GRG	gravity recoverable gold
Gs	times of gravity acceleration
HP	horse power
IFFC	injection flowing film centrifugation
KC	Knelson Concentrator
kg/min	kilogram per minute
L/min	litre per minute
LKC	Laboratory Knelson Concentrator
MLS	Mozley Laboratory Separator
MGS	Mozley Multi-Gravity Separator
O/A	overall
oz/st	ounce per short ton
PCOF	primary cyclone overflow

PCUF	primary cyclone underflow
Rec.	recovery of gold
RMD	rod mill discharge
RPM	rotations per minute
SAG	semi-autogenous
SAG DIS	SAG mill discharge
SB	SuperBowl
SB4	4 inch SuperBowl
SB21	21 inch SuperBowl
SCOF	secondary cyclone overflow
SCUF	secondary cyclone underflow
SLS	SL-type Separator
STF	shaking table feed
STT	shaking table tails
t/h	tonne per hour
T-I-F	test I feed
T-I-T	test I tail
T-II-F	test II feed
T-II-T	test II tail
T-III-F	test III feed
T-III-T	test III tail
T-IV-F	test IV feed
T-IV-T	test IV tail
T-V-F	test V feed
T-V-T	test V tail
USGPM	US gallon per minute
WSC	water saving cone
Wt.	weight

CHAPTER 1

INTRODUCTION

1.1 Background

Treatment methods for the recovery of gold from ores depend on the type of mineralization. Gold ores in which oxidation of the sulphides is essentially complete are best treated by cyanidation for the recovery of the free gold; gold ores which contain their major values as base metals, such as copper, lead and zinc, are generally treated by flotation; gold that is associated with pyrite and arsenopyrite, and usually with non-sulphide gangue minerals, is frequently treated with the combination of cyanidation and flotation (Mining Chemicals Handbook, 1989). However, no matter in which form gold exists, some is totally liberated in grinding circuits, and gravity concentration can therefore be incorporated in the recovery flowsheet.

1.1.1 Gold Behaviour in Grinding Circuits

In practice, most gold ore grinding circuits consist of two-stage grinding. Rod or SAG milling is used for primary grinding and ball mill for finer grinding. Usually Hydrocyclones are used as classifiers to separate particles fine enough for recovery from those requiring further grinding.

Because of its malleability and density, the behaviour of gold in grinding circuits is unusual and affects all important mechanisms: breakage, classification and liberation (Banisi et al, 1991). Laboratory studies of monosized gold and silica showed that gold produces fewer fines upon grinding: 75% of the mass reports to the next Tyler size class, as opposed to 45% for silica (Banisi, 1990). Gold, and particularly gravity recoverable gold (GRG), has a distinct behaviour in hydrocyclones, whereby more than 98 and even 99% of the GRG reports to the underflow stream, some of it very fine (<25 μm). As a result gold builds up to very high circulating loads, 2000-8000%, and often leaves the circuit only once it is overground (Banisi et al, 1991).

Grinding circuits surveys demonstrated that the gold was liberated in primary grinding circuits and concentrated in the primary cyclone underflow. At Casa Berardi, the grades of the primary and secondary underflow were respectively 114 and 44 g/t, with a feed grade of only 7 g/t, which represented gold circulating load of about 4200 and 1540 %, respectively (Woodcock, 1994). At Rosebery, gold was concentrated to 30 times the ore grade in the primary cyclone underflow, whereas only 7 times in the secondary cyclone underflow. The recovery of overground gold particles could be hindered by smearing, flattening, and tarnishing or passivation of the surfaces of liberated gold particles (Poulter et al, 1994). In extreme cases, gold may settle in the recovery circuit to be recovered only at mill shut-down. In all cases a significant inventory builds up, and constitutes a form of working capital, whose net present value can be very low, since it must be discounted from a future time corresponding to the cessation of milling activities. Therefore, there is a significant economic incentive to remove liberated gold from the grinding circuit as soon as possible to boost overall recovery and lower metallurgical and economic gold losses; and the primary cyclone underflow is arguably the best candidate for this purpose.

1.1.2 Gravity Concentrators Used to Recover Gold

Because of gold's very high circulating load, the primary gravity concentrator is usually put in the grinding circuit to treat part or all of the primary cyclone underflow to recover liberated gold. The primary gravity concentrate is then upgraded by a shaking table to obtain a final gold concentrate, which is directly smelted to produce bullion containing 90-98% gold plus silver (Huang, 1996).

Recovering gold from the circulating load of grinding circuits yields the following advantages: (i) the payment for gold bullion is more than 99% and is received almost immediately, while gold in flotation concentrate is only paid 92-95% three or four months later (Wells and Patel, 1991; Huang, 1996); (ii) gold overgrinding is reduced and the amount of gold locked up behind mill liners is minimized; (iii) the gold inventory in downstream processes is reduced; (iv) for cyanidation, fewer cyanidation stages or lower cyanide concentration can be used, and environmental pollution is decreased; (v) the overall gold recovery can be improved by reducing soluble losses and recovering large or slow leaching gold particles that would otherwise be incompletely leached (Loveday et al, 1982); (vi) for flotation, the risk of gold particles advancing to flotation that are too coarse to float is reduced; (vii) and overall gold recovery can also be increased by recovering gold smeared onto other particles or embedded by other particles (Banisi, 1990; Darnton et al, 1992; Ounpuu, 1992).

Before the early 1980's, conventional gravity separators such as sluice boxes, jigs and spirals were most commonly used to recover gold from grinding circuits. However, these separators can only recover medium size to coarse gold particles, and produce relatively low grade primary gravity concentrates, which need more upgrading. More recently, they have been replaced by a number of new gold centrifugal concentrators, such as Knelson and, to a lesser extent, Falcon SuperBowl. These centrifugal concentrators, especially the Knelson, have achieved worldwide acceptance, because of their remarkable ability to produce very high grade concentrates (e.g. both Knelson and SuperBowl achieved a concentration ratio above 200 at the New Britannia Mine) and recover gold over a wide range of particle size (even below 25 μm). Interestingly, the conventional separators have been replaced by Knelsons in some plants not only because of the latters' better performance, but also easier operation and maintenance, since conventional gravity circuits require significant operator attention to maintain recovery and produce quality concentrates (Laplante et al, 1998; Wells and Patel, 1991; Poulter et al, 1994).

Many applications also demonstrate that gold centrifugal concentrators can better the conventional separators in their metallurgical performance. At Boddington and Lac Minerals' Est Malartic Division, replacing jigs and spirals with 30-in Knelsons increased gold gravity recovery by about 30% (Hart and Hill, 1995; Hope et al, 1995). At Montana tunnels and St. Ives, a 30-in CD Knelson recovered respectively 14% and 37% gold from their two grinding circuits, which increased the overall gold recoveries by 2.7% and 1%, respectively (Darnton et al, 1992; Cloutt, 1995). At the New Britannia Mine, a 21-in SuperBowl and 30-in Knelson achieved very similar performance, respectively 36% and 41% overall gold recovery (this will be detailed in chapter 5).

1.2 Objectives of the Study

The objectives of this study are to (i) test a new gold centrifugal concentrator, the SuperBowl (SB4 and SB21), both at laboratory and plant scales; (ii) carry out grinding circuit surveys to determine the best stream for gravity recovery at the Mineral Hill and New Britannia Mines; (iii) evaluate the performance of a 30-in Knelson at Casa Berardi; (iv) test the efficiency of a shaking table as an upgrading unit at the Mineral Hill Mine; (v) characterize the gravity recoverable gold (GRG) in the New Britannia ore to provide a basis for analysing gravity circuit performance.

1.3 Thesis Structure

The thesis consists of seven chapters. This chapter introduces the background of this program, which includes gold's behaviour in grinding circuits and the application of gravity concentrators to recover gold. The objectives of the study and the thesis structure are also be presented here.

Chapter two describes the general principles of centrifugal separation. Several gravity centrifugal separators will be reviewed in their structure, separating mechanism, operating parameters and applications.

Chapter three analyses samples from Mineral Hill Mine in three parts: a 21-in plant SuperBowl is tested at five different operating conditions, shaking table performance is examined and a grinding circuit survey is performed to confirm the character of GRG in all sampled streams.

The evaluation of the Knelson performance at Casa Berardi is presented in chapter four. Two different fluidization water flows and cycle times are tested and the nature of GRG in the grinding circuit is further confirmed.

In chapter five, GRG in the New Britannia ore is first characterized; results of a grinding circuit survey are presented ad discussed. Comparative plant test work on the Knelson and SuperBowl is also presented in this chapter.

Chapter six presents the methodology and results of a test program for the SB4 model. Three types of synthetic ores are used to explore the effect of fluidizing water flow and feed rate.

General conclusions and suggestions for the future work are presented in chapter seven.

CHAPTER 2

GRAVITY CENTRIFUGAL SEPARATION

2.1 Introduction

Recovery of valuable minerals contained in fine particles is a difficult problem in mineral processing and particularly in gravity separation. With decreasing particle size (< 100 μm), the forces associated with the water flow become dominant over those associated with particle volume, in particular gravity (Traore et al, 1995). Even though jigs, sluices, cones and spirals have lower size limits extending somewhat below 100 μm, their primary purpose is not for this size range (Mills et al, 1979). Because of this, most of the valuable minerals contained in the fine particles prove to be irrecoverable with conventional methods of gravity separation. To solve this problem, a number of gravity separation methods and machines have been developed over the last few decades (Traore et al, 1995).

The development of the Bartles Mozley Concentrator and Bartles Crossbelt Concentrator in the 60s and 70s reduced the lower size of effective recovery to about 15 μm (Burt et al, 1995). The most significant achievements, however, were obtained recently by the use of centrifugal separation. The Mozley Multi-Gravity Separator (MGS), a new fine-particle gravity equipment, can recover cassiterite at sizes down to 10 μm (Tucker et al, 1991); the Knelson Concentrator and Falcon SuperBowl can recover the full size range gold, including the -25 μm fraction. Synthetic ore tests showed that a laboratory Falcon SB4 model can recover over 90% of tungsten in the 8-25 μm fraction from different types of feeds. A new continuous discharge centrifugal separator for separating fine and ultra-fine particles called SL-type Separator (SLS) can recover a minimum particle size of about 2 μm (possibly down to 1 μm) at centrifugal fields of 330-

660 Gs (Lu, 1994; Ren et al, 1994).

2.2 General Principles

Gravity concentration methods separate minerals of different specific gravity using their relative movement in response to gravity and one or more other forces, which often is the resistance to motion offered by a fluid, usually water or a slurry (Wills, 1997). Generally speaking, gravity processes depend on two actions: (i) stratification in a pulsed or moving dense bed usually with a wide size range feed as in jigs; (ii) film sizing with a thin flowing liquid film and usually, closely sized solids as in spirals, frames, and tilting concentrators. Tables and sluices use a combined action (Burt et al, 1985). A considerable number of theories exist for gravity concentrators and no one concept is adequate to explain the separation occurring in a given unit. Rather they suggest that several processes occur at different stages of the cycle, in different parts of the device, over different size ranges and under differing operating conditions (Kelly et al, 1982; Gaudin, 1939; Pryor, 1965; Burt, 1984; Wills, 1997; Sun, 1982).

Although factors such as density difference, particle size, particle shape of both the wanted and the unwanted mineral or minerals will affect the separation, centrifugal forces seem to have a most significant effect on the separation, even in "non-centrifuge" units (e.g. sluice, spiral, table). In this chapter, attention is focused on the operation of the centrifuge gravity concentrators.

2.2.1 Centrifugal Force Strength

Centrifugal fields can be generated in two different ways:(i) by introducing a fluid with a high tangential velocity into a cylindrical or conical vessel as in the hydrocyclone; (ii) by rotating all or part of the unit. In this case the fluid is introduced into some form of rotating bowl and is rapidly accelerated. Because the frictional drag within the fluid ensures that there is very little rotational slip or relative motion between fluid layers within the bowl, all the fluid tends to rotate at a constant angular velocity ω (Coulson and Richardson, 1990).

For a particle in a centrifuge bowl which is rotating at an angular velocity of ω, the centrifugal acceleration is $r\omega^2$, where r is the radius of rotation, compared with the gravitational acceleration of g. The ratio $r\omega^2/g$, or number of Gs, is one measure of the separating effect obtained in a centrifuge relative to that arising from the gravitational field, which is called centrifugal force strength (Sun, 1982). This value may be very high (up to 10^4) in some industrial centrifuges and more than an order of magnitude greater in the ultracentrifuges, while in mineral processing practice, it is as high as 1500 (Coulson and Richardson, 1990; Lu, 1994).

2.2.2 Motion of Particles in Centrifugal Field

Coulson and Richardson (1990) described the motion of particles in a fluid in details. Most of this section is based on their book.

In most practical cases, when a particle is moving in a centrifugal fluid, gravitational effects will be comparatively small and can be neglected. The equation for particle motion in a centrifugal field will be similar to that for motion in the gravitational field, except that the gravitational acceleration 'g' must be replaced by the centrifugal acceleration $r\omega^2$:

$$F_c = \frac{\pi}{6} d^3 (\delta - \rho) r \omega^2 \tag{2-1}$$

where F_c is the centrifugal force, d particle size, δ and ρ the specific density of the particle and fluid, respectively.

In the case of centrifuges used for separating fine solids from a suspension in a liquid, it is possible to consider only the Stokes' law region in calculating the drag between the particle and the liquid and neglect other factors. However, in the mineral industry, centrifuges are used to treat high density slurries, mineral particles are then subjected not only to centrifugal and drag forces, but also to forces such as buoyancy and momentum transfer resulting from inter-particle collisions. For example, the movement of a particle in a Knelson Concentrator can be decomposed into radial, tangential and axial components (Ling and Laplante, 1997). Therefore, the calculation of the motion of a particle under this situation is quite complex. To simplify, the following equations are all presented based on the Stokes' law region, neglecting forces other than the centrifuge and drag forces. The drag force F_d on the particle can then be given by:

$$F_d = 3\,\pi\,\mu\,d\,u \tag{2-2}$$

where F_d is inward drag force, μ is the viscosity of the fluid medium, and u is the particle velocity with respect to the fluid. As centrifuge acceleration is a function of the position r of the particle, for a spherical particle in a fluid, the equation of motion for the Stokes' law region is equal to:

$$F_c - F_d = \frac{\pi}{6}\,d^3\,\delta\,\left(\frac{d^2r}{dt^2}\right) \tag{2-3}$$

i.e.

$$\frac{\pi}{6}\,d^3\,(\delta-\rho)\,r\,\omega^2 - 3\,\pi\,\mu\,d\,\left(\frac{dr}{dt}\right) = \frac{\pi}{6}\,d^3\,\delta\,\left(\frac{d^2r}{dt^2}\right) \tag{2-4}$$

where t is the time taken for a particle to move. As the particle moves outwards, the accelerating force increases and the particle never acquires a constant velocity in the fluid. If the inertial terms on the right-hand side of equation (2-4) can be neglected:

$$\frac{dr}{dt} = \frac{d^2(\delta-\rho)\, r\, \omega^2}{18\,\mu} = u_0\,\frac{r\,\omega^2}{g} \qquad (2\text{-}5)$$

Therefore, the instantaneous velocity (dr/dt) is equal to the terminal velocity u_0 in the gravitational field, multiplied by a factor of $r\omega^2/g$. For a slurry in a centrifuge, the time taken for a particle initially situated in the slurry surface to reach the wall of the bowl is given by integrating equation (2-5):

$$t = \frac{18\,\mu}{d^2\,\omega^2\,(\delta-\rho)} \ln\left(\frac{R}{r_0}\right) \qquad (2\text{-}6)$$

where R is the radius of the bowl and r_0 is the radius of the slurry surface. These two equations imply that the greater the centrifugal acceleration, the less time will be taken for a particle to settle. Thus, in a centrifugal field, separation can be achieved at a greater rate.

Equations (2-5) and (2-6) are used here just for presenting a simplified relationship between a particle's settling velocity, time, and rotating velocity. In practice, the centrifugal force can really accelerate the separation processes, but the separation mechanism is not simple, nor is it only dependent on d, δ, ρ, μ and ω. For example, based on these two equations, all the silica particles larger than 83 μm will settle faster and take less time to reach the wall of the bowl than tungsten particles (of same specific gravity as pure gold) whose size is equal to or smaller than 25 μm. This is not true in most centrifuge separations, especially with Knelson Concentrators and SuperBowls, which are claimed to be able to recover the full size range of gold, even below 25 μm, in the presence of coarse silica (above 600 μm). Therefore, the ability of the centrifuge units to recover fine gold is based on other mechanisms, such as particle-particle collision and percolation or consolidation trickling (Laplante, 1993; Huang, 1996; Ling and Laplante, 1997).

2.2.3 Minimum Recoverable Particle Size

According to Lu (1994) and Sun (1982), the critical particle size for a particle suspended in centrifugal flowing film is given by:

$$d_{cr} = k_0 \sqrt[4]{\frac{g}{\omega^2 r}} \tag{2-7}$$

where d_{cr} is the critical particle size, k_0 is a proportional constant which is related to operating parameters. Effectively, this means that the minimum recoverable particle size will decrease with increasing rotating speed.

2.3 Gravity Centrifugal Concentrators

The earliest known gravity centrifugal concentrator was patented by Peck in 1891, but this technology was known in the West only until about 20 years ago (Burt et al, 1995). The Knudson bowl is perhaps the oldest centrifugal device for gold recovery, but little information is known about its metallurgical performance. Applications appear to be limited to alluvial operations (Laplante et al, 1994), although some operations based on the In-Line Pressure jig use it as a cleaner in Australia (Laplante, 1998). Centrifugal separators were developed in the (then) Soviet Union in the 1950's and the "Yuxi" Centrifuge units were used in China in the early 1960s. However, the earliest scientific study into centrifugal separation was by Ferrara (1960), who studied a rudimentary unit which then became known as Ferrara's tube: a 20 mm diameter 1100 mm long perspex pipe rotating at up to 2200 rpm. Even though results were very encouraging, the obvious mechanical difficulties inherent to the design made its commercial application impossible (Burt et al, 1995).

Since as recently the 1980s, several new centrifugal separators have been developed, such as the Multi-Gravity Separator (MGS) developed in England; the Kelsey Centrifugal Jig in Australia; gold semi-batch centrifugal concentrators, such as the Knelson, Falcon, and SuperBowl in Canada; and SL-type continuous discharge centrifugal separators (SLS) in China. In mineral processing practice, centrifugal concentrators can be divided into three basic types: vertical axis machines and their sub-set (Knelson, Falcon etc.), centrifugal jigs, and horizontal axis machines (MGS, SLS etc.). The orientation of the rotation axis is usually determined by the means adopted for introducing the feed and removing product streams.

The following sections will briefly describe some of the most recent common centrifugal concentrators in their structures, separation procedure, operating parameters and applications in ascending order of centrifugal force strength.

2.3.1 Mozley Multi-Gravity Separator (MGS)

The Mozley Multi-Gravity Separator (MGS) may be conceived as folding the horizontal surface of a shaking table into a drum, which is rotated to generate 6 to 24 times the acceleration of gravity (Chan et al, 1991). The drum's axis can be inclined to about 10°, and a sinusoidal shake is superimposed on the drum in an axial direction. The diameter of the drum tapers at 1° increasing from the high (concentrate) to the low (tailing) end. One of the original features of the MGS is the presence of scrapers inside the drum to drag the heavy minerals to the concentrate outlet, as shown in figure 2-1. The mine scale MGS consists of two drums, mounted 'back to back' to make the whole machine well balanced and virtually vibration-free.

The slurry is fed continuously mid way onto the internal surface via a meshed ring which reduces the turbulence caused by the introduction of the slurry. Wash water is added via a similar mesh close to the outer end of the drum. The slurry follows a spiral movement within the drum. Under the effect of the centrifugal force, the heaviest particles penetrate the slurry and are pinned to the inner surface of the drum. They are then dragged to the concentrate exit by the movement of the scrapers, which are driven slightly faster than the drum but in the same direction. An intermediate layer forms above the first consisting of finer and less dense particles. The top layer consists of the lightest particles, which are carried by the wash water to the rear of the drum and discarded as tailings. The oscillatory action disrupts these layers temporarily and improves the separation, by minimizing the entrainment of gangue into the concentrate.

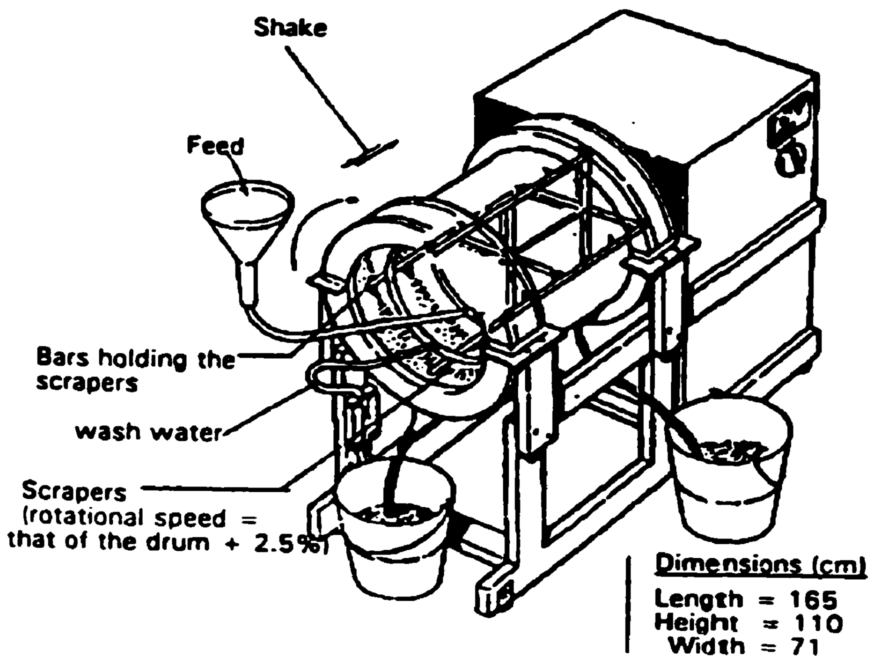

Figure 2-1 General features of the pilot MGS (Traore et al, 1995)

The most important variables governing the operation of the MGS are the rotational speed of the drum, the shake intensity (amplitude and frequency), the wash-water flowrate, the angle of tilt and the flowrate and pulp density of the feed slurry. Chan et al (1991) described their effects in details. With real ores, the most critical operating parameters and their effect can only be determined by actual testing (Traore et al, 1995; Belardi et al, 1995).

Two versions of the MGS are available. The smaller Pilot/Laboratory unit can treat up to 0.2 tonnes of solids per hour. The Mine Scale unit can treat up to 5 tonnes of solids per hour. The MGS makes it possible to extend gravity concentration to ultra-fines, tailings and sludges containing precious metals or high value minerals including tin, chromite, tungsten and the precious metals, in applications which were previously uneconomic. It is also claimed that its high unit capacity makes the processing of lower value and industrial minerals such as iron ore, baryte, anatase, coal etc possible (Chan et al, 1991). Reality paints a different picture. For example, the plant MGS at Renison Tin, Tasmania, upgrades a flotation concentrate from 20 to 40% SnO_2 at a relatively low capacity of 1 t/h. A recently introduced MeGaSep unit can treat up to 60 tonnes of solids per hour (http//:www.mozley.co.uk).

Plant-scale tests performed at Carnon Consolidated's Wheal Jane Tin Mine showed that the MGS achieved considerably higher concentrate grades than the conventional shaking tables (55% compared to 36% tin) at similar recovery (36%) and triple the throughput. It was expected that further optimization of the operating parameters would give significantly improved results (Chan et al, 1991).

2.3.2 Kelsey Jig

The Kelsey Centrifugal Jig design is based on a standard jig, operating within a 60 Gs centrifugal field. It consists of several hutches which are turned from a vertical to a horizontal orientation and rotate on a vertical axis, just like a rotating bowl surrounded by concentrate and tailing launders. Within this bowl, there is an impeller and cover to assist the slurry distribution evenly and a parabolic wedge wire screen to retain a ragging bed. The jig hutch design incorporates a side pulsing mechanism (Figure 2-2). In the model of J650 jig shown in Figure 2-3, there are eight concentrate hutches as described in Figure 2-2, which are positioned horizontally. These hutches hold the pulse water and discharge concentrate through their spigots.

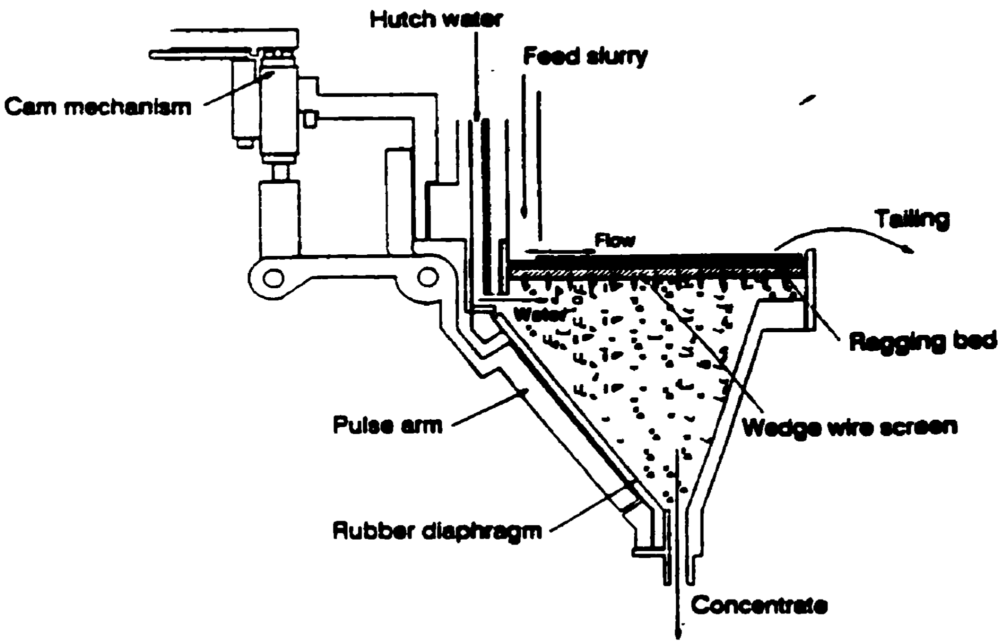

Figure 2-2 Kelsey Jig side pulsing hutch mechanism (Beniuk et al, 1994)

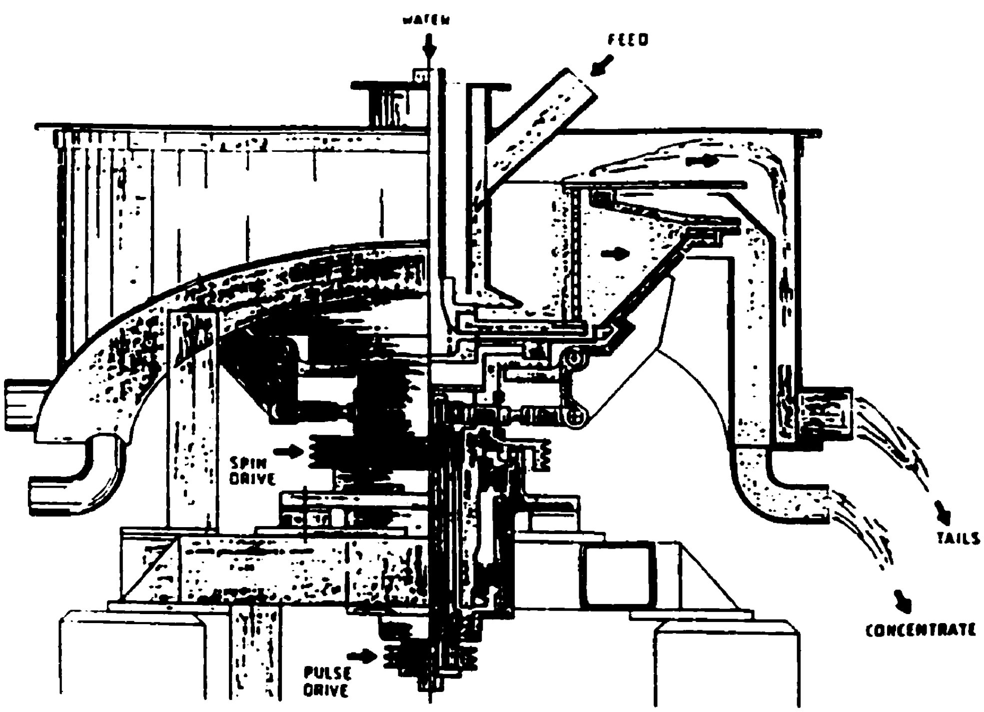

Figure 2-3 Model J650 Kelsey centrifugal jig (Beniuk et al, 1994)

Beniuk et al (1994) described the Kelsey Jig separation procedure as follows. Feed slurry enters the jig through an central feed pipe and is distributed onto the ragging bed by the centrifugal force. Hutch water is also fed through the inner central feed pipe into the concentrate hutches via water spigots. The high frequency strokes of the pulse arms create an inward pulse of water through the ragging bed and cause it to dilate and contract, much as it would in a classical jig, but at a much higher frequency. The water pulse impacts different accelerations to the feed and ragging particles according to their specific gravity. Therefore heavy particles can be separated from the lighter ones under different centrifugal forces. The heavy particles move through the ragging bed and the wedge wire screen into the hutches and are discharged through spigots into the concentrate launder. The low density particles are displaced from the surface of the ragging layer by incoming feed, and report to the tailing launder. The separation of fine particles can be achieved when high relative centrifugal forces are applied.

The major variables affecting the performance of the Kelsey Jig were discussed by Wyslouzil (1990) as: ragging material, jig rotational speed, pulse action, and pulse water. The relationship between feed particle size and ragging retention screen aperture size were shown to be critical to metallurgical performance and operational stability (Beniuk et al. 1994).

Many different scale Kelsey Jigs are available: two laboratory models J125 and J200; a pilot model, J470; and plant models J650, J1300[1]. The J1300 model incorporates many design improvements, the most recent being an automatic screen cleaning mechanism and has became the standard plant unit. A new model, the J1800, is scheduled for testing at Iron Ore of Canada, Limited, in the summer of 1998. It has been claimed that Kelseys are more suited to fine sands than ultrafines and were successfully applied in the pilot plant for a large, fine grained (-100+50 μm) heavy mineral sands project in Australia (Burt et

1. The Kelsey jig is identified by its screen diameter; thus the J1300 has a 1300-mm diameter.

al, 1995). It is true that they have been successfully applied to heavy mineral sands to separate zircon (s.g. 4.6-4.7) from kyanite (s.g. 3.6) in a number of plants, first in Western Australia and now in South Africa. Very high grades and recoveries are routinely achieved despite the low specific gravity differential. However, the unit has also proven its capabilities for fine feeds, such as cassiterite recovery at Renison (Beniuk et al, 1994), tantalite and cassiterite recovery at Gwalia's Greenbushes and Wogina plants.

2.3.3 Knelson Concentrator (KC)

The standard model KC (Figure 2-4) is a centrifuge that develops an average of 60 Gs and classifies a feed of top size of 6 mm or less by specific gravity (Knelson and Jones, 1994). It consists of a ringed bowl rotating at high speed with a drive unit. Clean water at high pressure is injected tangentially through holes in the inner bowl, counter-current to the rotation of the bowl. The feed slurry is introduced to the bottom of the bowl by gravity through a downcomer. Under the effect of the centrifugal force, the heavy particles will report to the riffles as concentrate, whereas gangue minerals will be rejected by the upward flow of slurry to the outer rim as tails.

Compared to other centrifuges, the Knelson Concentrator possesses quite different features either in design or separation mechanisms. Knelson (1988) claimed that this unit uses the principles of hindered settling and interstitial trickling enhanced by centrifugal force. The centrifugal force that would cause packing of material in the rings is partially offset by the fluidization water which is tangentially injected into the bowl from small holes at the bottom of parallel grooves. The fluidized bed behaves as a heavy liquid with the density of the pulp and thus a hindered settling condition prevails. Under this constantly agitated environment, concentration takes place with the particles of higher specific density displacing lighter ones and embedding themselves in the interstices of the gangue (Knelson and Jones, 1994).

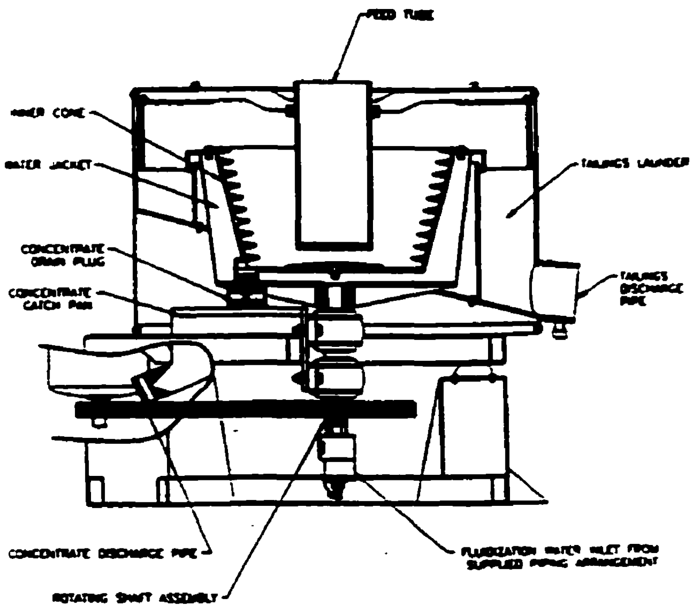

Figure 2-4 76 cm manual discharge Knelson Concentrator (Knelson and Jones, 1994)

Huang (1996) studied the basic behaviour of minerals and separation mechanisms of Knelson by using synthetic ores and bowls with different features. He suggested that the rings are completely fluidized only at the beginning of the separation; as soon as the material bed builds up in each ring, it becomes unfluidized. Therefore, there is almost no mass transfer between the fresh feed and the solids already recovered in the rings (the concentrate bed), as even fine dense particles cannot penetrate the bed. As a result, separation takes place at the surface of the concentrate bed, where the slurry is at least partially fluidized by the high pressure fluidizing water flow and the feed slurry flow. In the separation zone, the recovery or rejection of a particle mainly depends on the forces acting on it, such as centrifugal, drag (caused both the fluidization water and feed slurry) and momentum transfer caused by collisions between particles.

Laplante et al (1996b) systematically tested the effect of feed rate, density, size distribution and fluidization water pressure by using a 7.5 cm laboratory Knelson Concentrator (LKC). It was found that the efficiency of the Knelson is affected primarily by gangue density and feed rate, both capable of significantly lowering the recovery of fines or intermediate size gold at high values. A wide range of fluidizing water flow was found suitable for most separations. Feed size distribution had little effect on Knelson performance, provided gangue density was low, except for very coarse (+1 mm) feeds. A feed density from 0 to 70% solids is claimed to be handled without any detrimental effect to operational efficiency (Knelson and Jones, 1994).

Laplante et al (1998) presented comparative results of the performance of a 3-in Knelson Concentrator and a 4-in SuperBowl on flash flotation concentrates. The higher recovery of the SuperBowl below 25 or 37 μm was offset by a lower recovery between 37 and 212 μm, and both units yielded similar overall recoveries.

Twelve models of the Knelson Concentrator are available, from 7.5 to 76 cm (Knelson, 1988 and 1992). Knelson Concentrators can be classified into the Manual Discharge, Centre Discharge (CD), and Variable Discharge models. Since the first unit was commissioned in 1980, more than 800 Knelson Concentrators have been installed in over 60 countries. They have a large throughput capacity, e.g. a 76 cm KC can treat up to 70 tonnes material per hour, and the ability to treat a wide size range of material without desliming. A continuously operating Knelson Concentrator will be suitable for the base metal and coal industries (Knelson and Jones, 1994), and is still in the developmental stage.

The number of documented successful industrial applications is too large to be discussed here (e.g. Laplante, 1987; Darton et al, 1992; Hart and Hill, 1994; Cloutt, 1995; Hope et al. 1995; Vincent, 1997), the reader is referred to Ling (1998) for a detailed discussion.

2.3.4 Falcon SuperBowl (SB)

The Falcon SuperBowl is a new type of patented gold centrifugal concentrator which combines the proven fine gold recovery characteristics of single wall concentrators (Falcon Concentrator) with capabilities of backpressure technology. It is designed primarily for gold recovery from grinding circuit or alluvial operations, the fluidized bed SB gravity concentrators compliment the Falcon product line by providing coarser feed capabilities. This new technology results in a machine able to apply extremely high centrifugal force on the treated materials (up to 200 Gs) and enhance full size gold recovery while using less process water. Figure 2-6 shows the nominal specifications of the Model SB4 which is specifically designed for laboratory test work, small ore samples or concentrate cleaning with its main parts. The design and material of construction have been carefully selected to minimize the possibility of contamination between samples (Model SB4 operating guide).

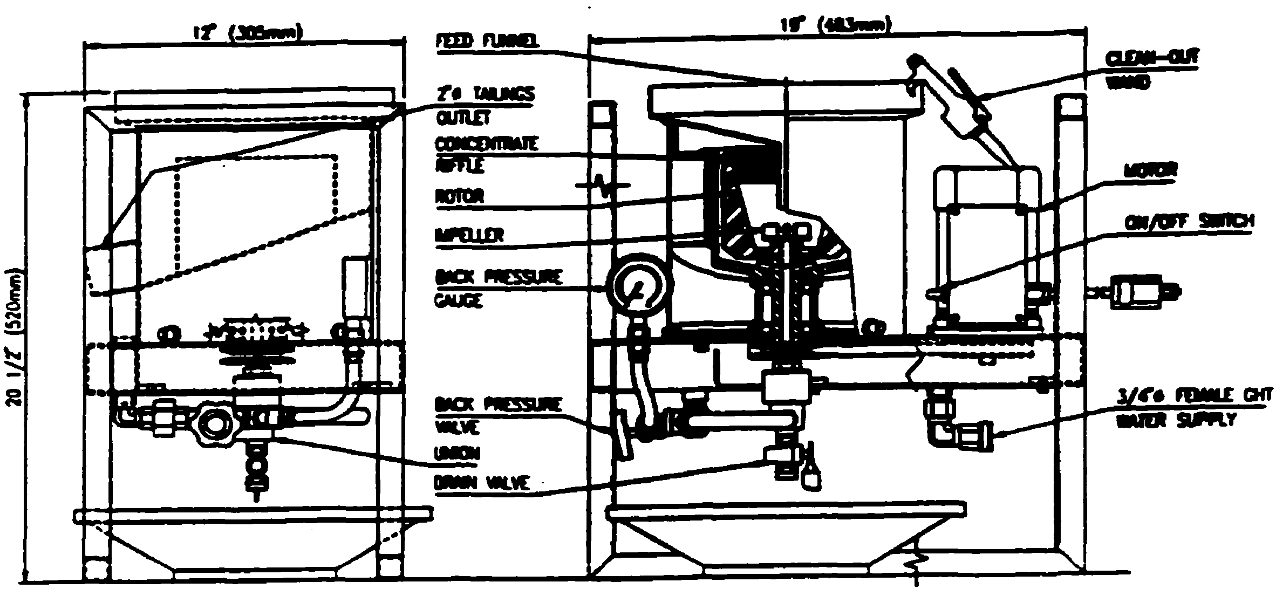

Figure 2-5 Nominal specifications of the Model SB4

The SuperBowl concentrator utilizes the difference in specific gravity between gold and gangue particles to effect a separation. Feed is introduced as a slurry through the central vertical feed pipe and accelerated by the rotor turning at sufficient RMP to impart up to 200 Gs to the material being processed. The extremely high centrifugal force magnifies the difference in specific gravity of different particles and the rotor geometry assists in the retention of gold or heavy particles in preference to light particles that are rejected with the process water. Backpressure (fluidizing) water is injected between the riffle rings in the upper part of the rotor from outside to allow heavy particles to move into the upgrading concentrate retention zone. After feeding all the materials, the rotor is stopped and the concentrate is rinsed down through the concentrate discharge ports.

Lancup (1998) used four different tungsten, silica and magnetite synthetic feeds to test the effect of feed density at different feed rates. The feeds consisted of 0.5% tungsten, 0-10% magnetite with a total mass of 20 kg each. The results showed that the concentrate bed was packed harder with increasing feed rate. The distribution of concentrate inside the bowl was observed with a packed concentrated bed. In the upper riffle part, tungsten was found predominantly in the inner section, i.e. outside the riffle, while the riffle grooves were mainly occupied by silica as observed by Huang (1996) with a separable KC bowl. In the lower smooth part, tungsten was also distributed on the top of the concentrate bed which is similar to what Buonvino (1994) observed for a batch Falcon-B6.

As part of this research program, a 4-in SB has been systematically tested with different synthetic ores to investigate the effect of its operating parameters (backpressure water flowrate) and feed conditions (feed rate, size distribution, gangue density). The test results will be presented in Chapter 6.

SuperBowls are currently offered in four sizes: a laboratory model SB4 and three plant models-SB12, SB21, SB38 with capacities from 0.25-60 tonnes per hour. RMS Ross Corporation has compiled a list of SuperBowls that are in operation throughout the world and showed that about 35 units are now used in 12 countries (Sutter, 1997). A model

SB52 (with a 200 ton per hour capacity) is currently in the final phase of development and will be introduced in 1998. The SB52 will incorporate many design features of the Falcon model C40 Continuous Concentrator.

Mineral Hill Mine incorporated a 21-in SuperBowl (SB21) in its grinding circuit. The plant tests showed that total gold stage recoveries were 41-66%, while stage GRG recoveries varied between 56 and 82%, and increased slightly at finer particle size. As the extremely low feed rate 1.6-8.5 t/h (maximum capacity is 15 t/h) resulted by an inadequate screening surface, total gold recovery was low, about 16%. This test work is detailed in Chapter 3. At New Britannia Mine, a SB21 achieved gold stage and plant recoveries of 42% and 40%, respectively, by treating the primary cyclone underflow at a capacity of 8.8 tonnes per hour. This work is also presented and evaluated in this work, in Chapter 5.

2.3.5 Falcon Concentrator

The Falcon Concentrators were initially developed to recover fine gold under a very high centrifugal field, up to 300 Gs. The only moving part, rotor, is smooth and combined by the upper cylinder and lower conical part which are called the migration and retention zone, respectively, as shown in Figure 2-6 (Robertson, 1998). The feed is screened at approximately 1 mm depending on the application and is introduced as a slurry through a central vertical feed pipe and is accelerated by an impeller. Rapid stratification according to specific gravity occurs as the material is driven up the sloping rubber lined rotor wall (migration zone) under the influence of an immense gravity field. The size of the field is varied by changing the rate of revolution of the rotor with a variable frequency drive. The concentrated heavy fraction is withdrawn continuously through a series of ports distributed evenly around the circumference of the rotor, whereas the light material flows upwards and out of the rotor into the tail launder. For a batch Falcon, the concentrate is washed out once the operating cycle is finished.

A batch Falcon Concentrator, B6, was tested on both synthetic and real ores to explore its mechanism and performance (Buonvino, 1993; Laplante et al, 1994). Based on the results of the overload test, three recovery phases are suggested: (i) initial unselective recovery, (ii) selective recovery, rapidly dropping with the bed saturation, and (iii) stable and near-zero recovery. Among the these, only the second recovery is desirable. The recovery of the particles is described in two different ways: coarse particles can partially bury themselves in the concentrate bed, while fine particles are captured when they lodge in capture sites created by the concentrate bed. Intermediate particles experience the lowest recovery, as they are too coarse for many of the capture sites, and too fine to bury themselves in the concentrate bed. The results showed that bowl geometry (angle) and gangue size distribution are the most significant operating parameters. Recovery increases with decreasing gangue and heavy particle size, and the small angle bowl is suitable for recovering heavy minerals, whereas the large angle bowl is designed to recover gold from lighter minerals.

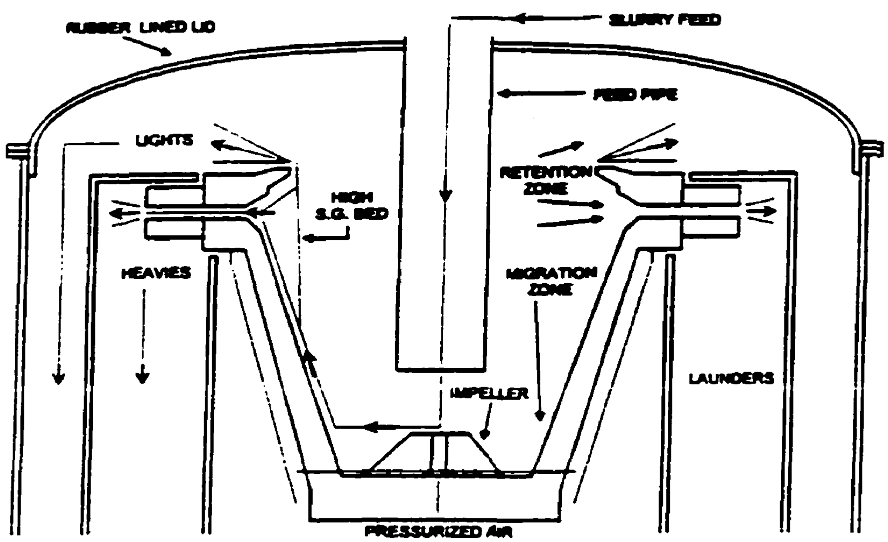

Figure 2-6 A sketches of continuous Falcon Concentrator

Two series of Falcon Concentrators at two different size are available: the batch (or semi-batch, as tails are removed continuously) units, B6 and B12, and continuous units, C10, C20 and C40. The continuous Falcon is suitable for a wide range of minerals and fine feed applications requiring low to high mass recoveries. It is claimed that Falcon concentrators have proven effective in many gravity applications as a primary concentrator, secondary or tertiary cleaning step, or as a scavenging unit (McAlister, 1992). Reality is that although many pilot runs have completed both at mine sites (Lupin Mine, N.W.T.; Carol-Lake, Labrador; New Celebration, Western Australia) and research centres (CRM, Québec; Southern Illinois University, Illinois), the only industrial application that resulted from this work as of July 1998 (to the author's knowledge) is the one at Tanco Mine (Manitoba), where a C20 is used as a rougher feeding two Holman tables, at the tail end of a spiral and table circuit. A C10 will eventually further upgrade the C20 concentrate, to reduce the flow to the Holman tables whilst increasing the yield of the C20. Further, many case studies confirmed that they were not suitable for all applications. This is mainly due to its separation limits, such as rapid overload of the concentrate bed and the creation of a bed of feed at the beginning of the test. Its lack of any water addition also limits its efficiency significantly (Laplante et al, 1994; Honaker et al, 1995; Lins et al, 1992; Huang, 1996).

2.3.6 SL-Type Separator (SLS)

The SL-type Separator is a new continuous discharge centrifugal separator with injection flow (injection flowing film centrifugation, IFFC). It is a horizontal unit and its main body is drum with a diameter progressively increasing toward the discharge end at an angle 2.6°. The units of different size use an adjustable centrifugal force which varies between 83 to 1500 Gs. Figure 2-7 shows a schematic diagram of SLS (Lu, 1994). Slurry is fed at an optimum pulp density onto the inner surface of a drum (1) through a pipe (3); because of the high rotation speed, a stratified bed of moving particles is formed. The heavy particles remain in contact with the drum surface and are carried away as a concentrate via an exit (9) by the impact of a high pressure water jet (4) acting against the longitudinal flow. The light particles, i.e. tailings, overflow downstream (10). The mechanism of the injection flowing film centrifugation is quite complex and was discussed in details by Lu (1994) and Ren et al (1994).

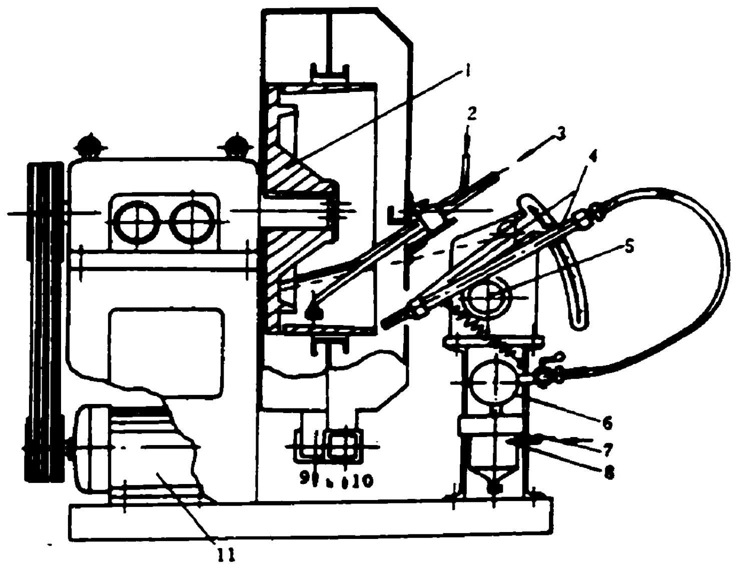

Figure 2-7 Schematic diagram of SL-type Separator (Lu, 1994)
1-drum; 2-low pressure water; 3-feed; 4-water jet; 5-low speed motor; 6-voltage-stabilizer; 7-high pressure water; 8-secondary cleaner; 9-concentrate; 10-tailing; 11-major motor

The main operating parameters are the centrifugal strength and the pressure of the water jet. Feed percent solids have some effect on the enrichment ratio and recovery. Lower percent solids are suitable for roughing, while higher ones for cleaning.

There are six types units available, SL-300, SL-600, SL-1200I, SL-1200II, SL-

1800, SL-2400, with capacities varying from 0.11 to 12 tonnes per hour. The SLS is aimed at recovering fine and ultrafine cassiterite slimes. By adjusting the operating parameters, it can be used to treat different size slimes with high recovery, even 1 μm cassiterite (Lu, 1994).

Commercial tests were conducted in the mineral processing plant of Dachang Tin Mine Company with three SL-600s in an open circuit as primary roughing and cleaning units to treat -10 μm cassiterite slimes discarded as final tailing and + 10 μm flotation feed slimes. The results showed that 55-60% recoveries could be achieved with an upgrading ratio of 10. Compared with flotation, the SLS yielded twice the concentrate grade at a 30% increase in recovery (Ren et al, 1994).

CHAPTER 3

ANALYSIS OF SAMPLES FROM MINERAL HILL MINE

3.1 Description of the Plant

The Mineral Hill Mine of TVX Gold Inc. is located in Park county, Montana, USA. As of December 31, 1995, the total proven and probable reserves were 287,000 ounces with an average grade of 11.1 g/t. The mine was placed under maintenance in the third quarter of 1996, because of political pressure from environmental groups. Before that, the mill processed 500 tonnes per day of ore at an average grade of 10 g/t Au (Minerals Yearbook, 1995; Mining 1998, 1997).

A two stage closed grinding circuit was used in this plant. Crushed ore was fed to a SAG mill, whose discharge fed to a primary cyclone. A bleed of primary cyclone underflow was screened at 1.7 mm and fed to a 21-in Falcon SuperBowl (SB), while the rest was reground by the SAG mill. The primary cyclone overflow was further classified by the secondary cyclone and its underflow was fed to a ball mill whose discharge was combined with the SB tails and screen oversize and returned to the primary cyclone. The SB concentrate was fed to a shaking table to produce a smelting grade concentrate. The secondary cyclone overflow was thickened and then fed to a leaching circuit. A simplified flowsheet of the grinding-gravity circuit is shown in Figure 3-1.

3.2 Objectives

The objectives of this test work were: (i) to determine the optimum operating conditions for the 21-in SuperBowl such as fluidizing water pressure and feed rate, (ii) to confirm the character of GRG (Laplante, 1996) in the grinding circuit and (iii) to examine the shaking table performance.

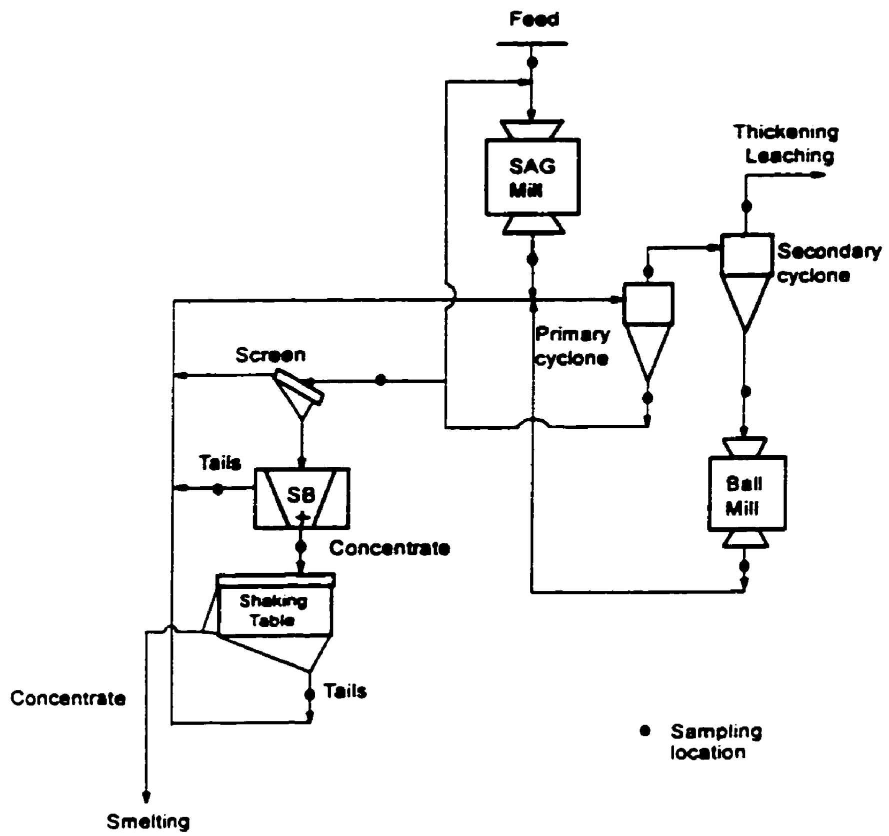

Figure 3-1 Simplified grinding/gravity circuit of the Mineral Hill Mine

3.3 Plant Sampling Procedure

Five tests were performed at the end of August 1996 by Mr. Bogdan Damjanovic with a 21-in SuperBowl at different fluidization water pressures (e.g. flowrate) and feed rates. Only the SuperBowl feed and tails were sampled. In a second phase, all the grinding circuit streams and shaking table feed and tails were sampled at the same time. The sampling locations are indicated in Figure 3-1, while Table 3-1 describes the samples extracted and the plant operating conditions. The fluidization water flowrate and solids content in SB tails were used to calculate the dry feed rate of the unit.

3.4 Laboratory Test Work

All samples except for the belt feed (not listed in Table 3-1) were dried and weighed, then screened at 850 μm (20 mesh) with a Sweco screen. Only the -850 μm fractions were fed to a 3-in Laboratory Knelson Concentrator (LKC), except for the oversize of the SAG mill discharge, which was crushed and ground to -850 μm and fed to LKC with the undersize fraction to estimate the overall grade. For the shaking table feed and tails, a different inner bowl labelled 'high grade' was used to minimize contamination risks; other samples were treated with the low grade bowl.

For most samples, the target feed rate was 600 g/min with a fluidization water pressure of 24 kPa (3.5 psi). For the finer samples, a lower feed rate and water pressure were used. As the shaking table samples had a higher specific gravity and coarser size distribution, a higher water pressure, 27 kPa (4.0 psi), was used with a feed rate of 700 g/min. Table 3-2 lists the laboratory operating parameters for all the eighteen samples.

3.4.1 LKC Operating Procedure

The feed was fed from a hopper via a vibrating feeder to the feeding tray of the LKC. At the beginning of each test, the target feed rate and fluidizing water pressure were adjusted. For samples less than 7 kg, the total tails were collected in a large drum; for the others, six tails samples were cut from the tail stream at regular time interval to obtain representative tail samples. After feeding all the material, fluidization water and the LKC motor were turned off at the same time and the feeding tray of the LKC was removed and adhering solids were washed down to tail drum. The concentrate bowl was then removed and all the concentrate was recovered. The feed rate was calculated using

Table 3-1 Description of the samples tested and the plant operating conditions

Sample	Description	Water Pressure (psi) Initial	Water Pressure (psi) Final	Water Flowrate (USgpm) Initial	Water Flowrate (USgpm) Final	Tails (%) Solids	Feed Rate (t/h)
T-I-F T-I-T	Test I Feed Test I Tail	28.0	28.0	94	91	8.54	1.96
T-II-F T-II-T	Test II Feed Test II Tail	22.0	20.0	82	76	1.62	0.29
T-III-F T-III-T	Test III Feed Test III Tail	30.0	28.5	106	95	1.58	0.37
T-IV-F T-IV-T	Test IV Feed Test IV Tail	25.0	27.0	88	89	1.59	0.33
T-V-F T-V-T	Test V Feed Test V Tail	28.0	29.0	90	93	6.47	1.44
PCOF	Primary Cyclone Overflow						
PCUF	Primary Cyclone Underflow						
SCOF	Secondary Cyclone Overflow						
SCUF	Secondary Cyclone						
BMDIS	Ball Mill Discharge						
SAG DIS	SAG Mill Discharge						
STF	Shaking Table Feed						
STT	Shaking Table Tail						

the recorded processing time.

Table 3-2 Operating parameters of the laboratory Knelson Concentrator

Sample	Mass (kg)	-850 μm Mass (kg)	-850 μm Mass (%)	Feed Rate (g/min)	Water Pressure (psi)
T-I-F	16.22	10.89	67	641	3.5
T-I-T	5.19	4.89	94	543	3.5
T-II-F	17.22	10.78	63	798	3.5
T-II-T	8.46	8.02	95	642	3.5
T-III-F	7.68	6.79	88	590	3.5
T-III-T	2.18	2.06	95	589	3.5
T-IV-F	19.55	11.94	61	645	3.5
T-IV-T	3.91	3.75	96	577	3.5
T-V-F	10.50	7.57	72	561	3.5
T-V-T	8.94	8.55	96	743	3.5
PCOF	6.10	5.87	96	367	3.0
PCUF	11.83	9.04	76	623	3.5
SCOF	2.32	2.32	100	258	2.0
SCUF	14.14	13.74	97	482	3.5
BMD	6.84	6.69	98	352	3.0
SAG DIS	8.24	8.24	100*	485	3.5
STF	19.73	17.26	87	639	4.0
STT	18.98	15.61	82	743	4.0

*Oversize was crushed and ground to -850 μm.

The LKC concentrate and six tails were filtered, dried and weighed for the assaying sample preparation. After settling for 12 hours, the bulk tails were also decanted, filtered, and dried.

3.4.2 Assaying Sub-Sample Preparation

For the larger samples, 50 grams were extracted from each of the six tails cuts and then combined to obtain two 300 gram subsamples. For the smaller (< 7 kg) samples, two 300 gram subsamples were extracted from the bulk tails. Each 300 gram subsample was wet screened at 25 μm, then dry screened from 600 μm down to 25 μm. The weight of each fraction was recorded separately and corresponding fractions then combined. The +105 μm fractions were pulverized. The entire concentrates were also screened down to 25 μm. The concentrate size fractions were fully assayed to eliminate the nugget effect (Woodcock, 1994). All the concentrate and tail size fractions were sent to Casa Berardi for fire assaying.

3.5 Results and Discussions

For each sample, a metallurgical balance of the laboratory Knelson test is presented and listed in Appendix A (Table A-1 to A-17). From this balance, the grade of the -850 μm fraction, the amount of GRG as a percent of the total gold, the grade of the non-GRG fraction and size-by-size data can be obtained (Xiao and Laplante, 1997b). Some important data will be shown in figures and discussed in the following three sections.

3.5.1 Plant SuperBowl Tests

Table 3-3 summarises the results of the five plant SB tests. Both GRG and gold recoveries were based on the -850 μm fraction, a legitimate approach, since there is virtually no GRG in the +850 μm fractions. The data of test I are used to illustrate how to calculate these recoveries (Table A-1 and A-2, p. 107). The SuperBowl was fed at 1.96 t/h, with a feed grade of 108.5 g/t, of which 20.8 g/t was not gravity recoverable, and 87.7 g/t (the difference) was. The SB tails had the same rate, but a much lower gold content, 37.0 g/t, of which 20.8 g/t was not gravity recoverable. Plant and stage recoveries can be calculated based on the plant capacity of 20 t/h and a feed grade of 10 g/t as follows:

$$\text{Gold stage recovery} = 100\% * [1 - 37.0\ g/t\ /\ 108.5\ g/t] = 66\%$$

$$\text{GRG stage recovery} = 100\% \bullet [1 - (37.0\ g/t - 20.8\ g/t)\ /\ (108.5\ g/t - 20.8\ g/t)] = 82\%$$

$$\text{Plant gold recovery} = 100\% \bullet (108.5\ g/t - 37.0\ g/t) \bullet 1.96\ t/h\ /\ (10\ g/t \bullet 20\ t/h) = 70\%$$

Table 3-3 Summary of the Plant SuperBowl tests

Test No.	Feed Rate (t/h)	-850 μm Feed Grade (g/t)	-850 μm Tails Grade (g/t)	Stage Gold Recovery (%)	Stage GRG Recovery (%)	Plant Gold Recovery (%)
I	1.96	108.5	37.0	66	82	70.0*
II	0.3	158.6	82.1	48	70	11.1
III	0.4	99.4	49.2	51	73	9.3
IV	0.3	109.1	42.0	62	78	11.1
V	1.4	92.2	54.7	41	56	27.0

*Test error, as the historical gravity circuit recovery was 15 to 20%.

Table 3-3 shows that extremely low SB feed rates were used, 0.3 to 1.96 t/h; this was caused by an inadequate screening surface for the bleed of the grinding circulating load fed to the SB21. As the rated capacity of the SB21 is about 15 t/h, all feed rates were considerably below designed feed rate, yielding information of questionable value. As a result, performance in the range of feed rates tested was higher than what it should normally be for such a unit and virtually independent of feed rate, as it is for the Knelson Concentrator at low feed rate (Laplante et al, 1996b; Laplante, 1997).

As fluidization water flow rates changed very little from test to test, they are not expected to impact on SuperBowl performance. Therefore, it is impossible to determine the optimized fluidization water flowrate and pressure. Nevertheless, valuable information can be extracted from this five tests.

The stage gold recoveries of the five tests varied between 41 and 66%; as expected, GRG recoveries were higher, 56 to 82%. Total gold recovery is not a true measure of SB21 performance as unliberated gold should not be recovered. Hence, GRG recovery is a better measure of unit performance. Since so little of the circulating load was fed to the SB21, total plant gold recovery was low, averaging 15% for the last four tests. The lower plant gold recovery resulted in the high circulating load of gold of 90-160 g/t (about 1800-3200%). With a higher SB21 feed rate, plant recovery would have increased significantly and the circulating load would have dropped significantly.

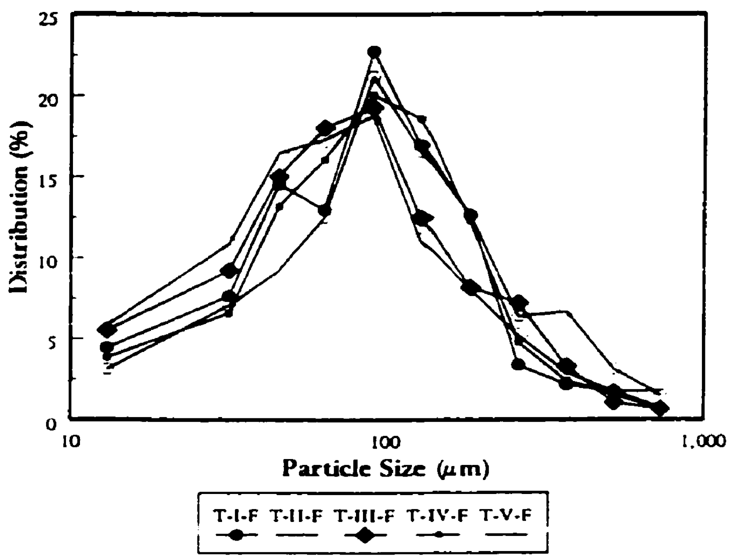

Figure 3-2 Gold size-by-size distribution of the -850 μm fraction of the SuperBowl feeds

Figure 3-2 gives gold's size-by-size distributions for the -850 μm SB feeds. They are very similar: less than 2% gold is in the 850-600 μm fraction, about 5% of the gold

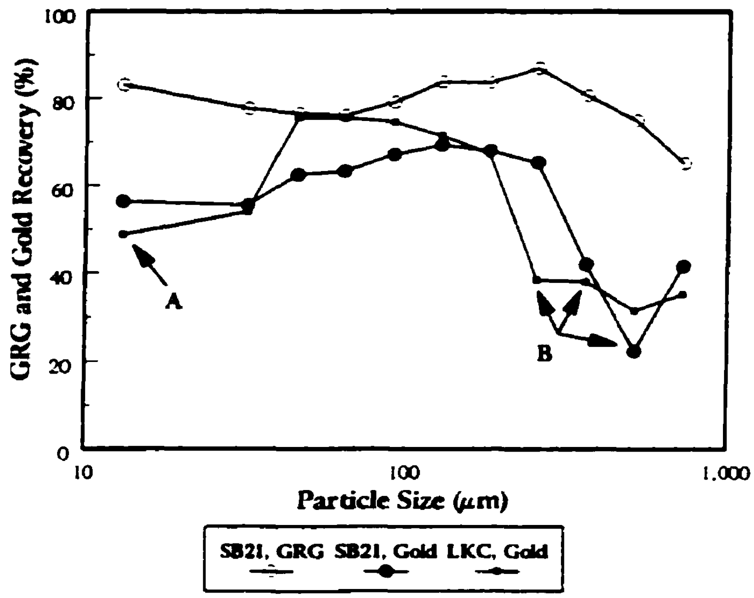

Figure 3-3 Size-by-size GRG and gold stage recovery of the -850 μm fraction of the sample

is below 25 μm and most gold is distributed in the 25-600 μm size range. This similarity makes it possible to calculate an averaged size-by-size performance for the five tests, which is shown in Figure 3-3. Three curves are presented. First, total gold recovery was calculated for the SB21, based on the total gold content of each size class in the SB21 feed and tails (as measured by the LKC). Second, the GRG content of the SB21 tails as measured by the LKC was used to calculate the SB21 GRG recovery. The amount of GRG in the SB21 feed was assumed equal to the total gold grade minus the non-GRG grade of the SB21 tails. Normally, the measured GRG content of the SB21 feed should have been used, but differences between the SB and KC performance made this approach less desirable. Third, the LKC recovery on the SB21 feed is shown. The SB21 outperforms the LKC below 25 μm (point A), which is the reason the GRG content of the SB21 cannot be used to calculate GRG recovery (which would then be greater than 100% for the -25 μm fraction). Note that total gold recovery is also greater for the SB21 for three of the coarsest size classes, but this could be due to the nugget effect (point B). Because so little gold is found above 212 μm (about 1% of total in each size class), and its liberation uncertain, it is impossible to rule in favour of either of the units. The nugget effect cannot be invoked for the -25 μm result; rather, the higher recovery of the SB21 was clearly the result of its higher acceleration, 120 Gs vs the LKC 40-70 Gs (from ring 1 to 5). Overall, the extremely low feed rate of the SB21 made its performance very similar to that of the LKC. Above 37 μm and below 212 μm, the LKC recovered more gold, because the higher Gs of the SB21 partially collapse the flowing slurry and make the percolation to the surface of the concentrate bed of particles in the 37-212 μm range more difficult.

3.5.2 Shaking Table Performance

Table 3-4 gives the shaking table test results. This information is important, as the shaking table feed (STF) is in fact the SB21 concentrate. About three quarters of the gold was concentrated by the shaking table to its concentrate, as the feed assayed 2896 g/t, and the tails 729 g/t (Table A-11 and A-12, p. 110). The GRG content of the two samples was underestimated because of the large feed masses (17.26 and 15.61 kg), which yielded very high laboratory Knelson grades, 10.7% gold for the feed and 2.1% for the table tails. With lower feed masses, the GRG content in the shaking table feed (STF) should be as high as 90%, as we obtained at Casa Berardi and many other plants. The GRG content of the table tails is probably much closer to the correct value, because the lower gold content resulted in a less severe overload (Huang, 1996). Thus, the GRG recovery of the shaking table was calculated using the GRG content of its tails. This yielded a GRG recovery of 86%, or

$$\text{GRG recovery} = 100\% * [1 - (729\text{-}386) / 2896\text{-}386)] = 86\%$$

This estimate of GRG recovery is an upper bound, as the GRG content of the STT was underestimated because of the high concentrate grade. The recovery of gold would be the lower bound (i.e. assuming that all gold in the STT is gravity recoverable).

Table 3-4 Shaking Table test results

Product	Grade (g/t)	LKC Rec. (%)	Stage Rec. (%)	GRG Rec. (%)
Feed	2896	50	75	86*
Tails	729	48		

* Based on the GRG content of the table tails.

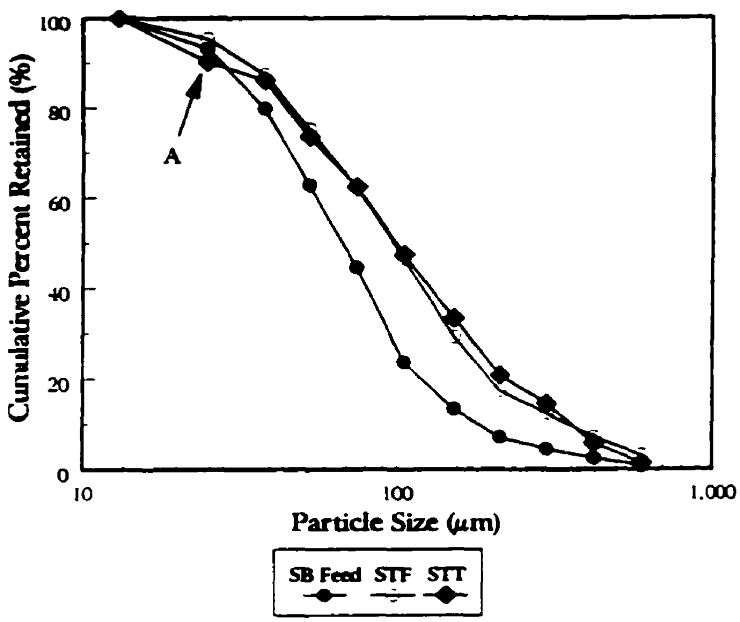

Figure 3-4 Gold cumulative size distribution of the -850 μm fraction of the shaking table feed, tails, and SuperBowl feed

Figure 3-4 shows that gold in the SuperBowl concentrate (STF) is much coarser than in its feed (PCUF). Between 105 and 600 μm, the gold distributions in the SB concentrate are all higher than its feed, while below 105 μm, the opposite results is obtained.

There is a good agreement between the STF and STT cumulative size distribution. But the STT has a higher content of -25 μm gold than the STF (point A), as the shaking table cannot recover it as well as the coarser fractions.

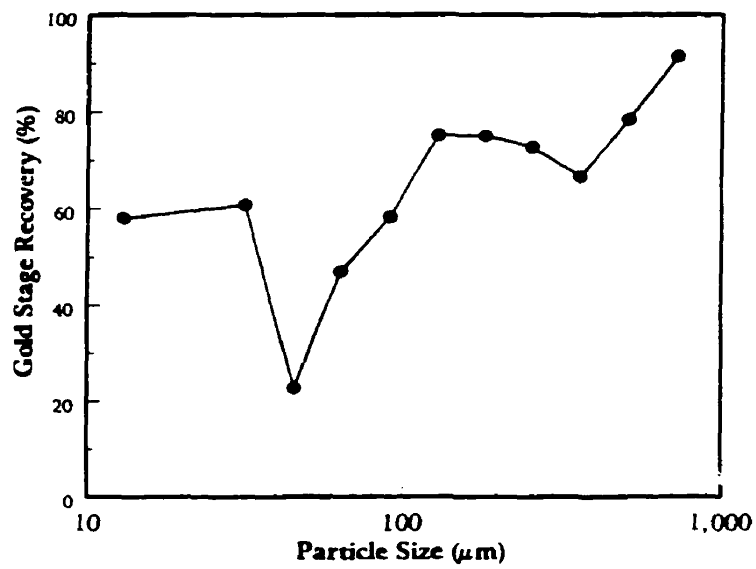

Figure 3-5 Shaking table gold stage recovery

Figure 3-5 shows the table gold recovery as a function of particle size. It can be assumed that nearly all of the gold is gravity recoverable, which means that GRG recovery would only be slightly higher. The low recovery of the 37-53 μm fraction could be due to either to sampling, screening or assaying errors. Generally, recovery drops with decreasing particle size, an indication that table performance, rather than liberation, is the main source of gold loss. Table performance could be significantly improved, as most industrial operations can achieve table recoveries of 80 to 95%. One possibility is to screen table tails at 300 μm and reprocess the undersize in a small diameter centrifuge to recover more smeltable gold (Huang, 1996).

3.5.3 Grinding Circuit Survey

Figure 3-6 shows that gold grades vary widely in the grinding circuit. The primary cyclone concentrates a large amount of gold to its underflow (an indication of low gravity efficiency), while the secondary cyclone has a low enrichment ratio. This confirms further that the primary cyclone underflow should be the target of the gravity circuit. Except for the secondary cyclone overflow, the GRG content is almost constant. The GRG content of SCUF is probably underestimated, as it should be at least the same as that of PCOF[1]. This could be attributed to poor sampling, as will be discussed later.

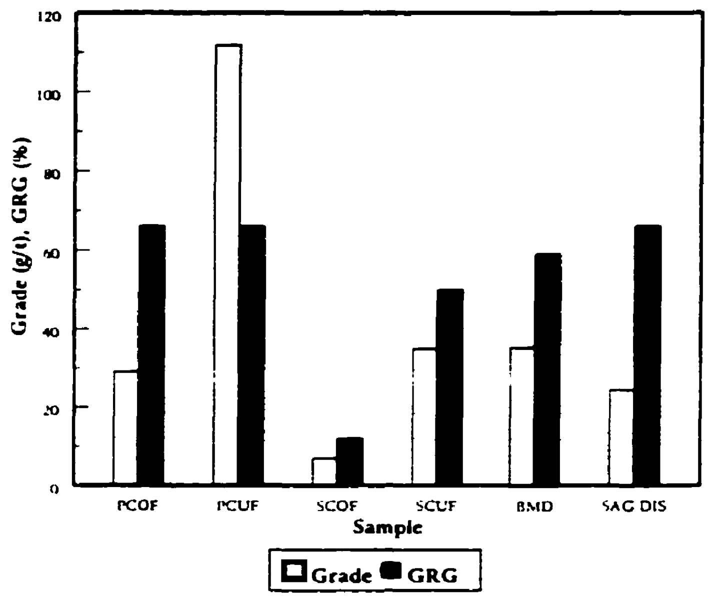

Figure 3-6 Grade and GRG content of the -850 μm fraction of grinding circuit streams

Figure 3-7 gives gold's cumulative size distribution in the -850 μm fraction of

1. The other possibility is that the GRG content of the PCOF was overestimated.

grinding circuit streams. It is evident that there is very little gold above 420 μm in all of the samples. The PCUF and SAG mill discharge (SAG DIS) had a lower gold content in the -25 μm fraction than the other products. As this very fine gold is difficult to recover by gravity, especially with a shaking table, gravity circuit performance should benefit from the low very fine gold content. Gold in the SCUF is noticeably finer than in the PCUF, which would make it more difficult to recover. Notice that the gold in the SCUF is slightly finer than in the PCOF, which obviously is physically impossible, and confirms the sampling problem noted earlier. Note that the size distribution of gold does not depend on the performance of the LKC, which rules out the laboratory procedure as the source of the problem. The sampling error probably stems from unsteady secondary cyclone operation, at least with respect to gold. Unsteady secondary cyclone operation during very stable primary cyclone operation had been noted at the Golden Giant Mine (Banisi et al, 1991).

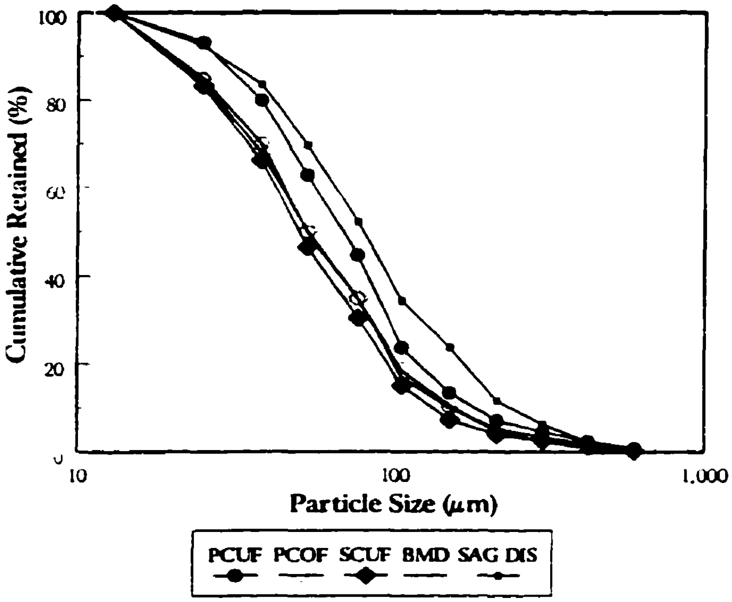

Figure 3-7 Gold cumulative size distribution of the -850 μm fraction of grinding circuit streams

3.6 Summary

By analysing the above data, the following conclusions can be made:

1. The SB21 recovered the full size range of GRG effectively, particularly below 25 μm. As the unit was underfed, it is impossible to assess what its performance would be under normal operating conditions.

2. The low feed rate affected overall gravity circuit efficiency with a plant gold recovery of only 15-20%, obviously much less than what could be reached. This was traced to inefficient screening ahead of the SuperBowl, not to the SuperBowl itself.

3. The extremely low feed rate made it possible to compare directly SB21 and LKC performance. The SuperBowl bettered the LKC below 37 μm, whereas the Knelson was slightly better in the 37-212 μm size range. Unfortunately, the low feed rate also made it impossible to assess how the SB would respond metallurgically and mechanically to normal feed rates.

4. The shaking table achieved a lower performance than that in most plants, which could be improved with a finer feed. This could be achieved either by screening the SB feed finer or screening the SB concentrate in the gold room. Scavenging of the fine table tails with a centrifuge would also improve overall performance significantly, and is the suggested route.

5. The grades of grinding circuit streams varied widely. Most of the gold was concentrated in the primary cyclone underflow, which was chosen quite appropriately as the stream to bleed as SB21 feed. Plant recovery could have been increased significantly with a much higher SB21 feed rate.

CHAPTER 4

EVALUATION OF KNELSON PERFORMANCE AT CASA BERARDI

4.1 Description of the Plant

Les Mines Casa Berardi, a joint venture owned by TVX Gold Inc. (60%) and Golden Knight Resources (40%), is located near La Sarre, Québec. The East mine was put into operation in September 1987, and as of May 1994, the remaining life was seven years. The average head grades of the East and West mines were 14 g/t and 6.2 g/t, respectively. Before it was shut down in 1996, the mill capacity was about 1800 tonnes per day.

The crushed ore was fed to a SAG mill closed by a screw classifier, whose overflow was then fed to two stages of cyclones, the overflow of the first feeding the second. Both cyclone underflows were fed to a ball mill, while a bleed of the primary cyclone underflow, about 30 t/h, was screened at 1.7 mm and fed to a 76 cm CD Knelson Concentrator. The ball mill discharge and the Knelson tails were returned to the primary cyclone pump sump. The Knelson concentrate was fed to a shaking table producing a smelting grade concentrate. The secondary cyclone overflow was thickened and then sent to cyanidation circuit. Figure 4-1 shows the simplified grinding and gravity circuit of Casa Berardi.

4.2 Objectives

The major objectives were: (i) to assess the impact of fluidization water flow rate; (ii) to explore the importance of cycle time; (iii) to establish the relationship between total and gravity-recoverable gold recovery and particle size and (iv) confirm the nature of GRG in the grinding circuit.

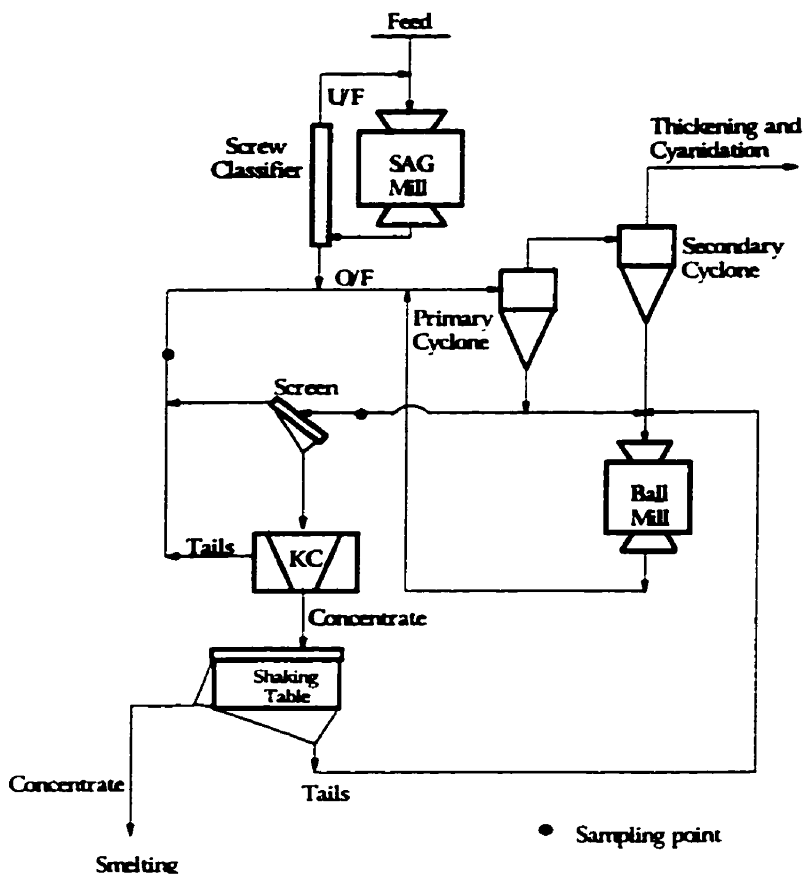

Figure 4-1 Casa Berardi simplified grinding/gravity circuit

4.3 Plant Sampling Procedure

The gravity recovery circuit was sampled on July 17, 1996 by A.R. Laplante and A. Farzanegan, after two days of mechanical problems both with the Knelson screen and SAG mill. Knelson feed, tails, and concentrate were sampled at two different water flow rates and different cycle times. For the 150 L/min water flow rate test, Knelson samples were extracted over a 40 minute cycle. In a second test, fluidization flow was increased to 350 L/min, and a full two hour cycle was used for gold recovery, with separate sampling of the Knelson feed and tails for cycle times of 0-40, 40-80 and 80-120 minutes. The concentrate was sampled at the end of the recovery cycle. As the screen undersize could not be accessed, the screen feed, i.e. the primary cyclone underflow was sampled instead. The Knelson tails samples were actually the Knelson tails combined with the screen oversize as indicated in Figure 4-1. This type of sampling problem is inevitable when the Knelson Concentrators Inc. screen is used. Table 4-1 lists the samples extracted.

4.4 Laboratory Test Work

The previous work had showed that there is very little GRG above 300 μm, so all the samples were screened at 300 μm, and only the -300 μm fraction of the samples was fed to a LKC. For the Knelson concentrates, the high grade bowl was used; other samples were treated with low grade bowl (both bowls are identical: the high grade bowl is generally used with Knelson concentrates or table tails).

The standard procedure described in the previous chapter in sections 3.4.1 and 3.4.2 was used to process all the samples with a fluidization water pressure of 17 kPa (2.5 psi) and a feed rate averaging 300 g/min. Other LKC operating parameters are also listed in Table 4-1. The masses were around 5 kg for most -300 μm samples, which is enough for the purposes of determining gold content. The mass of 30-in Knelson concentrate treated was lower for the first test, but remained adequate due to its higher gold content. All LKC concentrate and tails subsamples were sent to Casa Berardi for fireassaying.

4.5 Results and Discussions

A metallurgical balance for each sample is presented in Appendix B (Table B-1 to B-10). Table 4-2 summarizes the results for the two tests. For all samples, the much

Table 4-1 List of the samples extracted and the operating parameters for treatment with LKC

Test	Sample	Cycle Time (min)	Mass (kg)	-300 μm (kg)	-300 μm (%)	Feed Rate (g/min)	Fluidizing Water Pressure (psi)
150 L/min	KC Feed	0-40	7.16	5.20	73	325	2.5
	KC Tails	0-40	9.08	6.62	73	316	2.5
	KC Conc.	0-40	1.77	1.03	58	343	2.5
350 L/min	KC Feed	0-40	7.75	5.09	66	339	2.5
	KC Tails	0-40	7.28	4.91	67	317	2.5
	KC Feed	40-80	7.57	5.05	67	360	2.5
	KC Tails	40-80	5.64	3.85	68	320	2.5
	KC Feed	80-120	7.16	4.95	69	291	2.5
	KC Tails	80-120	6.96	4.73	68	350	2.5
	KC Conc.	0-120	9.47	5.29	56	302	2.5

lower grade of the 212-300 μm fraction and its very low gold distribution (1% or less) fully vindicate the decision not to treat the oversize. It also suggests that the Knelson feed is much too coarse, which will impact both on feed grade and Knelson performance, especially when the recovery cycle time is extended, because the +300 μm fraction can erode some of the -300 μm gold already recovered (Laplante et al, 1996a; Xiao and Laplante, 1997a).

Table 4-2 Summary of the plant Knelson tests

Test	Sample	Cycle Time (min)	-300 μm Gold (g/t)	GRG (%)	212-300 μm gold (%)
150 L/min	KC Feed	0-40	218.5	62.05	0.46
	KC Tails	0-40	225.5	60.60	0.57
	KC Conc.	0-40	9482	88.95	1.28
350 L/min	KC Feed	0-40	201.3	60.02	0.65
	KC Tails	0-40	194.2	60.11	0.66
	KC Feed	40-80	203.9	58.39	1.01
	KC Tails	40-80	169.0	65.96	0.61
	KC Feed	80-120	167.7	62.35	0.87
	KC Tails	80-120	135.1	57.55	0.69
	KC Conc.	0-120	16055	47.83	4.16

4.5.1 Test at 150 L/min

A fluidization water flow of 150 L/min for a 30-in Knelson, which was used at the time, corresponds to a 1.5 L/min flow rate for the LKC, whose concentrating surface is a hundredfold smaller. This is much lower than the optimum measured for such a unit, which is around 4 to 6 L/min (Huang, 1996).

Figure 4-2 shows that the size distributions of gold are in excellent agreement for

30-in KC feed and tails. The Knelson concentrate has slightly coarser gold, indicating that coarser GRG is preferentially recovered. Table 4-1 shows that coarse gangue is also preferentially recovered, as the -300 μm fraction drops from 73% of the weight for the KC feed and tails to 58% for the concentrate. Visual examination shows much tramp iron and sulphides were recovered in the concentrate, which is expected when an ore with a high content arsenopyrite is ground in a SAG circuit. This material affects both Knelson and table performance.

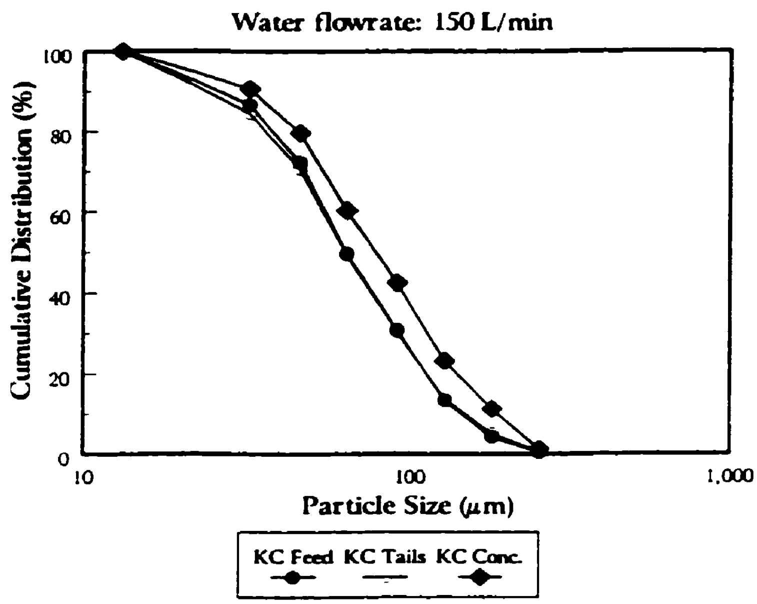

Figure 4-2 Gold distribution of the -300 μm fraction of the 30-in Knelson feed, tails, and concentrate

All the -300 μm fraction grades were high, 219 g/t for the Knelson feed and 226 g/t for its tails. The higher tails grade is an indication of the grade fluctuations in the grinding circuit. This is due to very significant fluctuations in feed grade (mostly because the feed comes from two separate mines, each with a very different gold and GRG content), coupled with a relatively low stage recovery, which results in the Knelson feed and tails grades being very similar.

GRG content is also high, 61-62% for the Knelson feed and tails, and 89% for the concentrate. The high grade and GRG content suggest that the gravity circuit is not quite effective in reducing gold's circulating load. The GRG content of the concentrate might be even higher than 89%, since the high LKC concentrate grade, 6.5% Au, probably produced some overload and some GRG was lost to the LKC tails. There is only 1% of the gold in the 212/300 μm fraction, which means that the table feed could be screened at about 425 μm, and oversize returned to the grinding circuit.

4.5.2 Test at 350 L/min

From Table 4-2, it is apparent that the 30-in KC feed and tails grades fluctuated widely. GRG content is also expected to have fluctuated, as the GRG content is more likely to vary than that of the non-GRG, which is often associated with finer, background gold. Although the total gold content is higher in the KC feed than its tails for all three time periods tested (0-40, 40-80 and 80-120 minutes), the difference between the two is not stable, 7, 35 and 32 g/t, respectively. GRG content also varies greatly, 58-62% for the feed and 58 to 66% for the tails. This makes the average GRG content of the tails (in % of total gold) higher than that of the feed, which is not physically possible.

The agreement in gold size distribution between the Knelson feed and tails (average of the three cycles) is again remarkable, as shown in Figure 4-3. The Knelson concentrate is also coarser than feed as in first test. The implication is that sampling, sample processing and assaying were reliable.

The feed mass of Knelson concentrate was too large and also produced overload of the LKC, observed by the low GRG content, only 48%, and the extremely high concentrate grade, 20.3% gold (shown in Table B-4, p. 115). Obviously plant KC concentrate sample masses must be chosen much more carefully, as the high concentrate grade obtained not only invalidates the GRG content determination, but also constitutes a severe contamination risk.

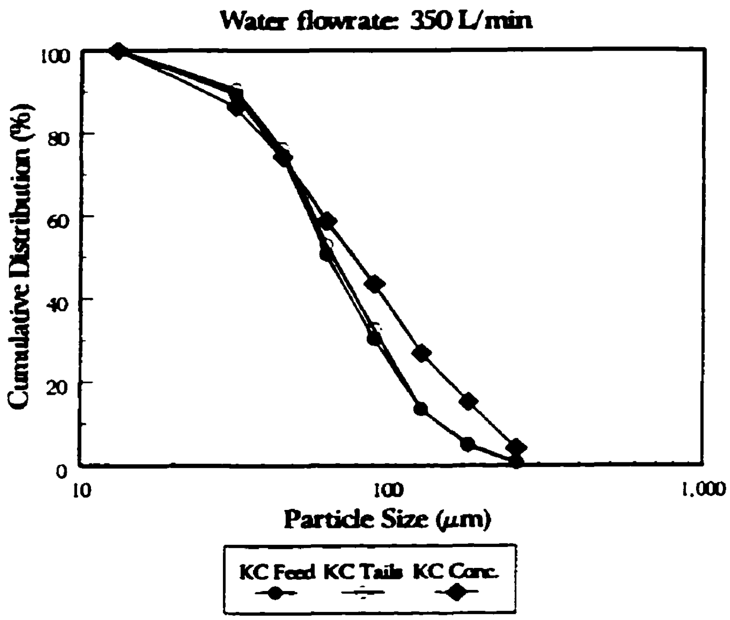

Figure 4-3 Gold distribution of the -300 μm fraction of the 30-in Knelson feed, tails, and concentrate

The concentrate grade at 120 minutes, 16,055 g/t, was relatively low, it should have been, at constant recovery, at least three times higher than that of the 40 minute cycle at 150 L/min, 9482 g/t, due to its longer cycle time. It is concluded that cycle time was too long at 120 minutes and hence produced the overloading of the plant KC.

4.5.3 Recovery Calculation

Both stage and overall gold recoveries can be calculated by assuming a concentrate mass of 45 kg and a feed rate to the Knelson of 30 t/h. The plant capacity will be assumed to be 1800 t/d with a feed grade of 6 g/t. The calculation of the stage gold recovery will be based on the -300 μm fraction and total recovery on the gold fed to the plant.

For the 40 minutes cycle test, the concentrate contained 58% of its weight in the -300 μm fraction at a grade of 9482 g/t. Feed and tails grades averaged 222 g/t, at 73% -300 μm. From this the following calculations are made:

$$Gold\ recovered = \frac{45kg \times \frac{58\%}{100\%} \times 9482g/t}{1000\ kg/t} = 248\ g$$

$$Gold\ fed\ to\ plant = \frac{6\ g/t \times 1800\ t/d \times 40\ minutes}{24\ hours/day \times 60\ minutes/hour} = 300\ g$$

$$Gold\ fed\ to\ Knelson = 222\ g/t \times 30\ t/h \times \frac{40\ minutes}{60\ minutes/hour} \times \frac{73\%}{100\%} = 3241\ g$$

Based on the above information, stage recovery is calculated to be 8%, whereas plant recovery was more than 82%, a very dubious estimate. It is possible that the 1 kg sample was not representative of the total Knelson concentrate. It would have been better to extract a larger primary sample and then split it to 1 kg (which is enough to eliminate nugget effects). The high grades of KC feed and tails might also be due to timing of the test, as sampling was performed several hours after a shut down of the 30-in KC. This may have lead to accumulation of GRG in the circulating load, which would also explain the high plant recovery. This type of problem can arise with punctual sampling surveys.

The performance of the long cycle time test can be estimated in a similar way:

$$Gold\ recovered = \frac{45kg \times \frac{56\%}{100\%} \times 16055g/t}{1000\ kg/t} = 405\ g$$

$$Gold\ fed\ to\ plant = \frac{6\ g/t \times 1800\ t/d \times 120\ minutes}{24\ hours/day \times 60\ minutes/hour} = 900\ g$$

$$Gold\ fed\ to\ Knelson = 191\ g/t \times 30\ t/h \times 2\ hours \times \frac{67\%}{100\%} = 7678\ g$$

For the test at 350 L/min, the increased cycle time was not followed by a corresponding increase in concentrate grade, and both plant and stage recoveries dropped respectively, to 45% and only 5% of the gold in the -300 μm fraction of the Knelson feed.

4.5.4 Comparing the Two Tests

Figure 4-4 shows the grades of -300 μm fraction of the Knelson feed and tails for the two tests. It is obvious that both feed and tails grades decreased as from test 1 to test 2, and even during test 2. As maintenance and mechanical problems were experienced shortly before sampling, a large and unsteady gold inventory in the grinding circuit existed. This may have resulted in high recoveries immediately after the running of the Knelson which partly explains the lower recoveries of the second test.

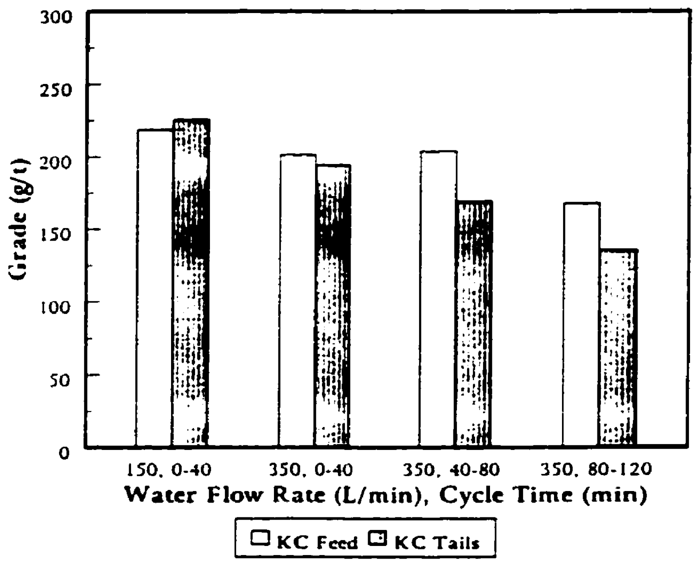

Figure 4-4 Grades of the -300 μm fraction of the 30-in Knelson feed and tails

Figure 4-5 compares the proportion of gold in the -25 μm fraction for the Knelson feed, concentrate and tails of test 1 and 2. It is shown that in test 1, at a fluid flow of 150 L/min, there was an upgrading of the -25 μm in the Knelson tails and a corresponding downgrading in the Knelson concentrate. This problem is only experienced at flows above optimum (obviously not the case here) or significantly below (a more likely occurrence here). At 350 L/min, the -25 μm gold is upgraded into the concentrate, an indication that the optimum flow has not been reached yet. Again, evidence suggests that the lower recovery of the second test was mainly linked to (i) the recovery cycle, which was too long at two hours, on account of the fineness of the gold recovered, and the coarseness of the feed and presence of significant amounts of arsenopyrite, and (ii) the unsteady nature of the gold inventory in the grinding curcuit.

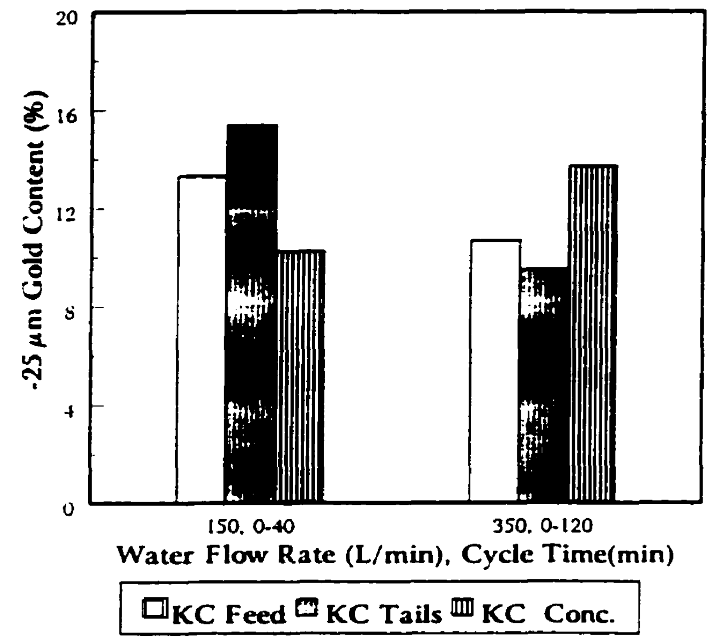

Figure 4-5 -25 μm gold content in the -300 μm fraction of the 30-in Knelson feed, tails, and concentrate

4.6 Summary

Test results confirmed the high variability of the gold content in the grinding circuit of Casa Berardi, which makes test work difficult. However, the following conclusions still can be reached:

1. The GRG content at Casa Berardi remained very fine and comparable to that of previous work. There was virtually no gravity recoverable gold coarser than 300 μm.

2. A fluidization flow rate of 350 L/min appeared more efficient than the 150 L/min previously used, which is still at the lowest limit of Knelson flows for a 30-in unit treating a -1.7 mm feed, especially for the -25 μm gold fraction.

3. The recovery cycle was too long at 120 minutes, which was the main cause for the lower recovery of the second test because of concentrate bed erosion.

4. The fineness of the GRG suggests that a finer feed would improve Knelson efficiency and possibly extend the optimum cycle time. Separate screening of the primary cyclone underflow at 400 μm would achieve this, but would require considerable modifications. Another approach would be to use the secondary cyclone underflow as part or all of the Knelson feed.

CHAPTER 5

TEST WORK AT THE NEW BRITANNIA MINE

5.1 Description of the Plant

The New Britannia Minesite is located in Snow Lake, Manitoba, 420 miles north of Winnipeg. The mine previously operated from 1949 to 1958, extracting some 5.39 million tons of ore at a 0.150 ounce per ton[1] gold grade. In February 1994, TVX Gold Inc. and High River Gold Mines Ltd. entered into a joint venture partnership to develop the Snow Lake property, naming TVX Gold Inc. the operator. Milling began at a rated throughput of 86.6 tons per hour to average 2000 tons/day, and the first gold pour took place in November 1995. In 1996, geological in-situ reserves were estimated at 5 million tons at a grade of 0.194 oz/ton for 967,776 ounces. Diluted mineable reserves are 4,631,527 tons at a 0.168 oz/ton grade for 777,351 ounces of gold. The principal gold bearing rock at the New Britannia Mine is made up of quartz-carbonate material in sequence of basic and acidic volcanic rocks contained within a shear zone. The ore itself is free gold associated with arsenopyrite (Halverson et al, 1996).

A simplified grinding circuit flowsheet is shown in Figure 5-1. The minus 0.75 inch fine ore from the 3000 ton fine ore bin is fed to a 11.5 ft. diameter by 15 ft. Koppers rod mill powered by a 1000 HP synchrous motor. Rod mill discharge at 78% solids is combined with the ball mill discharge at 67% solids and pumped to a 20 inch Krebs primary cyclone. The primary cyclone underflow feeds a 14 ft. diameter by 20 ft. long Nordberg ball mill powered by a 2000 HP synchronous motor. The primary cyclone overflow is pumped to seven 10 inch Krebs secondary cyclones and the secondary cyclone

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1. English units are used in this section, as per the main reference.

underflow is also fed the ball mill. The secondary cyclone overflow reports to a 40 ft. high rate thickener and then to leaching circuit (Halverson et al, 1996).

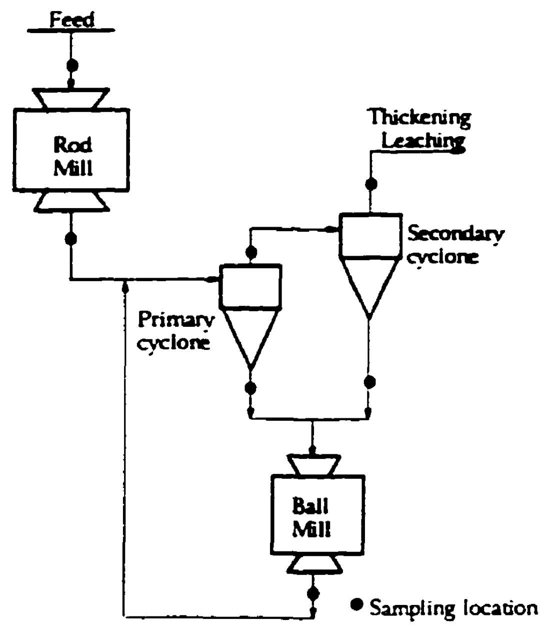

Figure 5-1 Simplified grinding circuit of New Britannia Mine

5.2 Objectives

After the closure of the Mineral Hill Mine and Casa Berardi operations, research was re-oriented to TVX's New Britannia Mine. The objectives were to: (i) determine the size-by-size gravity recoverable gold (GRG) content in this ore; (ii) assess the behaviour of gold in the grinding circuit; (iii) estimate the possibility of installing a gravity centrifuge in the grinding circuit; (iv) compare SB and KC performance at mine site.

5.3 Plant Sampling Procedure

A grinding circuit survey was completed on May 1997. Seven samples of rod mill feed and discharge, ball mill discharge, primary cyclone under and overflow, and secondary cyclone underflow and overflow were extracted and sent to McGill University by the end of May.

5.4 Laboratory Test Work

All the samples were dried and weighed first, and three quarters of the rod mill discharge sample were combined with the rod mill feed sample as the feed for the GRG test. Other samples were split and screened at 850 μm with a Sweco; the six undersize fractions were processed with the same LKC methodology presented in Chapter 3. Subsamples were sent back to New Britannia for fire assaying. Table 5-1 identifies the samples treated and their operating conditions for the LKC tests.

Table 5-1 Description of the samples and the LKC operating conditions

Sample	Total Mass (kg)	-850 μm (%)	-850 μm (kg)*	Feed Rate (g/min)	Water Pressure (psi)
Rod Mill Feed	27.84				
Rod Mill Discharge (RMD)	26.36	93.9	6.16	513	3.5
Ball Mill Discharge (BMD)	23.96	100	6.01	300	3.0
Primary Cyclone Underflow (PCUF)	31.95	88.3	7.24	517	3.5
Primary Cyclone Overflow (PCOF)	15.02	100	7.68	384	3.0
Secondary Cyclone Underflow (SCUF)	27.57	100	7.02	390	3.0
Secondary Cyclone Overflow (SCOF)	11.26	100	5.63	216	2.0

* For LKC tests

<u>GRG measurement</u>

Woodcock (1994) presented a new methodology to characterize gravity recoverable gold (GRG) in an ore. The procedure consists of a three-step recovery by using a 3-in laboratory Knelson Concentrator to treat a sample mass of 40 to 120 kg (typically 50 kg). The sample is crushed and rod milled to 100% -850 μm for the first processing, and the last two recovery steps treat the tails of the previous stage, ground to achieve further gold liberation. Stage two is normally performed on 24-28 kg ground at 45-55% -75 μm, and stage three on 21-24 kg ground at 75-80% -75μm.

The Knelson tests are performed at increasingly lower feed rates and fluidization water pressures to match the finer feed, typically from 1000 g/min and 25 kPa (4 psi) for stage I to 400 g/min and 12 kPa (2 psi) for stage III. Because the test is optimized, it yields the maximum amount of GRG; actual plant recoveries will be lower due to limitations in equipment efficiency, the usual approach of processing only a fraction of the circulating load, and the need to produce a concentrate of smeltable grade (Laplante, 1996).

The LKC operating procedure and assaying subsample preparation were performed as described in 3.4.1 and 3.4.2. Table 5-2 shows the processing conditions for each stage. Note that the feed to stage II was finer than normal, because the feed to stage I was also finer than average.

Table 5-2 Processing parameters of LKC for GRG measurement

Stage	Mass (kg)	Fineness (%)	Feed Rate (g/min)	Water Pressure kPa (psi)	Feed Grade (g/t)
I	45.67	100% -850 μm	1110	28 (4.0)	5.0
II	23.75	67% -75 μm	560	25 (3.5)	2.8
III	22.1	82% -75 μm	330	17 (2.5)	1.9

5.5 Results and Discussions

5.5.1 GRG Test

For each stage, a metallurgical balance is presented in Appendix C and the overall metallurgical balance is calculated based on concentrate assays of the three stages and the last stage tail assays (more accurate because of its fineness and removal of GRG). Concentrates are assayed to eliminate any 'nugget' effect.

Figure 5-2 shows the size distribution of the feed to the three stages. The F_{80} for each stage were 310, 90, and 61 μm, respectively.

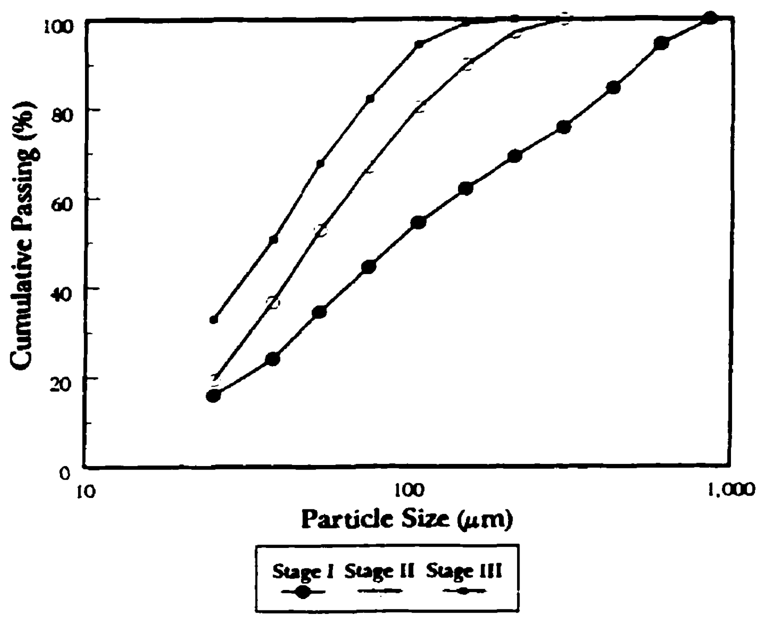

Figure 5-2 Cumulative passing of the feed for three GRG stages

The metallurgical balances of the three stages are presented in Appendix C (Tables C-1 to C-3, p. 119). Overall results are presented in Table C-4 (p. 120). Assay consistency can be assessed by comparing the tail grade of stage I, 3.3 g/t, to the calculated head of stage II, 2.8 g/t; and the tail grade of stage II, 1.8 g/t, to the calculated head of stage III, 1.9 g/t. Agreement is fair for the first comparison and good for the second. Assays are also consistent for the tails fractions and trend consistently for the concentrates (e.g. from 29.0 to 397.5 g/t for size fractions of stage III). It is concluded that assays can be considered reasonable.

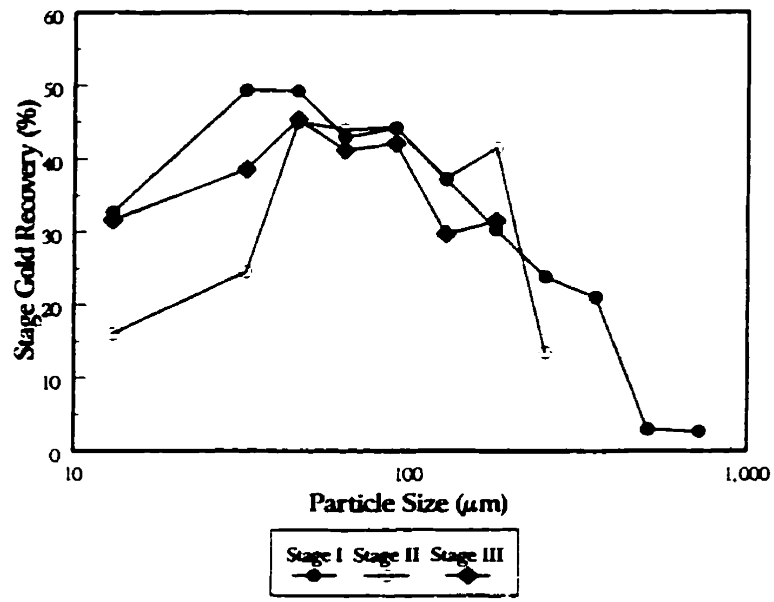

Figure 5-3 Size-by-size recoveries for each stage

Figure 5-3 shows gold size-by-size recovery for the three stages. The apparently lower recoveries of the two finest size classes for stage II could be anomalous and due to concentrate assays that are too low, but evidence is inconclusive. The lower recoveries above 212 μm are consistent, and indicate that the New Britannia ore contains very little coarse gold.

Figure 5-4 cumulates gravity recovery in two different ways. First, it is cumulated from the coarsest to the finest size class - i.e. as a cumulative percent retained. The last point to the left of each curve is the total recovery, for which a minimum particle size of 13 μm was arbitrarily chosen. Second, recovery is cumulated from stage I to stage III. Thus the highest point on the highest curve (stage I to III) shows the total GRG content, 75%. All curves show that there is very little coarse gold: only 4% of the gold in the ore is coarser than 212 μm. There is a substantial amount of gold below 25 μm, 11% of the total gold ore and 15% of the GRG.

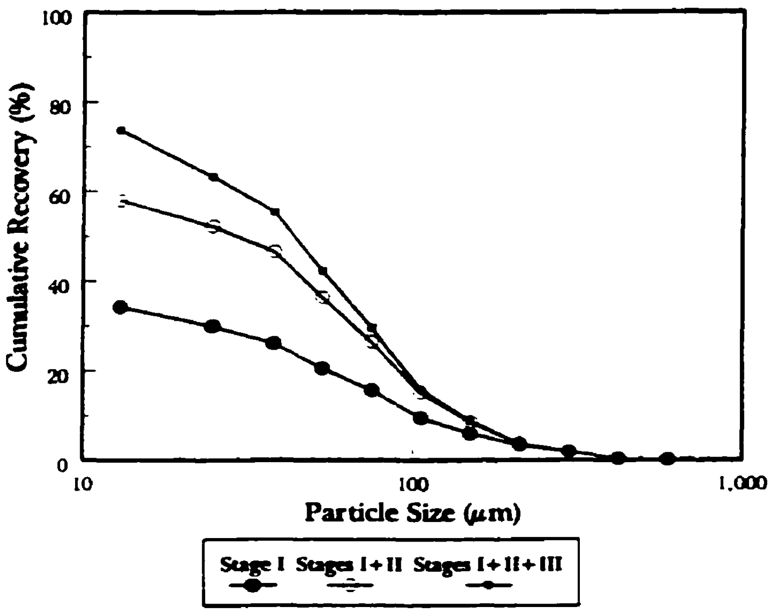

Figure 5-4 Cumulative recoveries for each of the three stages

Table C-4 shows that 38% of the gold was recovered in stage I, 22% in stage II, and 15% in stage III. In other words, of a feed of 4.6 g/t, 1.7 g/t was recovered in stage I, 1.0 g/t in stage II, and 0.7 g/t in stage III; 1.2 g/t was not gravity recoverable.

Laplante (1996) reported that the lowest GRG content was found to be 25% and the highest 94% among thirty-eight samples of gold ores tested at McGill for possible gravity recovery. The average GRG content was 63% with a standard deviation of 19%. Therefore, New Britannia ore has an above-average GRG content.

5.5.2 Grinding Circuit Survey

Detailed results are listed in Appendix C (Table C-5 to C-10, pp. 121-122) which provides the grade, GRG content, and the non-GRG grade of all samples.

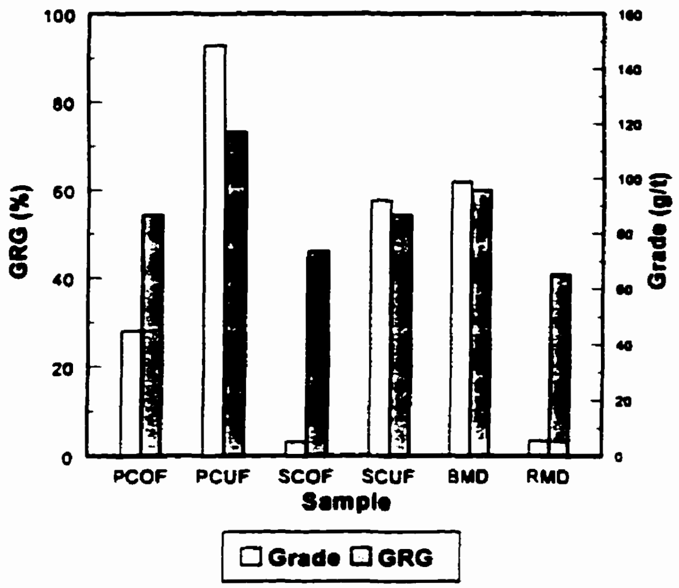

Figure 5-5 Grade and GRG content of the -850 μm fraction of grinding circuit streams

Figure 5-5 shows that GRG contents of all the streams are very high, from 40.7% for the rod mill discharge to 73.2% for the primary cyclone underflow. Grades are extremely variable, from 5.1 g/t for the secondary cyclone overflow to 148.6 g/t for the primary cyclone underflow.

Gold size-by-size distribution in Figure 5-6 shows that the highest gold distribution is around 37 μm, unlike Mineral Hill Mine which is 100 μm (Figure 3-8). This further confirms that gold is finely disseminated in New Britannia ore.

The size-by-size recovery by LKC of three possible centrifuge feeds, primary cyclone underflow, ball mill discharge, and secondary cyclone underflow, is shown in Figure 5-7. It decreases with increasing particle size for the two cyclone underflows, whereas the ball mill discharge shows a relatively size independent recovery, but with significant noise.

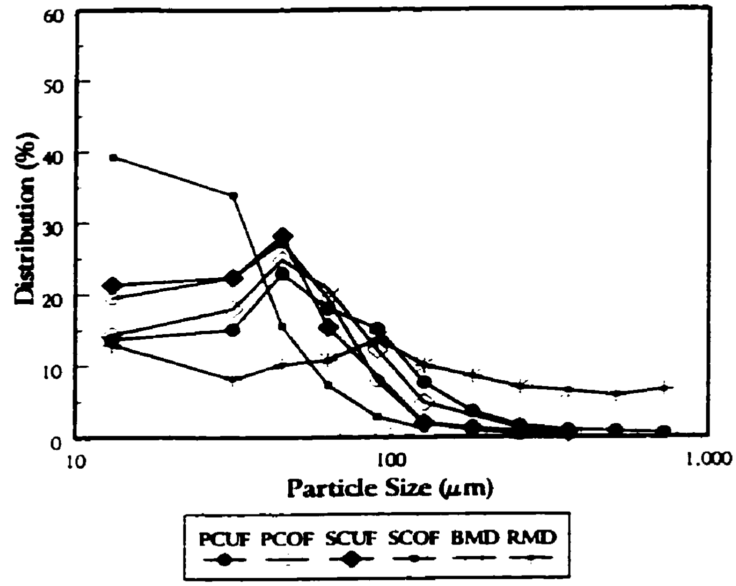

Figure 5-6 Gold size-by-size distribution of the -850 μm size classes of grinding circuit streams

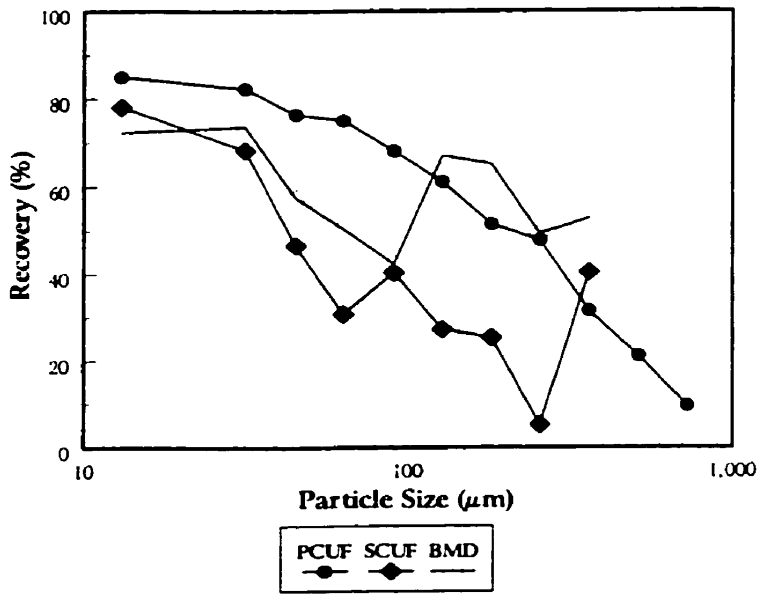

Figure 5-7 Size-by-size gold recovery by LKC of the -850 μm size classes of grinding circuit streams

Based on the above information, results of each stream is discussed in details in the following sections (Xiao and Laplante, 1997c).

Rod mill discharge (RMD)

Rod mill discharges typically contain 20 to 30% GRG. A high GRG content, 40.7%, indicates that there is at least that much GRG in the ore. With further grinding by ball mill, GRG content will be much higher in other streams. In this case, the high GRG content is not due to coarse gold, but to the relatively fine size distribution of the RMD, 46% -75 μm and a P_{80} of about 300 μm.

Ball mill discharge (BMD)

The ball mill discharge contains a significant amount of GRG, 60%, mostly below 105 μm, as was the GRG in the test on the ore itself. Its fine distribution implies that the ball mill has excess capacity and its discharge could be fed to the Knelson without screening. This could require pumping of the gravity concentrate and tails, which is a significant drawback.

Primary cyclone underflow (PCUF)

This stream is the best target for gravity recovery, with 73% GRG and the highest grade, 148.6 g/t. As expected from the size distribution of GRG (Figure 5-4), there is very little coarse gold recovered. In fact, whereas 51% of the mass is above 212 μm, only 3% of the gold is, and even less of the GRG. Clearly, this stream would be an even better candidate for gravity recovery if screened around 200 or 300 μm, rather than the typical 1800 μm.

Primary cyclone overflow (PCOF)

The high GRG content, 54%, and grade, 44.8 g/t, are highly unusual for cyclone overflows, but have been observed elsewhere (e.g. Casa Berardi, Woodcock, 1994) for primary cyclones whose overflow feeds secondary cyclones. The primary cyclones act as crude sizers to reject oversize (possibly because apex diameters are too small), and the fine separation is effected in the secondary cyclones.

Secondary cyclone underflow (SCUF)

This stream has a relatively high grade and GRG content, 92.0 g/t and 54%, which makes it the second choice feeding gravity unit without screening. However, it contains much less GRG than the primary cyclone underflow, and is therefore not as attractive a target.

Secondary cyclone overflow (SCOF)

This sample is very fine, 86% -75 μm, and has a very similar grade to that of the rod mill discharge, 5.1 g/t. The relatively high GRG content (for a SCOF), 46%, suggests that although much of the gold has been overground by the time it reports to the secondary cyclone overflow, it still remains coarse enough to cause problems in the cyanidation circuit.

All six samples have relatively higher GRG contents than what is normally encountered. The potential of gold gravity recovery in New Britannia Mine is very high; about 50% of gold could be recovered by treating the primary cyclone underflow with a gravity centrifuge.

5.6 Plant Trial

Plant trials were performed at the New Britannia Mine to test gold gravity recovery by comparing 20-in Knelson Concentrator (KC) to a Falcon 21-in SuperBowl (SB). Both primary and secondary cyclone underflows were tested, but only the week-long trial on primary cyclone underflow obtained stable and comparable results and will be discussed in the following section (Jean, 1998).

5.6.1 Falcon SuperBowl

The SB plant trial was performed by P. Jean on two thirds of the feed rate of the KC on a bleed of the primary cyclone underflow, for four days, from November 17 to 20, 1997. The rationale for using a lower feed rate with the SB for the comparative work is that the 21-in SB has two thirds the fluidized inner surface of the 20-in KC. A 21-in SB was operated under a fluidizing water flow of about 208 L/min (55 USGPM, the lower limit) with a cycle time of 2 hours. The concentrates and tails samples of three cycles were combined for assaying and calculation, each data set making up one test. As the feed of SB was sampled before the screen, it was not the true sample, and the SB head grade was therefore calculated based on its concentrate and tails assays and masses.

Table 5-3 Summary of Falcon 21-in SuperBowl test results (from Jean, 1998)

Test	Feed		Concentrate			Tails
	Rate (t/h)	Grade (g/t)	Weight (lbs)	Grade (g/t)	Recovery (%)	Grade (g/t)
SB-1	8.5	41.5	123.1	11542	33.6	27.4
SB-2	7.7	32.2	131.6	9684	42.6	18.5
SB-3	9.1	41.5	146.9	13411	43.7	23.3
SB-4	8.7	51.4	136.2	13518	34.4	33.6
SB-5	9.3	54.5	148.1	13103	31.9	37.0
SB-6	10.4	38.7	148.2	8775	27.0	28.1
SB-7	10.9	47.3	145.2	20441	47.8	24.7
SB-8	10.8	56.9	140.1	23164	44.0	31.9
SB-9	7.4	64.8	136.2	19381	46.0	35.0
SB-10	6.5	77.5	139.0	27224	62.7	28.8
SB-11	6.0	55.2	131.9	11213	37.1	34.6
SB-12	7.1	45.3	129.9	12451	42.0	26.1
SB-13	8.3	44.9	133.5	12623	37.7	27.8
SB-14	7.7	35.0	135.3	12544	52.7	16.5
Average	8.5	49.0	137.5	14934	41.7	28.1

Table 5-3 summarizes the plant week-long trial results for Falcon SB. The SB recovered 42% of the gold in its feed, with a standard deviation of 9%. This stage recovery can be translated into a plant recovery. Using the published data of a 2000 st/d and a head grade of 5.2 g/t (Halverson et al, 1996), plant recovery was equal to:

$$\text{Plant gold recovery} = \frac{137.5\ lbs \bullet 0.4536\ kg/lbs \bullet 14934\ g/t}{2000\ st/day \bullet 6\ hours / 24\ hours/day \bullet 907.2 kg/st \bullet 5.2\ g/t} = 40\%$$

The 40% recovery is more than half of the GRG in the ore, which was measured at 74.6% (section 5.5.1).

5.6.2 Knelson Concentrator

A 20-in CD KC was used to treat a bleed of primary cyclone underflow from September 15 to 18, 1997. The KC was operated under a fluidizing water flow of 265-284 L/min (70-75 USGPM). Other operating conditions and calculation were similar to those of the Falcon SB. Overall results are presented in Table 5-4.

Table 5-4 Summary of 20-in Knelson test results (from Jean, 1998)

Test	Feed		Concentrate			Tails
	Rate (t/h)	Grade (g/t)	Weight (lbs)	Grade (g/t)	Recovery (%)	Grade (g/t)
KC-1	15.8	58.3	130.3	25574	30.1	40.8
KC-2	11.9	43.5	136.2	16734	36.7	27.4
KC-3	12.3	39.1	139.2	14605	35.2	25.4
KC-4	12.8	40.8	116.3	11419	21.2	32.2
KC-5	12.4	77.2	121.7	27615	29.3	54.5
KC-6	12.4	53.8	159.2	12671	25.3	40.1
KC-7	12.4	52.1	211.4	9307	25.4	38.7
KC-8	12.4	46.0	183.2	10490	28.2	32.9
KC-9	11.1	39.1	207.0	10205	37.8	24.3
KC-10	12.3	39.1	161.9	12928	36.3	25.0
KC-11	12.3	44.9	147.3	13912	31.0	30.9
KC-12	11.4	39.4	153.2	15445	39.6	23.7
Average	12.6	45.6	155.6	15074	31.3	32.9

The Knelson, operated at a higher feed rate than the 21-in SB, recovered less gold, 31% of its feed, into a concentrate of very similar grade, 15074 vs. 14934 g/t. However, the higher feed rate yielded a higher overall gold recovery, despite the lower stage recovery:

$$\textit{Plant gold recovery} = \frac{155.6\ lbs * 0.4536 kg/lbs * 15074\ g/t}{2000\ st/day * 6\ hours/24\ hours/day * 907.2\ kg/st * 5.2\ g/t} = 45\%$$

When recovery is regressed as a function of feed rate and unit type, the following regression is generated:

$$R = 55.5 \pm 18.5\% - 1.4 \pm 1.6Q - 5.0 \pm 5.6x$$

R: gold recovery, %

Q: feed rate to recovery unit, t/h (dry)

x: dummy variable, (x=0 for the 21-in SB, x=1 for the 20-in KC)

The regression has no significant parameter (even at a low confidence level, such as 80%), and predicts a higher recovery for the SB of 5.0% over the KC. This improvement has a statistical error of 5.6%, and cannot therefore be considered significant. It does suggest a slight advantage for the SuperBowl; additional work is clearly warranted.

A water saving cone (WSC) with an experimental 'retainer ring' was also tested with the 20-in Knelson, and fragmental results suggested a higher gold recovery, but cannot be directly compared to the above data, as a shorter recovery cycle, 1 hour, and average feed rate, 10.6 st/h, were also used. Because of time limitations, no one-week trial of the WSC was completed. This is unfortunate, as the WSC is rapidly becoming the standard for Knelson operation.

The SuperBowl and Knelson units produced very similar performances (i.e. they were not statistically different, even at a low confidence level). The methodology used, with the lower feed rate to the SB, was very questionable, because both units have a similar size and, with the Knelson WSC, a similar water consumption.

Because the two units are similar, it may well be argued that a fair comparison should be based on similar operating conditions - i.e. feed rate. This feed rate should approach realistic plant operation, which calls for maximum gold production, obtained at a very high feed rate. It can thus be argued that both units were tested at a feed rate which was too low to assess their full impact on overall recovery. In other words, because the economic and metallurgical impact of these units is maximized at maximum throughput, to maximize the mass of gold recovered, comparisons at lower feed rates can be misleading. Despite these limitations, test work did demonstrate that the SB was similar to the Knelson in performance at feed rates below optimum.

5.6.3 Detailed SB and KC Performance

To gain additional insight into the comparative performance of the two centrifuge units, samples of feed and tails were extracted at the end of each test series, and shipped to McGill for further processing. The samples were screened at 850 μm, and the undersize processed with a 3-in Knelson at a feed rate of 400-500 g/min and fluidizing water pressure of 27 kPa (4 psi), using the protocols outlined in sections 3.4.1 and 3.4.2. Size fractions were assayed at Spectrolab, Rouyn-Noranda. Table 5-5 details experimental conditions and overall results; Appendix C (Table C-11 to C-14, pp. 123-124) and Figures 5-8 and 5-9 present size-by-size data.

Table 5-5 shows the extremely low stage gold recovery for the KC (20%), which averaged 31% for the full week trial (Table 5-4). This might be due to its relatively low feed grade, 40.8 g/t compared to that of the SB feed, 50.0 g/t. The laboratory test work yielded similar results to the plant trial, i.e. the SB recovered more gold from its feed. Such a direct comparison is unfair to the KC, as its higher overall gold recovery (because of its higher feed rate) caused a higher drop in the circulating load of gold in the grinding circuit. This lower load corresponded to a decrease in the GRG content, from 69% in the SB feed to 59% in the KC feed.

Table 5-5 Sample descriptions and the overall results

Sample	-850 μm Mass (%)	-850 μm Mass (kg)*	-850 μm (g/t)	GRG (%)	Stage Gold Rec. (%)	Stage GRG Rec. (%)
SB-14 Feed	64	6.39	50.0	69	48	66
SB-14 Tails	86	8.56	25.9	49		
KC-12 Feed	74	7.34	40.8	59	20	40
KC-12 Tails	89	8.81	32.5	40		

* Processed by LKC

Total gold recoveries are based on the differences between feed and tails gold grades, whereas GRG recoveries were calculated assuming that all gold recovered is GRG, and the unit's tails GRG content is that recovered by the 3-in LKC (as in section 3.5.1). There appears to be some scatter in the data for both units, in the coarse range for the SB, presumably a nugget effect, and in the fine range for the KC, possibly screening errors (e.g. a tear in the 25 μm concentrate screen, the most probable cause).

Despite this scatter, a clear picture emerges. For the SB, gold recovery is maximum at intermediate particle size, and drops significantly below 37 μm and above 425 μm. GRG follows a similar trend, but at a higher recovery. For the KC, gold recovery is around 30 to 40%, except above 300 μm.

GRG recovery is constant and from 50 to 70% over the full size range, indicating that the loss of gold recovery above 300 μm is due to a lack of liberation (i.e. most +300 μm gold in the KC feed is not gravity recoverable). It is surprising that the KC outperformed the SB below 37 μm (on account of the higher Gs of the SB), and further test work would be needed to confirm this finding.

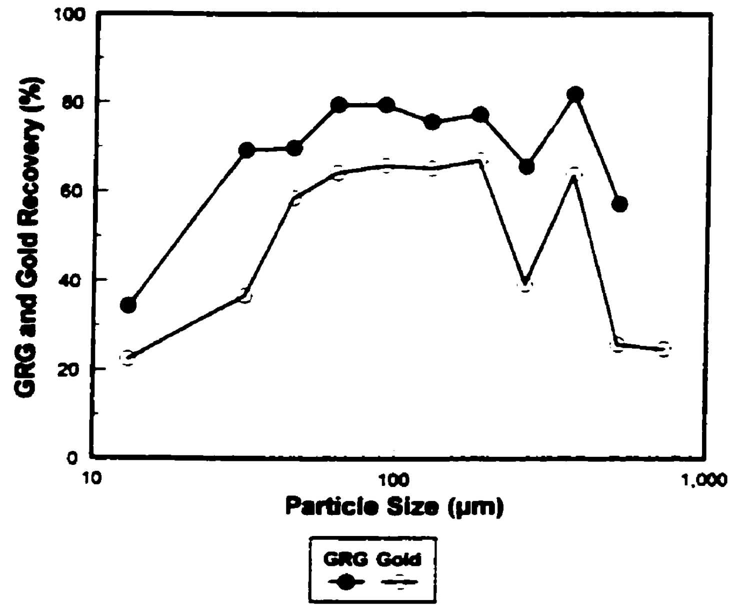

Figure 5-8 Size-by-size GRG and gold stage recovery of the 21-in SuperBowl

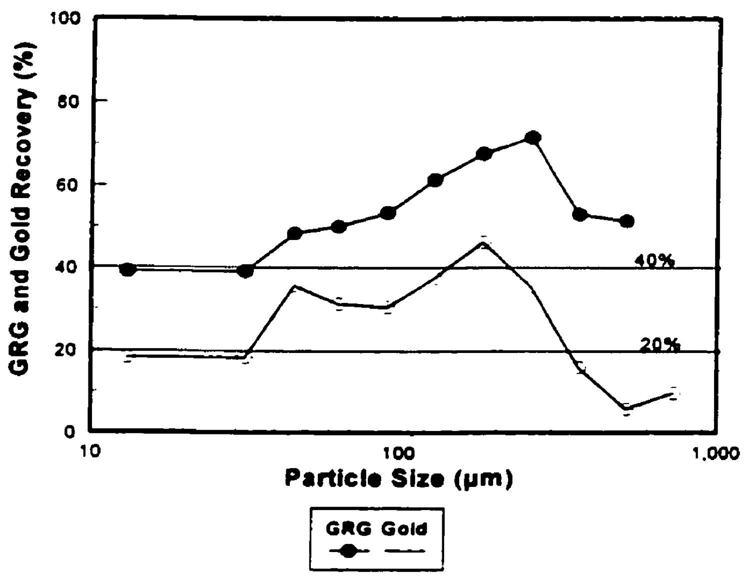

Figure 5-9 Size-by-size GRG and gold stage recovery of the 20-in Knelson (20%: average gold recovery; 40%: average GRG recovery, from Table 5-5)

Figure 5-9 is also surprising in that both total and gravity-recoverable gold recoveries, calculated on a size-by-size basis, are above what was calculated from the overall total and gravity-recoverable gold content (i.e. Table 5-5). Upon inspection, the problem was traced to the feed sample, which in fact is the screen feed. The -850 μm fraction of this sample is much coarser than that of the KC tails and as a result the impact of the +212 μm fraction is exaggerated. A similar problem was noted with the SB feed. To correct both problems, the feed total and gravity-recoverable gold content was recalculated using the size distribution of the tails of each respective units (in fact assuming that size fractions in the screen feed had the correct total and gravity-recoverable gold content). Table 5-6 shows the new estimates of total and gravity-recoverable gold content for the SB and KC feeds, as well as the total and gravity-recoverable gold recoveries based on these new estimates (old estimates are shown in brackets).

Table 5-6 Corrected estimates of the overall results

Sample	-850 μm (g/t)	GRG (g/t)	GRG (%)	Stage Gold Rec. (%)	Stage GRG Rec. (%)
SB-14 Feed	56.1 (50.0)	39.7	71 (69)	54 (48)	69 (66)
KC-12 Feed	48.1 (40.8)	28.8	60 (59)	32.5 (20)	55 (40)

All estimates are up from the previous ones, more so for KC, which displayed more size effects (gold and GRG content) than the SB. This is an important finding as screen feed are often used to represent KC feed samples, because of the inaccessibility of the screen undersize (the true KC feed) as was the case for the Casa Berardi tests. This is particularly true with the screen supplied by Knelson Concentrators Inc. for their 30-in units.

5.6 Summary

A GRG test confirmed that 74.6% of gold in New Britannia ore is gravity recoverable, more than 78% of it below 105 μm, an indication of abundant finely disseminated gold that can nevertheless be liberated.

Both the GRG test and the data generated with the six samples from the grinding circuit confirmed the very good potential of the New Britannia ore for gravity recovery. The primary cyclone underflow is the most interesting target because of its extremely high GRG content and grade.

Plant trials yielded very good results, as more than 50% of the GRG was recovered from a bleed of the primary cyclone underflow. The SB and KC achieved performances that were statistically not significantly different, despite the fact that they recovered different amount of gold from the feed and circuit, due to their different feed rates. However, units should be compared on the basis of optimum economic impact, which occurs when equivalent units (in capacity) are operated at maximum feed rate, to maximize gold production. This was not the case in this test work.

CHAPTER 6

TESTING A FALCON 4-IN SUPERBOWL MODEL

6.1 Introduction

Several methods can be used to evaluate the efficiency of gravity separators. The conventional method includes first optimizing operating parameters, then feeding an ore to the separator, analysis of the products for the valuable minerals, and plotting a curve of concentrate grade (or gold) versus recovery. Most other methods for the assessment of gravity separators are based on the use of synthetic feeds. For example, Diamond Research Laboratory in Johannesburg manufactures a range of plastic markers, using a different colour for each relative density, and extrudes these markers into pellets. These markers have been used very successfully in the assessment of heavy-medium separation. It was believed that markers fine enough to be used in fine separations such as spirals or shaking tables could not be manufactured. Even if they could be manufactured, it was argued that their recovery for re-use and for the identification of the various colours would be difficult (Guest and Dunne, 1985). However, a program of fundamental spiral research undertaken at the JKMRC over the last three years has drawn heavily upon the use of the different coloured density tracers as synthetic feedstocks (Edward et al, 1993).

Walsh and Rao (1988) first used radiotracers to evaluate a compound water cyclone as a fine-gold concentrator. Subsequently, the technique was applied to a Pan American jig, a static wedge wire screen and an elutriator (Walsh, 1989). Clarkson (1990) has also used the technique to full-scale sluicing operations. Walsh and Kelly (1992) applied it again to investigate the performance of a spiral. More recently, Clarkson (1997) used it to the detection of gold traps in a grinding circuit.

For the evaluation of gravity centrifugal concentrators, Buonvino (1994) used magnetite and silica to test a Falcon B6, a fine material concentrator. In the Knelson concentrator overloading test, Huang (1996) chose fine metallic tungsten to mimic the behaviour of gold, as its density (19.3 g/cm^3) is identical to that of pure gold. Magnetite was used as a substitute for the main gangue mineral of gold gravity concentrates, pyrite; the two minerals have approximately specific density, around 5.0, and but magnetite can be easily recovered by both gravity and magnetic separations. Silica, with a specific density of 2.65, was used to mimic the light gangue, the main component of most ores.

The obvious advantage is that synthetic feeds can be totally liberated, and their size distribution, shape, density and grade can be controlled (Guest and Dunne, 1985). This can eliminate the impact of middling particles which have a negative impact on unit performance, especially for gravity separation. Huang also pointed out that, for gold gravity studies, there are at least another three additional benefits for the use of synthetic ores. First, when trying to achieve high concentrate grades with a fixed concentrate mass, the use of gold would be cost prohibitive. Second, the risk of contamination for parallel work with much lower head grades is eliminated. Third, the use of a tracer with a controlled shape can shed light on the behaviour of gold particles with much lamellar shapes.

In this study, the same materials (tungsten, magnetite, and silica) were chosen to compose the synthetic ores to evaluate the performance of a new gold centrifugal concentrator, the laboratory 4-in Falcon SuperBowl Model (SB4).

6.2 Objectives

The objectives of this program was to explore the effect of fluidization water flowrate and feed capacity on the SB4 performance with different gangue size and density under a 120 Gs centrifugal field. The possibility of treating flash concentrates was also evaluated.

6.3 Feed Preparation

6.3.1 Tungsten

The tungsten was obtained from Zhuzhou Cemented Carbide Works of China and in the size range of -600 +212 μm, with irregular shapes. To obtain the desired size distribution, the tungsten was first screened; coarse fractions were ground in a ball mill and then screened from 600 μm down to 25 μm. Each size fraction was then stored separately. The -25 μm tungsten fraction was further classified with a Warman cyclosizer to remove -8 μm particles to reduce the experimental error. The size distribution of this fraction is shown in Table 6-1.

Table 6-1 Size distribution of the -25 μm tungsten fraction

Cone	Size (μm)	Wt. (g)	Wt. (%)
1	-25+19	19.70	30
2	-19+12	25.00	38
3	-12+8	6.88	11
4	-8+6	2.74*	4
5	-6+4	0.68*	1
	-4	10.15*	16
	Total	65.15	100

(*: removed for the sample)

6.3.2 Magnetite

The magnetite was obtained from a 40 kilo sample of cobber concentrate[1] sample

1. A cobber concentrate is a rougher concentrate produced with a low intensity magnetic separator (wet drum) in a taconite flowsheet.

of Iron Ore Company of Canada Limited. The sample was first processed with a shaking table, its middlings were then cleaned with a hand magnet; both shaking table concentrate and magnetic middlings, about 16 kilos, were screened on a Rotap from 600 μm down to 25 μm; part of the coarse fractions were ground with a Sepor 30 cm length x 30 cm diameter rod mill and screened repeatedly to obtain the desired size distribution. Finally, each fraction was cleaned with a hand magnet and the non-magnetic fraction rejected to ensure an easy separation of tungsten from magnetite. The specific gravity for each size class was measured, and was in excess of 4.7 for all size fractions. This was deemed adequate to mimic most sulphides, whose density ranges from 4.1 to 5.0 (pyrite). Denser sulphides such as arsenopyrite (6.1) or galena (7.5) would require separate testing. For galena (or cassiterite), ferrosilicon would provide a convenient magnetic substitute.

Table 6-2 Size distribution of fine magnetite or silica synthetic feed

Size (μm)	Total Wt. (%)	Tungsten (g)	Magnetite/Silica (g)	Tungsten (%)	Tungsten Dist'n (%)
300	3.7	15.8	356	4.25	10.6
212	7.6	18.5	744	2.43	12.3
150	14.1	24.5	1386	1.74	16.4
106	15.2	21.7	1493	1.43	14.5
75	13.4	17.1	1327	1.27	11.4
53	11.2	12.0	1104	1.08	8.0
38	11.5	14.8	1133	1.29	9.8
25	12.3	13.5	1219	1.09	9.0
-25	11.0	12.1	1088	1.10	8.0
Total	100.0	150.0	9850	1.50	100.0

As one purpose of this study is to evaluate the performance of the SB treating a flash flotation concentrate, which is mostly composed of sulphides, the size distribution of a flash concentrate from mine Lucien Beliveau (Putz, 1994) was chosen to be the size distribution of fine magnetite and tungsten synthetic feed. The overall tungsten grade for different feeds was 1.5% with a total mass of 10 kilos. The size-by-size weight and distribution of both tungsten and magnetite are shown in Table 6-2.

6.3.3 Silica

Different size fractions of silica were prepared by grinding and screening silica sand and flour obtained from Unimin Canada Ltd. A relatively coarse distribution was chosen for the coarse silica and tungsten synthetic feed, which was based on Woodcock's (1994) first GRG test with the Alaska-Juneau (AJ) ore, as shown in Table 6-3. A fine silica-tungsten feed was also prepared, of a size distribution identical that of magnetite in Table 6-2.

Table 6-3 Size distribution of coarse silica synthetic feed (100% -850 μm)

Size (μm)	Total Wt. (%)	Tungsten (g)	Silica (g)	Tungsten (%)	Tungsten Dist'n (%)
600	15.0	13.4	1491	0.89	8.9
425	14.9	19.6	1474	1.31	13.1
300	11.5	20.8	1124	1.81	13.8
212	8.4	16.4	828	1.95	11.0
150	8.1	19.0	788	2.36	12.7
106	6.3	14.0	612	2.23	9.3
75	6.3	12.3	622	1.94	8.2
53	4.5	8.6	436	1.93	5.7
38	4.9	6.6	482	1.34	4.4
25	4.2	4.0	417	0.94	2.7
-25	15.9	15.3	1576	0.96	10.2
Total	100.0	150.0	9850	1.50	100.0

6.4 Test Procedure

A shown test procedure is in Figure 6-1. The synthetic ore was first processed with the SB4; its concentrate was then dried and screened size-by-size. Each size fraction was treated by a Mozley Laboratory Separator (MLS) to separate tungsten from silica or magnetite. For the magnetite feed, tungsten recovered by MLS was further cleaned with a hand magnet to remove the residual magnetite.

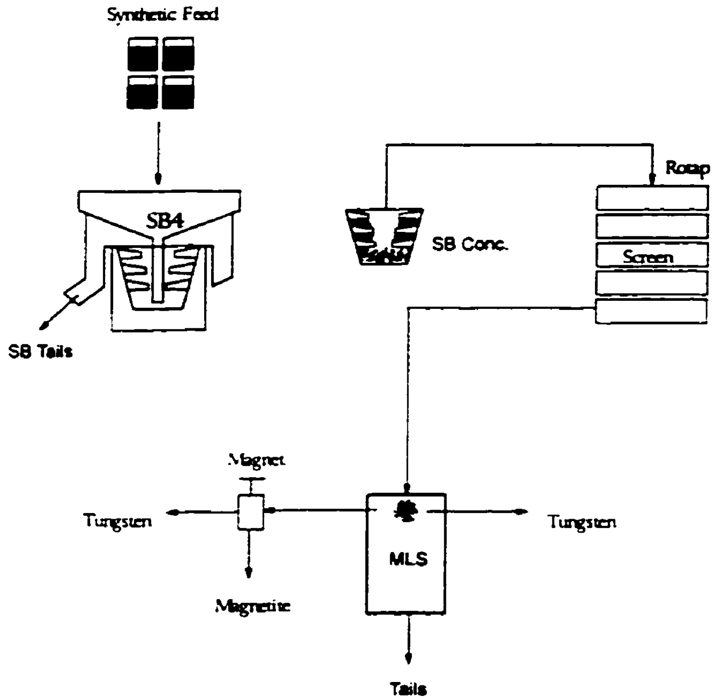

Figure 6-1 Simplified test procedure

6.4.1 Operating the 4-in SuperBowl

The prepared material was blended thoroughly and split into ten 1-kg sub-samples. To avoid tungsten settling to the bottom of the feed tank, because of its density, dry samples were fed to the SB4 manually at identical time intervals. For all tests, feed solids percent was kept at 40%, except for the tests at high feed rate, 5 kg/min, for which a density of 48% solids was used (the maximum water flow that could be handled by the unit without spill). The detailed operating procedure is as follows (adapted from the MODEL SB4 OPERATING GUIDE):

1) Turn on supply water to unit. Ensure it is cleaned of old samples and rotating union drain is closed.
2) Check rotor basket is secure and impeller bolt tight. Install the funnel lid. Put a barrel to collect tails.
3) Start concentrator motor and open fluidizing water valve at the same time. Adjust the fluidizing and slurrying water flow rates. Feed the materials in a given time.
4) After all of the sample has been processed, simultaneously shut down concentrator motor and slowly shut off fluidizing water while the rotor coasts to a stop. (As shutting off the fluidizing water supply too soon will cause the concentrate to pack in the riffles). Conversely, if the fluidizing water supply is shut down too slowly, the concentrate may be flushed out of the riffles and report to tails.
5) Disconnect the power to the SB4, remove the funnel lid and unscrew the impeller bolt in the rotor bottom. Lift out the plastic rotor bowl and carefully rinse the concentrate into a pan.
6) Place a bowl under the rotor shaft, open the valve on the union and rinse any particles from inside the water jacket through the hollow rotor shaft combining it with the bowl concentrate.
7) Thoroughly rinse the tails launder and the entire machine to collect all the sample for the reuse. After settling for 10 hours, decant and dry the tails.

6.4.2 Analysis of 4-in SuperBowl Concentrate

The SB4 concentrate was decanted, dried, and screened on a Rotap for 20 minutes from 600 μm down to 25 μm. The amount of tungsten in each size fraction was determined with a Mozley Laboratory Separator (MLS) and a hand magnet. To confirm the reliability of this method, preliminary tests were performed with different material combinations at different sizes. The MLS operating parameters and corresponding test results repeated very well at the same operating conditions; at least 97% tungsten was recovered with little variance by this separation method.

Table 6-4 MLS operating parameters

Size (μm)	Sample Tungsten	Water Flowrate (L/min)	Table Shape	Frequency (rpm)	Amplitude (cm)	Slope (°)
>106	Silica	0.5	V	70	2.5	1.75
	Tungsten	1.5				
<106	Silica	2.5	Flat	90	3.5	
	Tungsten	3.0				

Table 6-5 MLS preliminary test results

Size (μm)	Sample Tungsten: 10 (g)	Test No.	Tungsten (g)	Recovery (%)	Average Rec. (%)
+300-425	Silica: 20	1	9.91	99.1	99.2
		2	9.92	99.2	
		3	9.92	99.2	
	Magnetite: 10	1	9.90	99.0	99.1
		2	9.91	99.1	
		3	9.92	99.2	
+53-75	Silica: 10	1	9.80	98.0	98.3
		2	9.84	98.4	
		3	9.86	98.6	
	Magnetite: 10	1	9.65	96.5	97.4
		2	9.75	97.5	
		3	9.81	98.1	

6.4.3 Test Arrangement

Sixteen tests were performed with three types of synthetic feeds at three or four different fluidizing water flowrates (10, 15, 20, 25 L/min) and three different feed rates (1, 2, 4 or 5 kg/min). First, the optimum fluidization water flowrate was determined for each sample with a feed rate of 1 kg/min. The feed rate was then tested at the optimum fluidization water flowrate. This approach minimized the required number of tests, an important consideration, as the feed samples had to be re-used, which would eventually lead to loss of tungsten and inaccurate test results.

6.5 Results and Discussions

For each test, size-by-size concentrate mass and tungsten content were recorded, and the overall tungsten recovery and concentrate grade were calculated (all are shown in Appendix D). Because of the high tungsten recoveries and screening errors, some size fractions had more than 100% recovery, making size-by-size tungsten recovery calculations impossible (but redundant).

6.5.1 Observation of the Concentrate Bed

According to the SB4 user manual, the optimized fluidizing water flowrate is obtained when concentrate collected in the riffles just start to slump out of the lower riffle, as it should not be packed hard. It also suggests to use as low a water backpressure as possible.

During each test, the concentrate bed was inspected and its formation was recorded; it is shown in Table 6-6. Concentrate bed loosened with increasing fluidization water flowrate, whereas feed rate had little effect. The optimum water flowrate for these three samples (fine silica, coarse silica, and fine magnetite) would be 10, 15, and 20 L/min, respectively, according to the bed sloughing criterion suggested in the operating guide of the user manual.

Table 6-6 Observation of 4-in SuperBowl concentrate bed

Sample	Water Flowrate (L/min)				Feed Rate (kg/min)		
	10	15	20	25	1	2	4/5
Fine Silica	Slump	Slump	No Bed	-	Slump	Slump	No Bed
Coarse Silica	Packed Soft	Slump	No Bed	-	Slump	Slump	Slump
Fine Magnetite	Packed Hard	Packed	Packed Soft	Slump	Packed	Slump	Slump

Lancup (1998) had tested the effect of feed rate, and found that the concentrate bed was packed harder with increasing feed rate; no such relationship was found in this work. The difference may be due to the following reasons: (i) Lancup's feed solids percent was much higher than that used in this work, about 60-70%, and a lower slurrying water flowrate was used; (ii) step 4 in the operating procedure, which is a key factor affecting the concentrate bed formation, was performed differently.

Visual observation of the concentrate bed showed that coarse tungsten particles were on the bed surface of both the fluidized and non-fluidized fractions, when concentrate was packed hard (the surface of slumping concentrate beds could not be examined). This was also observed by Lancup (1998), and Huang (1996), for a 3-in Knelson. Huang also used a separable bowl to recover the 3-in KC concentrate, once frozen, and analyse its content. The innermost layer was found to contain most of the tungsten at a very high grade, 87.2%. Buonvino (1994) examined the dynamics of solids bed formation of a B6 Falcon Concentrator -- a non-fluidized separating rotor, and found that the bed builds up quickly with a non-selective recovery and that recovery of heavy particles occurs predominantly on the surface via capture sites.

6.5.2 Effect of Fluidization Water Flowrate

Figures 6-2 and 6-3 show the tungsten recovery and concentrate mass of the three synthetic feeds at different fluidization water flows. For the coarse and fine silica samples, tungsten recoveries were very high, more than 97%; both recovery and concentrate mass decreased slightly with increasing water flowrate. The lowest fluidizing water flow tested yielded the highest recoveries, but must be very near the optimum flow, given the very high recoveries achieved, 97-99%. As tungsten recovery from the coarse silica sample at 10 and 15 L/min did not change very much (Δ=0.3%), the optimum fluidization water flowrate should be between 10 and 15 L/min. For the fine silica sample, the optimum water flowrate should be between 5 to 10 L/min. The slightly higher tungsten recovery with coarse silica gangue was probably due to the coarser tungsten size distribution (23% -75 μm vs. 35% -75 μm for the fine silica gangue).

The fine magnetite sample showed quite a different behaviour. At fluidization flow rates of below 15 or above 20 L/min, tungsten recovery decreased; in the range of 15 to 20 L/min, the highest recovery was obtained, above 95%, which was about 3% lower than for silica gangue, coarse or fine. The concentrate mass also decreased with increasing water flowrate because of the increasingly looser concentrate bed. Lower recovery coincided with a very tightly packed concentrate bed, suggesting that poor fluidization was the cause of the loss of recovery. Conversely, an excessive fluidization flow, such as 25 L/min with the fine magnetite gangue, caused some of the tungsten to be washed out of the concentrate.

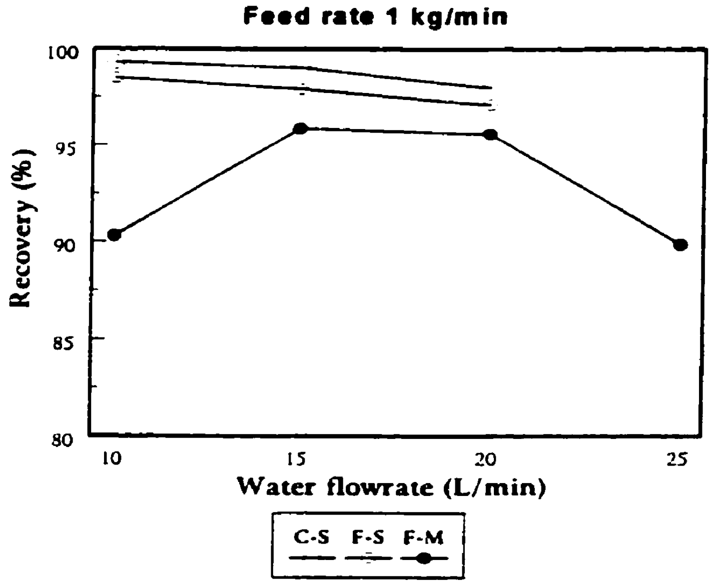

Figure 6-2 Tungsten recovery of the three feeds at different water flowrates (C-S: coarse silica; F-S: fine silica; F-M: fine magnetite;)

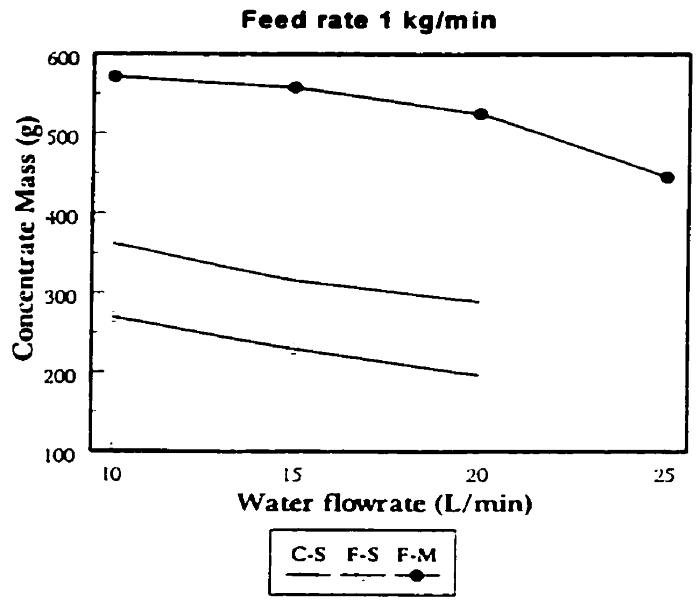

Figure 6-3 Concentrate mass of the three feeds at different water flowrates (C-S: coarse silica; F-S: fine silica; F-M: fine magnetite;)

Figure 6-3 shows that the concentrate mass was twice as high for the fine magnetite than the fine silica gangue. This can be explained by assuming the same concentrate volume was obtained, since the specific gravity of magnetite is twice that of silica - i.e. 5.25 and 2.65, respectively.

Tungsten recovery can be correlated with bed packing to test the manufacturer's suggestion that the optimum fluidization flow corresponds to a concentrate bed that begins to slump at the end of the test. Table 6-6 suggests that the highest recovery is achieved when the bed is softly packed; only when it is packed hard is the flowrate clearly too low. Because gold is lamellar (flaky), and will have a lower terminal settling velocity than tungsten, the tungsten results are likely to predict an optimum flow equal to or above that of gold. The manufacturer recommends a higher flowrate, one that is clearly too high for optimum results.

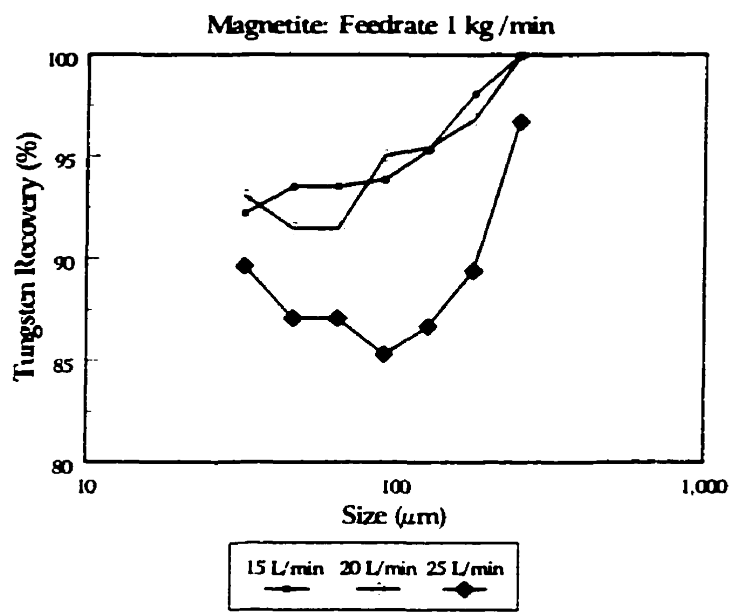

Figure 6-4 Size-by-size tungsten recovery of the fine magnetite feed at different water flowrates

Because of the relatively lower recovery of the fine magnetite feed, its size-by-size tungsten recovery was calculated, and is shown in Figure 6-4. Because of screening errors, the following size classes were combined in the result: -25 and 25-37 μm, 212-300 and 300-425 μm, and an average recovery is used for 37-53 and 53-75 μm fractions. Tungsten recovery decreased for all the size fractions at a water flow of 25 L/min. For the two lower water flows, the overall and size-by-size tungsten recoveries changed very little. The U curve of the 25 L/min test was also obtained by Ling with a 3-in Knelson (1997) and Buonvino with a B6 Falcon (1993). In Ling's work, a similar feed was tested with a variable speed 3-in KC at three different centrifugal fields. The size-by-size tungsten recoveries at 60 and 115Gs all presented a similar U shape at a low fluidizing water flow, 2 L/min, but over a much more narrow size distribution, whereas at high flow rate, the U shape disappeared. Buonvino obtained a relatively narrow middle range (plateau) size-by-size recovery curve by treating magnetite and silica feed with a B6 Falcon Concentrator (a non-fluidized unit).

This information suggests that the 4-in SB operates at least partly under a non-fluidized condition, even at relatively high fluidizing water flow rate (due to its smaller riffled surface) and therefore has a separating behaviour which combines some features of the Knelson Concentrator, and others of the batch Falcon (section 2.3.5).

6.5.3 Effect of Feed Rate

For the fine silica gangue feed, the maximum feed rate could only reach 4 kg/min, whereas the other two samples were processed at a maximum feed rate of 5 kg/min. All three gangues were tested at a water flowrate of 15 L/min, which is equal or very close to the optimum.

Detailed results are in Appendix D. Figure 6-5 shows that tungsten recovery was constant for the coarse silica sample over the full range of feed rate (from 1 to 5 kg/min). For the fine silica feed, the tungsten recovery start to decrease from 97.8% at 2 kg/min to 96.5% at 4 kg/min, a very slight drop. The fine magnetite sample showed a decreasing recovery with increasing feed rate, which means that a high density feed was more sensitive to feed rate changes (the overall drop remains low, only 4%). Laplante et al (1996b) had reported a similar finding for a 3-in Knelson Concentrator.

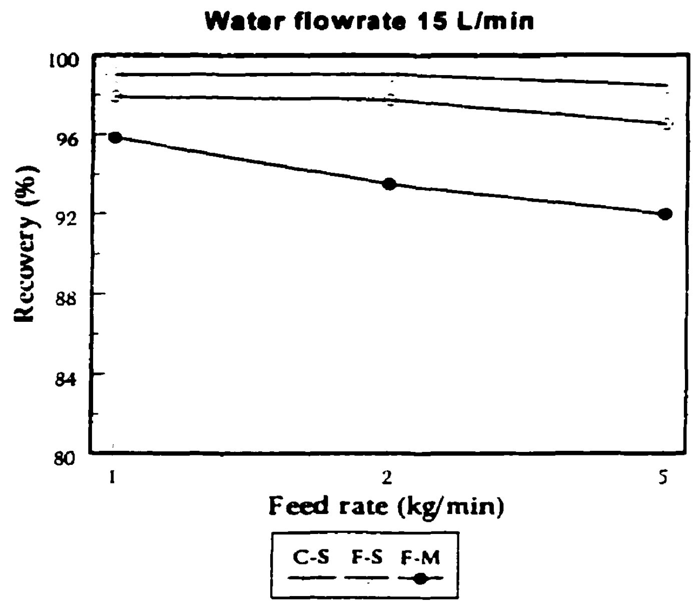

Figure 6-5 Tungsten recovery of the three feeds at different feed rates
(C-S: coarse silica; F-S: fine silica; F-M: fine magnetite;)

Figure 6-6 shows that feed rate had little if any effect on the concentrate mass recovered. Only with the fine magnetite gangue did an increase in feed rate affect concentrate mass, which dropped from 558 to 468 g when feed rate increased from 1 to 5 kg/min. Laplante et al (1994) had reported a similar trend for a B6 Falcon Concentrator tested at the Snip Mine, but the effect was much more significant. The mass recovered is clearly first a function of gangue density, then gangue particle size, and finally fluidization water flow.

Figure 6-7 was obtained by adjusting the data in the same way as Figure 6-4 (grouping of same size classes to decrease the effect of experimental errors). Tungsten

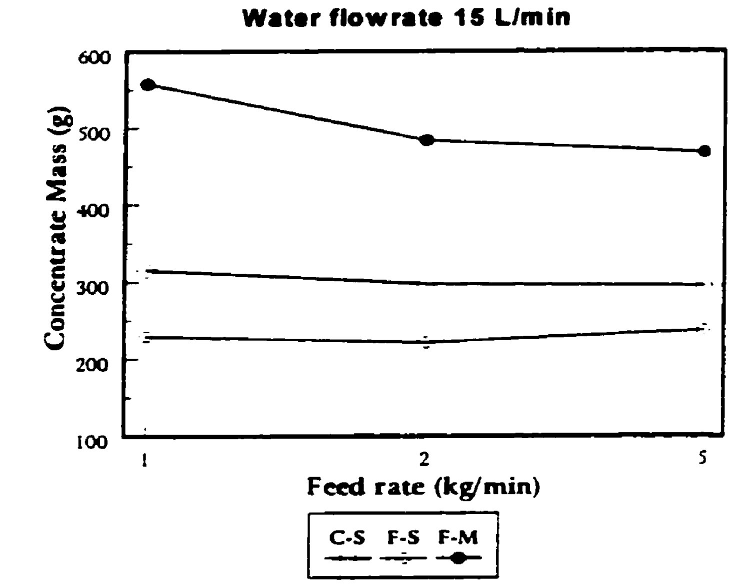

Figure 6-6 Concentrate mass of the three feeds at different feed rates (C-S: coarse silica; F-S: fine silica; F-M: fine magnetite;)

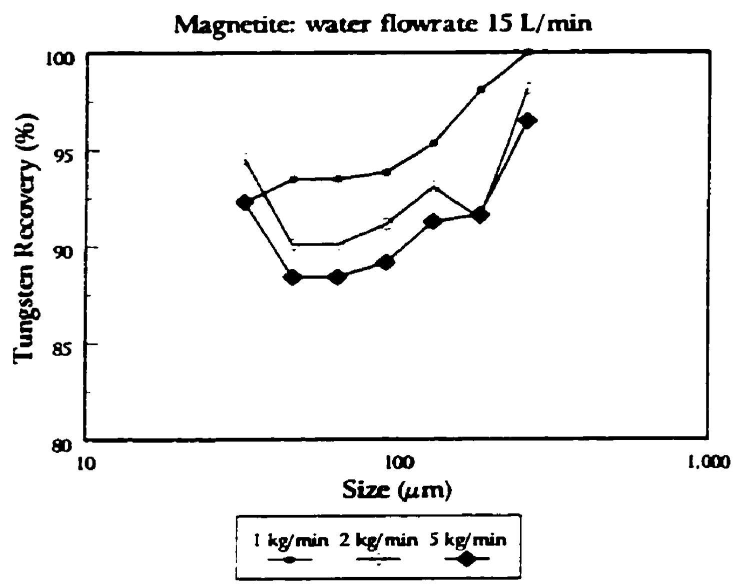

Figure 6-7 Size-by-size tungsten recovery of the fine magnetite feed at different feed rates

recovery decreased for all the size fractions and the two curves at 2 and 5 kg/min were almost parallel. The obvious U shape confirms further the similarities between the batch Falcon and SuperBowl operation, and also suggests that the lower recovery of the finest fraction of the test at a feed rate of 1 kg/min might be caused by experimental error, especially when considering the concentrate mass of this fraction (see Appendix D).

6.5.3 Effect of Gangue Size and Density

As mentioned earlier, the size-by-size tungsten recovery could be obtained for most tests, but the concentrate mass distributions still can reveal the different effects of both gangue size and density on the 4-in SB performance. Figures 6-8 and 6-9 give the mass distributions of the three feeds and their concentrates at a water flowrate of 15 L/min and feed rate of 1 kg/min.

<u>Gangue size</u>

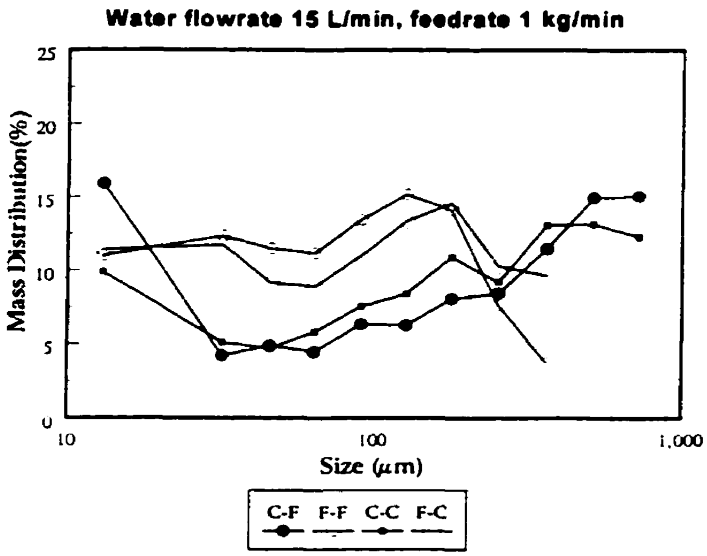

Figure 6-8 Comparison of concentrate size distribution for silica gangue (C-F: coarse feed; F-F: fine feed; C-C: coarse concentrate; F-C: fine concentrate;)

For both fine and coarse silica gangue, the concentrate size distribution was very similar to that of the feed. The small differences between feed and concentrate size distributions can be informative. For fine silica, the coarser fraction (+212 μm) was preferentially recovered, but not at the expense of the finest fraction (-37 μm), but the intermediate size range (37-150 μm). This was even more obvious at a lower fluidization flow (10 L/min, Appendix D, page 127). Very fine (-25 μm) silica rejection did take place with the coarser sample, but not because the coarsest fraction was recovered, as there was slightly less +425 μm material in the concentrate than in the feed. For batch centrifuges, concentration mechanisms are complex, and differ very significantly at the beginning of the recovery cycle (Huang, 1996).

Gangue density

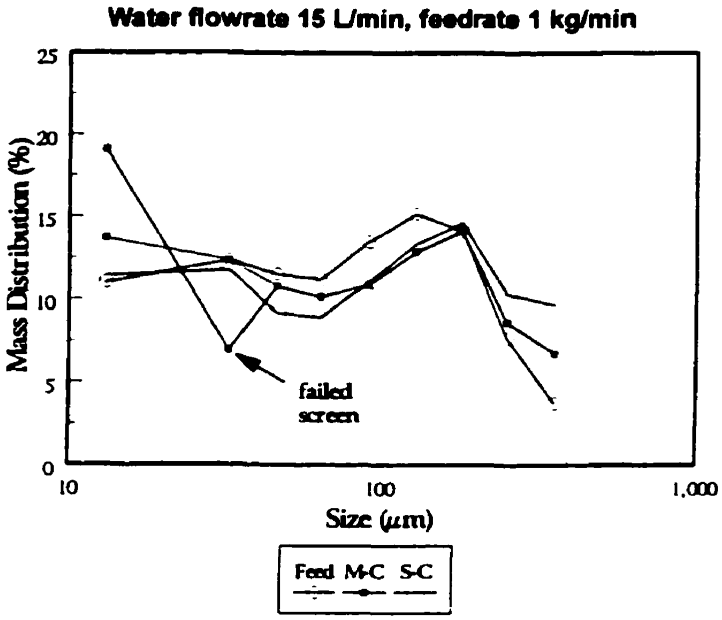

Figure 6-9 Comparison of concentrate size distribution for the fine gangue (M-C: magnetite concentrate; S-C: silica concentrate;) (hatched line: corrected mass for the M-C;)

Figure 6-9 compares fine silica and magnetite feed and concentrate distributions. The fine silica curves have been discussed above. What is striking is the similarity between the magnetite and silica concentrate curves. Note that the mass in the two finest classes of the magnetite concentrate had to be corrected because of screen failure (the 25-37 μm fraction was assumed equal to that of the feed, and the -25 μm mass corrected accordingly). The higher density of the magnetite has favoured recovery of the -25 μm fraction (even when corrected) rather than that of coarse (+212 μm) magnetite.

6.6 Laboratory vs. Plant Performance

Can the performance of the 4-in and 21-in SB units be directly compared? If GRG performance is used (to account for liberation problems) and feed rates are scaled up or down on the basis of constant loading (t/h of feed per m^2 of concentration area), as was suggested for another gravity unit, the Reichert cone (Holland-Batt, 1978), this may be feasible.

With the coarse silica gangue, tungsten recovery was still above 98% at a feed rate of 5 kg/min, which corresponds to a feed rate of

$$\frac{5\ kg/\mathrm{min} * 60\ \mathrm{min}/hour * (\frac{21\ inch}{4\ inch})^2}{1000\ kg/t} = 8.3\ t/h$$

for the 21-in SB, at constant loading (t/h of feed per m^2 of concentration area). Yet, the measured GRG recovery was much lower at New Britannia (76%, Table 5-6), or even Mineral Hill at much lower feed rates (56-82%, Table 3-3). The differences can be explained by:

i) the synthetic feed was 100% -850 μm, whereas the plant feeds were coarser.

ii) the plant feeds contained some sulphides.

iii) the plant SB operating variables were not fully optimized.

iv) gold's size distribution, shape factor and density in plant feeds were not as conducive to recovery as those of tungsten.

v) small differences in geometry (i.e. groove cross-sectional shape) between the 4-in and 21-in SBs make a direct comparison impossible.

All five hypotheses are likely and further test work will be required to assess their respective contributions. Since Laplante et al (1998) found that the performance of a 3-in KC and 4-in SB on flash flotation concentrates was very similar, clearly the differences between GRG and tungsten recoveries are not specific to the SB, but are a general problem of semi-batch centrifuge units.

6.7 Summary

The SB4 recovered more than 90% of tungsten for all tests with a fine magnetite gangue, and more than 95% of tungsten for all tests with a coarse or fine silica gangue.

(1) The 4-in SB achieved extremely high recoveries over the full size range of tungsten (almost 100%) for low density feeds. Results for the magnetite feed confirmed that flash concentrates can be effectively treated with a SB at a suitable feed rate and fluidizing water flowrate.

(2) Within the range tested, fluidization water flowrate had a limited effect on the 4-in SB performance for both the coarse and fine silica feeds, and a significant one for the magnetite feed. Test work confirmed that different feeds had a different optimum fluidization water flowrate.

(3) Feed rate spanned a wide range for the light gangues without a significant effect on recovery. With a high density gangue, increasing feed rate had a more deleterious effect on recovery, which appeared to be uniform for all size fractions. Recovery nevertheless remained high, 92%, at a feed rate of 5 kg/min.

(4) The efficiency of the 4-in SB is affected primarily by gangue density. A high density feed was sensitive to changes of both feed rate and fluidizing water flowrate and had a lower overall performance than lower density feeds.

(5) Feed size had little effect on the overall 4-in SB metallurgical performance with a low density gangue, but significantly changed the concentrate size distribution. With a fine feed, recovery of the very fine (-25 μm) particles went up significantly.

(6) Tungsten recoveries achieved with the 4-in SB were much higher than GRG measured in full scale units in Chapters 3 and 5, at equivalent or lower loadings. The differences could have been caused by a number of factors, many of which could be investigated at plant and/or laboratory scale.

CHAPTER 7

CONCLUSIONS

7.1 Conclusions

The practical and fundamental experiments accomplished the desired objectives and obtained results which are summarized below:

1. Both laboratory and plant performances confirmed that SuperBowl is an effective gold centrifugal concentrator within the range of variables tested.

- The concentrate bed formation and size-by-size recovery of the SB4 suggested that SuperBowl combines characteristics of both Knelson and Falcon Concentrators (as of its bowl structure), which makes it operate mainly under non-fluidized conditions and suits the recovery of fine particles. Gangue density was the most significant factor affecting its performance, as observed for Knelson Concentrators at plant and laboratory scales.

- Comparison of a 21-in SB with a 20-in KC at the New Britannia Mine demonstrated that both machines could achieve similar performances (i.e. with no statistically significant difference) at plant scale, at relatively low feed rate. This comparison, however, was incomplete, and it did not address some key issues which are outlined in Chapter 5.

- The 21-in SB at the Mineral Hill Mine recovered the full size range of GRG, especially below 25 μm. As the unit feed rate was limited by the screen ahead of it to 0.3-2 t/h, the overall plant recovery was relatively low, only 15-20%. By installing a large capacity screen, the gravity recovery would have increased significantly.

2. Grinding circuit surveys determined that the best stream for gravity recovery was the primary cyclone underflow for both the Mineral Hill and New Britannia Mines. Compared to other streams in the grinding circuits, the primary cyclone underflows had the highest gold grade, and the highest proportion of gravity recoverable gold. Feeding part or all of this stream to a gravity centrifuge would achieve both the highest gold primary concentrate grade and recovery.

3. Test work at Casa Berardi further confirmed the fineness of its GRG, with virtually no gravity recoverable gold coarser than 212 μm, which suggests that a finer feed would increase the Knelson performance. Because of the unsteady gold inventory in the grinding circuit, both stage and overall gold recoveries could not be measured accurately. It was concluded that 350 L/min is a better fluidization flow rate than 150 L/min, and that the recovery cycle was too long at 120 minutes.

4. The GRG content in the New Britannia ore, 75%, was above average among thirty eight ores tested, but relatively fine, more than 78% GRG below 105 μm. This suggests a finer screen should be used to remove coarse barren particles feeding to the Knelson or SuperBowl and achieve a better gravity circuit performance than that obtained in the plant trials.

7.2 Recommendations

Mineral Hill (to be implemented upon recommissioning of the mine and mill)

- A GRG characterization test should be performed on a representative sample of the ore to assess the full potential of gravity recovery.

- Gravity recovery should be pursued at Mineral Hills, but with a much larger bleed of the primary cyclone underflow.

- Scavenging of fine gold from the table tails should be tested.

Casa Berardi (to be implemented upon recommissioning of the mine and mill)

- The potential of gravity recovery at Casa Berardi has been known for a number of years. Unfortunately, this potential has been only partially tapped because of a failure to take into account the very fine nature of the gravity recoverable gold.

- An efficient screening circuit should be installed to prepare for the 30-in KC a feed which is optimal for gold recovery. This would probably require screening of a primary cyclone underflow bleed at 400 to 500 μm.

- Once an adequate feed has been secured, operating parameters (recovery cycle time and fluidization flow rate) should be optimized. It is likely that much lower flow rates and larger recovery cycles could be used with a finer feed.

New Britannia

As gravity test work was terminated because of the low gold price and a failure to detect an impact on overall gold recovery, it is difficult to formulate any specific recommendation at this point. It is likely that despite the significant amount of test work, the full potential of gravity recovery, both metallurgical and economic, has not been fully determined. Recoveries of more than 50% by gravity appear feasible, given the high GRG content of the ore. This can only be achieved if a much higher throughput is bled and fed to a centrifuge (in excess of 30 t/h). This could be tested on either a 20-in Knelson or 21-in SuperBowl, whenever gold's price will have recovered enough to justify the expenditure.

TVX

A corporate approach to gravity recovery may be as important as good site technological expertise and commitment to operational excellence. Although in principle gravity circuits are easily retrofitted in operating plants, the constraints of circuit layouts are such that retrofit exercises are seldom optimum and are at times very far in efficiency from what could have been achieved had gravity been considered at the green field stage. The relevance of gravity recovery can be established easily in a project, with relatively inexpensive testing (Woodcock and Laplante, 1993) early in the design stage. Generally little test work is needed to generate the information required for appropriate flowsheet design. It is then up to the design team to allow for the appropriate space in the plant layout to allow for both primary and secondary (gold room) recovery. The most successful operations are those where the gravity units are fed significant tonnage, to maximize gold production by gravity. This implies that the screening stage ahead of the centrifuge must also be properly designed. Thus the corporate technical team must exercise judicious decision-making at the design stage and have the vision to incorporate gravity recovery in the original plant flowsheet, or at least to make retrofit not only possible but also effective, should the potential for gravity recovery be such that the decision to use gravity recovery must be delayed.

7.3 Future work

This study was meant to be exploratory, and did not seek to elucidate in detail concentration mechanisms in SuperBowls. Nevertheless, results clearly indicated that the bottom part of the concentrate bed was similar to that of a Batch Falcon, whereas the top one was similar to a Knelson Concentrator. There was no clear transition between the two. Further work should determine the relative contribution of these two zones to overall recovery, and compare the nature of the concentrate bed and effect of operating variables with their effect on both the "Falcon" and "Knelson" concentrate zones. The following questions could then be answered:

1. Is the lower part of the concentrate bed as sensitive to concentrate bed erosion as the Falcon (Laplante and Nickoletopoulos, 1997)?
2. Does the lower part make a significant contribution to fine gold recovery, as suggested by the manufacturer?
3. Could bowl geometry be improved for example by adding a fluidized ring lower down the tapered section?

This study did not address the effect of rotating speed which can significantly affect the SB performance. Not only could rotating velocity be optimized, but it would be possible to investigate the effect of ramping velocity up or down throughout a recovery cycle to mitigate the effects of concentrate bed erosion.

As discussed in Chapter 6, factors likely to affect the success of an industrial unit should be investigated in as many industrial SB circuits as possible. The existing gold market has led to the closing (temporary or permanent) of a number of operations (e.g. Mineral Hill, Casa Berardi, Madsen gold) where SBs were or could have been operated, making this type of survey difficult to carry. Ultimately, successful industrial operations, rather than academic surveys, will determine the fate of a given industrial unit.

That the 21-in SB and 20-in KC yielded similar performances is not really a novel finding, as both units are semi-batch centrifuges with back-flow fluidization water. The comparison is far from complete, as it should have included the very important following components:

1. Because absolute gold recovery is achieved at maximum feed rate, both units should have been tested at increasing feed rates, up to the point where gold recovery no longer increases. For the 20-in KC, indications are that this may be as much as 70 t/h (used at the Kundana Mine, Hillman (1997)).
2. The WSC cone tested on the 21-in KC yielded higher gold recoveries than the fourth generation cone used for the extended testing period. Further testing of the KC should include the WSC, whose use is rapidly spreading (Laplante, 1998).
3. Factors such as mechanical reliability, ease of operation, wear-part life, water consumption, sensitivity to operations variables and disturbances, all of which can reduce operating availability and increase operating costs, are critical in equipment selection. In the test work at New Britannia, there was some evidence that the 21-in SB was more sensitive to operating conditions than the 20-in KC, from test observations (Jean, 1998) and the lower feed density and rate used for the week-long trial. These factors are best studied in industrial environments, and it is unlikely that the McGill University research team is best qualified to carry out this type of work.

The test work at Casa Berardi indicated that short-term tests may be useful to gain an understanding of how full scale units work, but may be too vulnerable to feed fluctuations (flow rate, gold grade, GRG content and size distribution) to identify optimum operating conditions. It is suggested that an evolutionary operation (EVOP) method (Mular, 1971) would be statistically and operationally more robust, and its use should be tested and documented at a chosen site.

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APPENDIX A

TEST RESULTS AT THE MINERAL HILL MINE

Table A-1 to A-10: Plant SuperBowl Test

Table A-11 to A-12: Shaking Table Performance

Table A-13 to A-17: Grinding Circuit Survey

Table A-1 Mineral Hill Mine -850 μm T-I-F, 641 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	37.24	18.16	62	26.42	869	8.13	7.4	73.58	906	8.32	9.6	0.74
420	42.39	20.67	121	24.77	1492	13.96	10.4	75.23	1534	14.09	13.4	1.75
300	38.69	18.87	171	25.80	1608	15.05	11.8	74.20	1647	15.12	15.5	2.16
210	22.08	10.77	472	26.19	1204	11.27	24.4	73.81	1226	11.26	32.5	3.37
150	20.44	9.97	4990	68.21	1251	11.71	38.0	31.79	1271	11.68	117.6	12.65
105	15.77	7.69	10153	79.84	1093	10.23	37.0	20.16	1109	10.18	180.9	16.97
75	13.36	6.52	16466	81.89	1018	9.53	47.8	18.11	1031	9.47	260.5	22.74
53	7.33	3.57	16613	79.43	575	5.39	54.8	20.57	583	5.35	263.1	12.97
37	4.82	2.35	29181	81.71	596	5.58	52.8	18.29	601	5.52	286.4	14.57
25	1.75	0.85	33842	65.55	383	3.59	81.2	34.45	385	3.54	234.7	7.65
-25	1.18	0.58	26236	59.10	595	5.57	36.0	40.90	596	5.48	87.8	4.43
Total	205.05	100.00	4190	72.71	10685	100.00	30.2	27.29	10890	100.00	108.5	100.00

Table A-2 Mineral Hill Mine -850 μm T-I-T, 543 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	31.16	16.12	19	44.45	248	5.29	2.9	55.55	280	5.72	4.6	0.72
420	39.12	20.23	28	20.37	526	11.19	8.0	79.63	565	11.55	9.4	2.92
300	35.60	18.41	59	22.97	672	14.30	10.4	77.03	707	14.46	12.8	5.01
210	20.04	10.37	198	48.03	537	11.43	8.0	51.97	557	11.39	14.8	4.57
150	19.63	10.15	339	36.76	572	12.19	20.0	63.24	592	12.11	30.6	10.01
105	16.58	8.58	692	44.12	523	11.13	27.8	55.88	539	11.03	48.2	14.38
75	14.97	7.74	1349	51.55	524	11.16	36.2	48.45	539	11.03	72.6	21.66
53	7.97	4.12	2206	65.47	290	6.17	32.0	34.53	298	6.09	90.2	14.85
37	4.95	2.56	2499	51.13	296	6.29	40.0	48.87	301	6.15	80.5	13.38
25	2.11	1.09	2656	37.07	189	4.02	50.4	62.93	191	3.90	79.2	8.36
-25	1.21	0.63	1232	19.82	321	6.83	18.8	80.18	322	6.58	23.4	4.16
Total	193.34	100.00	430	45.93	4697	100.00	20.8	54.07	4890	100.00	37.0	100.00

2.9 (original assay is 28.6 g/t)

Table A-3 Mineral Hill Mine -850 μm T-II-F, 798 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	38.75	22.11	95	14.23	965	9.10	23.0	85.77	1004	9.31	25.8	1.51
420	38.15	21.77	485	34.74	1565	14.76	22.2	65.26	1604	14.88	33.2	3.11
300	35.71	20.38	1386	43.32	1627	15.34	39.8	56.68	1663	15.42	68.7	6.68
210	21.06	12.02	2960	57.09	1294	12.20	36.2	42.91	1315	12.20	83.0	6.39
150	16.40	9.36	9105	69.12	1334	12.58	50.0	30.88	1351	12.53	159.9	12.63
105	9.78	5.58	20929	72.18	1022	9.64	77.2	27.82	1032	9.57	274.9	16.58
75	7.26	4.14	38006	76.10	950	8.96	91.2	23.90	957	8.88	378.7	21.20
53	3.84	2.19	37909	68.09	632	5.96	108.0	31.91	636	5.90	336.4	12.50
37	2.66	1.52	38726	65.49	414	3.91	131.0	34.51	417	3.87	377.2	9.20
25	0.98	0.56	37182	30.13	371	3.49	228.0	69.87	372	3.45	325.5	7.07
-25	0.66	0.38	22440	27.83	431	4.06	89.2	72.17	431	4.00	123.4	3.11
Total	175.25	100.00	6070	62.21	10605	100.00	60.9	37.79	10780	100.00	158.6	100.00

23.0 (original assay is 230 g/t)

Table A-4 Mineral Hill Mine -850 μm T-II-T, 642 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	14.86	9.63	44	34.14	154	1.96	8.2	65.86	169	2.10	11.4	0.29
420	26.41	17.11	45	8.51	456	5.80	28.0	91.49	483	6.02	28.9	2.12
300	26.09	16.91	132	15.99	713	9.06	25.4	84.01	739	9.21	29.2	3.27
210	17.81	11.54	243	23.70	774	9.84	18.0	76.30	792	9.87	23.1	2.77
150	18.51	11.99	1265	50.69	1035	13.16	22.0	49.31	1054	13.14	43.8	7.02
105	14.24	9.23	1972	42.89	954	12.13	39.2	57.11	968	12.07	67.6	9.95
75	14.16	9.18	4337	51.81	1098	13.96	52.0	48.19	1113	13.87	106.5	18.00
53	9.96	6.45	6399	47.97	934	11.87	74.0	52.03	944	11.77	140.7	20.18
37	7.77	5.03	6978	43.29	800	10.17	88.8	56.71	807	10.07	155.1	19.02
25	2.74	1.78	8802	32.16	436	5.55	116.6	67.84	439	5.48	170.8	11.39
-25	1.77	1.15	5472	24.62	511	6.50	58.0	75.38	513	6.40	76.7	5.98
Total	154.32	100.00	1777	41.66	7866	100.00	48.8	58.34	8020	100.00	82.1	100.00

8.2 (original assay is 82 g/t)

Table A-5 Mineral Hill Mine -850 μm T-III-F, 590 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	23.02	13.28	84	43.55	212	3.21	11.8	56.45	235	3.47	18.9	0.66
420	34.23	19.75	65	31.11	547	8.27	9.0	68.89	582	8.57	12.3	1.06
300	36.77	21.21	271	44.74	892	13.48	13.8	55.26	929	13.68	24.0	3.30
210	22.06	12.73	616	27.91	820	12.39	42.8	72.09	842	12.40	57.8	7.21
150	18.36	10.59	1794	59.76	798	12.06	27.8	40.24	816	12.02	67.5	8.17
105	12.51	7.22	4009	59.51	659	9.96	51.8	40.49	671	9.89	125.5	12.49
75	10.82	6.24	8627	71.66	671	10.14	55.0	28.34	682	10.04	191.0	19.30
53	6.33	3.65	13352	69.39	489	7.39	76.2	30.61	496	7.30	245.8	18.05
37	4.84	2.79	15002	71.51	432	6.53	67.0	28.49	437	6.43	232.6	15.04
25	2.19	1.26	15635	55.01	347	5.25	80.6	44.99	350	5.15	178.0	9.22
-25	2.22	1.28	7306	43.61	749	11.32	28.0	56.39	751	11.06	49.5	5.51
Total	173.35	100.00	2375	61.00	6617	100.00	39.8	39.00	6790	100.00	99.4	100.00

Table A-6 Mineral Hill Mine -850 μm T-III-T, 589 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	8.93	6.29	8	43.91	38	1.98	2.4	56.09	47	2.28	3.5	0.16
420	17.80	12.53	24	34.54	116	6.03	7.0	65.46	133	6.48	9.3	1.22
300	25.70	18.09	52	27.69	213	11.09	16.4	72.31	238	11.58	20.2	4.76
210	18.71	13.17	57	23.09	213	11.10	16.8	76.91	232	11.24	20.1	4.59
150	19.03	13.40	106	30.95	230	11.97	19.6	69.05	249	12.07	26.2	6.42
105	16.13	11.36	374	48.62	204	10.66	31.2	51.38	220	10.70	56.3	12.23
75	15.75	11.09	489	46.69	230	12.00	38.2	53.31	246	11.94	67.1	16.26
53	8.93	6.29	919	47.01	179	9.35	51.6	52.99	188	9.14	92.8	17.21
37	6.27	4.41	1312	50.32	151	7.87	53.8	49.68	157	7.63	104.0	16.12
25	2.48	1.75	1576	30.97	124	6.49	70.0	69.03	127	6.16	99.4	12.44
-25	2.30	1.62	1018	26.86	220	11.46	29.0	73.14	222	10.78	39.2	8.59
Total	142.03	100.00	291	40.76	1918	100.00	31.3	59.24	2060	100.00	49.2	100.00

Table A-7 Mineral Hill Mine -850 μm T-IV-F, 645 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	30.71	17.84	144	51.78	895	7.61	4.6	48.22	926	7.75	9.2	0.66
420	35.09	20.38	242	42.41	1602	13.61	7.2	57.59	1637	13.71	12.2	1.54
300	32.87	19.09	392	41.42	1822	15.48	10.0	58.58	1855	15.54	16.8	2.39
210	19.93	11.58	1685	53.81	1441	12.25	20.0	46.19	1461	12.24	42.7	4.79
150	18.07	10.50	6424	72.61	1479	12.57	29.6	27.39	1497	12.54	106.8	12.27
105	13.26	7.70	14310	78.38	1229	10.44	42.6	21.62	1242	10.40	194.9	18.58
75	10.68	6.20	18801	76.82	1130	9.61	53.6	23.18	1141	9.56	229.1	20.06
53	5.44	3.16	33641	87.34	611	5.19	43.4	12.66	617	5.16	339.8	16.08
37	3.63	2.11	38411	81.12	614	5.22	52.8	18.88	618	5.18	278.1	13.19
25	1.44	0.84	37804	63.27	384	3.27	82.2	36.73	386	3.23	223.0	6.60
-25	1.02	0.59	30706	62.79	559	4.75	33.2	37.21	560	4.69	89.1	3.83
Total	172.14	100.00	5659	74.78	11768	100.00	27.9	25.22	11940	100.00	109.1	100.00

Table A-8 Mineral Hill Mine -850 μm T-IV-T, 577 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight* (g)	% %Weight	Grade* (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	16.90	9.86	37	59.65	70	1.97	6.0	40.35	87	2.33	12.0	0.67
420	26.76	15.61	11	15.81	212	5.92	7.4	84.19	239	6.36	7.8	1.18
300	31.81	18.56	25	19.00	361	10.08	9.4	81.00	393	10.47	10.7	2.66
210	20.61	12.02	60	28.70	375	10.47	8.2	71.30	395	10.54	10.9	2.74
150	20.56	11.99	332	50.62	450	12.57	14.8	49.38	470	12.54	28.7	8.57
105	17.44	10.17	386	37.44	408	11.39	27.6	62.56	425	11.33	42.3	11.43
75	16.98	9.91	1051	55.50	464	12.98	30.8	44.50	481	12.84	66.8	20.44
53	9.79	5.71	1781	58.63	380	10.61	32.4	41.37	389	10.39	76.4	18.90
37	6.44	3.76	2352	55.52	323	9.02	37.6	44.48	329	8.78	82.9	17.34
25	2.40	1.40	2914	41.80	215	6.02	45.2	58.20	218	5.81	76.8	10.63
-25	1.73	1.01	1681	33.96	321	8.98	17.6	66.04	323	8.62	26.5	5.44
Total	171.42	100.00	448	48.84	3579	100.00	22.5	51.16	3750	100.01	42.0	100.00

* Corrected

Table A-9 Mineral Hill Mine -850 μm T-V-F, 561 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	39.55	20.75	127	39.62	539	7.31	14.2	60.38	579	7.64	21.9	1.82
420	40.57	21.28	71	24.12	871	11.81	10.4	75.88	912	12.05	13.1	1.71
300	33.49	17.57	209	34.48	978	13.25	13.6	65.52	1011	13.36	20.1	2.91
210	19.82	10.40	489	26.80	843	11.42	31.4	73.20	863	11.40	41.9	5.18
150	18.38	9.64	2012	65.77	917	12.42	21.0	34.23	935	12.35	60.1	8.05
105	13.59	7.13	3775	66.51	755	10.24	34.2	33.49	769	10.16	100.3	11.05
75	11.51	6.04	7495	65.76	736	9.98	61.0	34.24	748	9.88	175.4	18.79
53	6.28	3.29	14068	73.05	509	6.90	64.0	26.95	515	6.81	234.6	17.32
37	4.52	2.37	19844	77.99	362	4.90	70.0	22.01	366	4.84	314.1	16.47
25	1.70	0.89	24947	55.89	337	4.56	99.4	44.11	338	4.47	224.2	10.87
-25	1.22	0.64	16709	50.04	533	7.22	38.2	49.96	534	7.05	76.3	5.83
Total	190.63	100.00	2308	63.02	7379	100.00	35.0	36.98	7570	100.00	92.2	100.00

Table A-10 Mineral Hill Mine -850 μm T-V-T, 743 g/min, 3.5 psi

Size (μm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	26.64	13.05	80	44.56	233	2.79	11.4	55.44	259	3.03	18.5	1.02
420	39.45	19.32	30	17.93	616	7.38	8.8	82.07	655	7.66	10.1	1.41
300	38.24	18.73	94	32.67	926	11.09	8.0	67.33	964	11.28	11.4	2.35
210	22.41	10.97	308	36.99	864	10.36	13.6	63.01	887	10.37	21.0	3.99
150	21.52	10.54	750	44.40	1011	12.11	20.0	55.60	1032	12.07	35.2	7.78
105	18.24	8.93	1589	52.76	976	11.69	26.6	47.24	994	11.62	55.3	11.76
75	16.89	8.27	3910	67.81	1024	12.27	30.6	32.19	1041	12.18	93.5	20.84
53	9.02	4.42	6295	60.71	756	9.06	48.6	39.29	765	8.95	122.2	20.01
37	6.36	3.11	7901	60.42	599	7.17	55.0	39.58	605	7.07	137.5	17.80
25	2.47	1.21	7698	45.65	407	4.88	55.6	54.35	410	4.79	101.7	8.91
-25	2.96	1.45	2071	31.88	936	11.21	14.0	68.12	939	10.98	20.5	4.11
Total	204.20	100.00	1259	55.03	8346	100.00	25.2	44.97	8550	100.00	54.7	100.00
	80.0	(original assay is 8 g/t)										

Table A-11 Mineral Hill Mine -850 μm STF, 639 g/min, 4.0 psi

Size (μm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	28.08	12.06	14104	25.37	1482	8.70	786	74.63	1510	8.75	1034	3.12
420	45.76	19.66	12333	28.09	2147	12.61	673	71.91	2193	12.70	916	4.02
300	45.54	19.57	18427	31.30	2749	16.15	670	68.70	2795	16.19	959	5.36
210	29.66	12.74	28645	34.72	2035	11.95	785	65.28	2064	11.96	1185	4.89
150	17.46	7.50	197460	58.56	1946	11.43	1254	41.44	1963	11.37	2999	11.78
105	23.55	10.12	264874	72.11	1377	8.09	1752	27.89	1401	8.11	6176	17.31
75	13.96	6.00	329202	58.02	1428	8.39	2328	41.98	1442	8.36	5492	15.84
53	9.98	4.29	362933	56.10	1053	6.18	2692	43.90	1063	6.16	6074	12.92
37	12.19	5.24	284497	57.01	1042	6.12	2509	42.99	1055	6.11	5768	12.17
25	3.36	1.44	225078	18.97	640	3.76	5047	81.03	643	3.73	6196	7.97
-25	3.22	1.38	210221	29.39	1127	6.62	1443	70.61	1130	6.55	2038	4.61
Total	232.76	100.00	109355	50.92	17027	100.00	1441	49.08	17260	100.00	2896	100.00

Table A-12 Mineral Hill Mine -850 μm STT, 743 g/min, 4.0 psi

Size (μm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	55.73	21.10	425	14.55	1779	11.59	78	85.45	1835	11.76	89	1.43
420	70.70	26.77	655	9.16	2495	16.26	184	90.84	2566	16.44	197	4.44
300	50.40	19.08	1529	7.78	3046	19.85	300	92.22	3096	19.84	320	8.71
210	33.56	12.71	6201	29.14	2181	14.21	232	70.86	2215	14.19	322	6.28
150	21.52	8.15	32200	49.01	1872	12.20	385	50.99	1894	12.13	747	12.42
105	13.65	5.17	80147	67.92	1038	6.76	498	32.08	1051	6.74	1532	14.16
75	6.93	2.62	165059	66.69	742	4.84	770	33.31	749	4.80	2290	15.07
53	4.11	1.56	193694	63.15	388	2.53	1198	36.85	392	2.51	3217	11.08
37	4.72	1.79	209431	69.47	314	2.05	1382	30.53	319	2.04	4459	12.51
25	1.17	0.44	139966	34.62	194	1.26	1596	65.38	195	1.25	2427	4.16
-25	1.62	0.61	140369	20.50	1297	8.45	680	79.50	1298	8.32	854	9.75
Total	264.11	100.00	20680	48.00	15346	100.00	386	52.00	15610	100.00	729	100.00

Table A-13 Mineral Hill Mine -850 μm PCOF, 367 g/min, 3.0 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	8.04	5.62	71	82.03	63	1.09	2.0	17.97	71	1.20	9.9	0.41
420	12.89	9.01	107	54.55	185	3.24	6.2	45.45	198	3.38	12.8	1.49
300	16.77	11.72	25	16.79	346	6.05	6.0	83.21	363	6.19	6.9	1.47
210	12.67	8.86	77	32.59	388	6.78	5.2	67.41	401	6.83	7.5	1.76
150	15.02	10.50	331	60.09	516	9.01	6.4	39.91	531	9.04	15.6	4.86
105	16.02	11.20	471	62.67	576	10.06	7.8	37.33	592	10.09	20.3	7.08
75	19.86	13.88	1134	75.29	725	12.65	10.2	24.71	745	12.68	40.2	17.59
53	15.00	10.49	1176	68.73	617	10.78	13.0	31.27	632	10.77	40.6	15.09
37	14.40	10.07	1663	69.63	562	9.81	18.6	30.37	576	9.81	59.7	20.22
25	6.94	4.85	2459	66.91	469	8.19	18.0	33.09	476	8.11	53.6	15.00
-25	5.43	3.80	2730	57.96	1280	22.35	8.4	42.04	1285	21.90	19.9	15.04
Total	143.04	100.00	782	65.77	5727	100.00	10.2	34.23	5870	100.00	29.0	100.00

Table A-14 Mineral Hill Mine -850 μm PCUF, 623 g/min, 3.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	34.61	17.56	36	14.18	438	4.96	17.2	85.82	473	5.23	18.6	0.87
420	36.27	18.40	133	29.37	699	7.90	16.6	70.63	735	8.13	22.3	1.63
300	33.40	16.95	230	36.55	926	10.47	14.4	63.45	960	10.61	21.9	2.08
210	22.18	11.25	480	41.26	1024	11.58	14.8	58.74	1046	11.57	24.7	2.55
150	23.24	11.79	1685	62.30	1463	16.54	16.2	37.70	1486	16.44	42.3	6.22
105	17.96	9.11	3013	52.05	1305	14.76	38.2	47.95	1323	14.64	78.6	10.29
75	14.98	7.60	10477	74.41	1174	13.27	46.0	25.59	1189	13.15	177.5	20.87
53	7.40	3.75	18616	74.50	708	8.01	66.6	25.50	715	7.91	258.5	18.30
37	4.67	2.37	31569	85.85	353	4.00	68.8	14.15	358	3.96	479.8	17.00
25	1.57	0.80	45833	53.25	439	4.96	144.0	46.75	440	4.87	306.9	13.37
-25	0.82	0.42	44988	53.53	314	3.55	102.0	46.47	315	3.48	218.9	6.82
Total	197.10	100.00	3392	66.17	8843	100.00	38.7	33.83	9040	100.00	111.8	100.00

Table A-15 Mineral Hill Mine -850 μm SCUF, 482 g/min, 3.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	8.85	5.43	87	75.26	53	0.39	4.8	24.74	62	0.45	16.6	0.21
420	17.86	10.95	35	12.26	246	1.81	18.2	87.74	264	1.92	19.3	1.07
300	20.58	12.62	54	18.78	586	4.32	8.2	81.22	607	4.42	9.8	1.24
210	15.71	9.63	63	14.71	1025	7.55	5.6	85.29	1040	7.57	6.5	1.41
150	19.09	11.71	276	33.41	2100	15.47	5.0	66.59	2119	15.43	7.4	3.30
105	20.65	12.66	677	37.51	2841	20.92	8.2	62.49	2861	20.83	13.0	7.80
75	24.07	14.76	1365	44.89	2689	19.81	15.0	55.11	2713	19.75	27.0	15.31
53	16.58	10.17	2412	52.08	1448	10.67	25.4	47.92	1465	10.66	52.4	16.06
37	12.91	7.92	4457	61.00	1088	8.02	33.8	39.00	1101	8.02	85.6	19.73
25	4.78	2.93	9024	52.78	654	4.82	59.0	47.22	659	4.80	124.0	17.09
-25	2.00	1.23	22309	55.56	846	6.23	42.2	44.44	848	6.17	94.7	16.80
Total	163.08	100.00	1477	50.38	13577	100.00	17.5	49.62	13740	100.00	34.8	100.00

Table A-16 Mineral Hill Mine -850 μm BMDIS, 352 g/min, 3.0 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	3.83	2.90	42	40.55	14	0.21	17.4	59.45	17	0.26	22.8	0.17
420	6.64	5.03	75	34.83	46	0.70	20.4	65.17	52	0.78	27.3	0.61
300	9.31	7.05	197	46.01	111	1.69	19.4	53.99	120	1.80	33.2	1.69
210	8.30	6.28	273	46.90	181	2.76	14.2	53.10	189	2.82	25.6	2.05
150	12.87	9.74	813	74.57	405	6.18	8.8	25.43	418	6.25	33.5	5.96
105	16.12	12.20	752	65.48	726	11.07	8.8	34.52	742	11.09	24.9	7.86
75	22.17	16.78	948	54.36	1063	16.21	16.6	45.64	1085	16.22	35.6	16.42
53	19.00	14.38	1201	63.26	850	12.96	15.6	36.74	869	12.98	41.5	15.32
37	18.00	13.62	1466	62.73	784	11.96	20.0	37.27	802	11.99	52.5	17.87
25	9.85	7.46	2188	56.78	683	10.42	24.0	43.22	693	10.36	54.7	16.12
-25	6.03	4.56	3123	50.24	1696	25.86	11.0	49.76	1702	25.44	22.0	15.92
Total	132.12	100.00	1044	58.60	6558	100.00	14.9	41.40	6690	100.00	35.2	100.00

Table A-17 Mineral Hill Mine -850 μm SAG DIS, 485 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	23.23	15.54	9	14.73	378	4.68	3.2	85.27	401	4.87	3.5	0.70
420	26.51	17.73	34	28.23	573	7.08	4.0	71.77	599	7.27	5.3	1.59
300	25.09	16.78	133	42.13	674	8.33	6.8	57.87	699	8.49	11.3	3.93
210	17.29	11.57	375	60.22	630	7.79	6.8	39.78	647	7.85	16.6	5.35
150	17.80	11.91	843	61.36	762	9.42	12.4	38.64	780	9.47	31.4	12.15
105	13.37	8.94	1079	68.35	726	8.98	9.2	31.65	739	8.97	28.5	10.48
75	10.66	7.13	2519	74.45	808	9.99	11.4	25.55	819	9.94	44.0	17.91
53	5.92	3.96	4762	79.95	580	7.16	12.2	20.05	585	7.11	60.2	17.51
37	4.54	3.04	4607	74.84	663	8.20	10.6	25.16	668	8.11	41.8	13.88
25	2.30	1.54	4396	55.78	573	7.08	14.0	44.22	575	6.98	31.5	9.00
-25	2.79	1.87	2192	40.56	1723	21.30	5.2	59.44	1726	20.95	8.7	7.49
Total	149.50	100.00	887	65.83	8091	100.00	8.5	34.17	8240	100.00	24.4	100.00

APPENDIX B

TEST RESULTS AT CASA BERARDI

Table B-1 to B-3: Test at 150 L/min

Table B-4 to B-10: Test at 350 L/min

Table B-1 Casa Berardi -300 μm KC Tails (150 L/min, 0-40min), 316 g/min, 2.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	8.09	6.60	380	36.45	323	4.97	16.6	63.55	331	5.00	25.5	0.57
150	19.04	15.53	1592	45.42	1059	16.30	34.4	54.58	1078	16.28	61.9	4.47
105	24.62	20.08	2393	44.86	1244	19.15	58.2	55.14	1269	19.17	103.5	8.80
75	30.20	24.63	4519	54.71	1244	19.15	90.8	45.29	1274	19.25	195.7	16.71
53	19.30	15.74	9111	62.08	815	12.54	131.8	37.92	834	12.60	339.5	18.97
37	13.63	11.12	16165	69.58	611	9.40	157.8	30.42	624	9.43	507.3	21.21
25	5.11	4.17	28760	70.87	329	5.06	183.8	29.13	334	5.04	621.3	13.89
-25	2.61	2.13	50856	57.81	873	13.43	111.0	42.19	875	13.22	262.3	15.38
Total	122.60	100.00	7379	60.60	6497	99.99	90.5	39.40	6620	99.99	225.5	100.00

Table B-2 Casa Berardi -300 μm KC Feed (150 L/min, 0-40min), 325 g/min, 2.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	9.06	7.18	469	80.63	243	4.79	4.2	19.37	252	4.85	20.9	0.46
150	20.06	15.89	1206	55.22	846	16.67	23.2	44.78	866	16.65	50.6	3.86
105	23.83	18.88	2072	47.84	979	19.30	55.0	52.16	1003	19.28	102.9	9.08
75	30.53	24.18	3523	53.80	1047	20.64	88.2	46.20	1078	20.72	185.5	17.59
53	20.37	16.14	6159	58.90	677	13.33	129.4	41.10	697	13.40	305.6	18.75
37	14.76	11.69	12046	69.23	504	9.94	156.8	30.77	519	9.98	495.0	22.60
25	5.32	4.21	21609	70.58	272	5.35	176.4	29.42	277	5.33	588.1	14.33
-25	2.31	1.83	43945	67.08	506	9.98	98.4	32.92	509	9.78	297.5	13.32
Total	126.24	100.00	5585	62.05	5074	100.00	85.0	37.95	5200	100.00	218.5	100.00

Table B-3 Casa Berardi -300 μm KC Conc. (150 L/min, 0-40min), 343 g/min, 2.5 psi

Size	CONCENTRATE				TAILS				FEED			
(μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	12.21	9.16	8070	78.60	55	6.19	483	21.40	68	6.57	1852	1.28
150	35.45	26.60	24683	88.71	209	23.27	534	11.29	244	23.70	4041	10.10
105	30.63	22.98	33938	89.67	190	21.15	631	10.33	220	21.39	5262	11.87
75	27.33	20.50	63916	92.69	185	20.63	744	7.31	212	20.62	8875	19.29
53	13.73	10.30	116976	92.18	120	13.36	1138	7.82	134	12.97	13046	17.84
37	8.89	6.67	193771	91.15	79	8.83	2111	8.85	88	8.55	21452	19.35
25	3.33	2.50	273128	85.36	34	3.81	4573	14.64	37	3.64	28453	10.91
-25	1.72	1.29	400827	75.46	25	2.76	9059	24.54	26	2.57	34517	9.35
Total	133.29	100.00	65178	88.95	897	100.00	1204	11.05	1030	100.00	9482	100.00

Table B-4 Casa Berardi -300 μm KC Feed (350 L/min, 0-40min), 339 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	8.97	7.00	438	59.24	347	6.98	7.8	40.76	355	6.98	18.7	0.65
150	25.77	20.11	922	48.28	1061	21.38	24.0	51.72	1086	21.34	45.3	4.80
105	26.51	20.69	1961	54.45	1035	20.87	42.0	45.55	1062	20.86	89.9	9.32
75	29.75	23.22	3517	55.95	938	18.91	87.8	44.05	968	19.02	193.2	18.25
53	18.21	14.21	6557	57.15	629	12.67	142.4	42.85	647	12.71	323.0	20.39
37	12.79	9.98	11087	63.17	452	9.10	183.0	36.83	464	9.13	483.3	21.90
25	4.32	3.37	21614	63.03	228	4.60	240.0	36.97	232	4.57	637.2	14.45
-25	1.81	1.41	42085	72.55	272	5.49	105.8	27.45	274	5.39	382.9	10.25
Total	128.13	100.00	4800	60.02	4962	99.99	82.6	39.98	5090	99.99	201.3	100.00

Table B-5 Casa Berardi -300 μm KC Tails (350 L/min, 0-40min), 317 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	7.82	5.95	520	64.48	361	7.56	6.2	35.52	369	7.52	17.1	0.66
150	18.39	14.00	1018	48.03	1044	21.85	19.4	51.97	1063	21.64	36.7	4.09
105	24.90	18.95	1807	47.29	999	20.91	50.2	52.71	1024	20.85	92.9	9.98
75	34.52	26.27	3157	54.30	887	18.56	103.4	45.70	922	18.77	217.8	21.05
53	23.14	17.61	4972	58.27	561	11.75	146.8	41.73	584	11.90	337.9	20.71
37	15.53	11.82	8595	64.49	435	9.11	168.8	35.51	451	9.18	459.0	21.71
25	5.17	3.94	16742	69.25	214	4.47	180.0	30.75	219	4.45	571.5	13.11
-25	1.91	1.45	32075	73.90	277	5.79	78.2	26.10	279	5.67	297.6	8.69
Total	131.38	100.00	4362	60.11	4779	100.00	79.6	39.89	4910	100.00	194.2	100.00

Table B-6 Casa Berardi -300 μm KC Feed (350 L/min, 40-80min), 360 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	10.30	7.74	736	72.76	364	7.40	7.8	27.24	374	7.41	27.8	1.01
150	22.72	17.08	839	46.04	1006	20.46	22.2	53.96	1029	20.37	40.2	4.02
105	27.47	20.65	1374	42.36	1027	20.89	50.0	57.64	1055	20.89	84.5	8.66
75	31.79	23.90	2708	50.30	924	18.80	92.0	49.70	956	18.93	179.0	16.62
53	20.36	15.31	5142	52.14	595	12.10	161.6	47.86	615	12.18	326.5	19.50
37	13.83	10.40	12402	66.41	456	9.28	190.2	33.59	470	9.30	549.6	25.09
25	4.53	3.41	20878	64.72	221	4.49	233.6	35.28	225	4.46	648.8	14.19
-25	2.01	1.51	39701	71.11	324	6.58	100.2	28.89	326	6.45	344.7	10.90
Total	133.01	100.00	4519	58.39	4917	100.00	87.1	41.61	5050	100.00	203.9	100.00

Table B-7 Casa Berardi -300 μm KC Tails (350 L/min, 40-80min), 320 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	9.72	7.30	270	66.59	263	7.09	5.0	33.41	273	7.09	14.4	0.61
150	21.28	15.98	775	63.22	761	20.49	12.6	36.78	783	20.33	33.3	4.01
105	25.85	19.42	965	50.55	763	20.52	32.0	49.45	789	20.48	62.6	7.59
75	32.29	24.25	2242	63.94	690	18.56	59.2	36.06	722	18.75	156.8	17.40
53	21.69	16.29	3649	61.79	454	12.21	107.8	38.21	476	12.35	269.3	19.69
37	15.41	11.57	7262	69.45	355	9.54	138.8	30.55	370	9.61	435.5	24.77
25	5.08	3.82	14072	70.11	175	4.71	174.0	29.89	180	4.68	565.8	15.67
-25	1.82	1.37	27558	75.05	256	6.88	65.2	24.95	258	6.69	259.5	10.27
Total	133.14	100.00	3223	65.96	3717	100.00	59.6	34.04	3850	100.00	169.0	100.00

Table B-8 Casa Berardi -300 μm KC Feed (350 L/min, 80-120min), 291 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	18.95	15.55	309	80.73	349	7.24	4.0	19.27	368	7.44	19.7	0.87
150	25.45	20.88	685	53.79	998	20.68	15.0	46.21	1024	20.68	31.7	3.90
105	22.90	18.79	1213	45.45	981	20.31	34.0	54.55	1003	20.27	60.9	7.36
75	23.48	19.26	2991	53.69	902	18.67	67.2	46.31	925	18.69	141.4	15.75
53	14.25	11.69	7310	59.53	608	12.60	116.4	40.47	623	12.58	281.0	21.07
37	10.47	8.59	13222	67.57	470	9.73	141.4	32.43	480	9.71	426.5	24.68
25	4.09	3.36	21176	67.59	220	4.57	188.4	32.41	225	4.54	570.6	15.43
-25	2.31	1.89	29080	74.06	299	6.20	78.6	25.94	302	6.09	300.7	10.92
Total	121.90	100.00	4247	62.35	4828	100.00	64.8	37.65	4950	100.00	167.7	100.00

Table B-9 Casa Berardi -300 μm KC Tails (350 L/min, 80-120min), 350 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (μm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	10.04	8.09	270	61.74	336	7.29	5.0	38.26	346	7.31	12.7	0.69
150	20.35	16.39	775	56.52	963	20.91	12.6	43.48	983	20.79	28.4	4.37
105	26.27	21.16	965	45.52	948	20.58	32.0	54.48	974	20.60	57.2	8.71
75	30.88	24.88	2242	57.39	868	18.85	59.2	42.61	899	19.01	134.2	18.87
53	19.01	15.31	3649	52.73	577	12.53	107.8	47.27	596	12.60	220.8	20.58
37	12.20	9.83	7262	59.39	436	9.48	138.8	40.61	449	9.48	332.5	23.33
25	3.81	3.07	14072	60.47	201	4.37	174.0	39.53	205	4.34	432.0	13.87
-25	1.57	1.26	27558	70.63	276	5.99	65.2	29.37	277	5.87	220.8	9.58
Total	124.13	100.00	2964	57.55	4606	100.00	58.9	42.45	4730	100.00	135.1	100.00

Table B-10 Casa Berardi -300 µm KC Conc. (350 L/min, 0-120min), 302 g/min, 2.5 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
210	13.08	6.53	77654	28.78	387	7.60	6499	71.22	400	7.56	8826	4.16
150	24.42	12.19	139312	36.26	703	13.82	8503	63.74	728	13.76	12892	11.05
105	24.14	12.05	125553	30.33	659	12.95	10564	69.67	683	12.91	14628	11.76
75	33.84	16.89	164466	39.42	827	16.25	10343	60.58	861	16.27	16401	16.62
53	32.35	16.15	188749	47.45	866	17.01	7809	52.55	898	16.98	14325	15.15
37	36.06	18.00	209304	56.92	904	17.76	6320	43.08	940	17.77	14107	15.61
25	20.24	10.10	294886	58.95	480	9.43	8661	41.05	500	9.45	20247	11.92
-25	16.20	8.09	493009	68.52	264	5.18	13920	31.48	280	5.29	41654	13.72
Total	200.33	100.00	202780	47.83	5090	100.00	8706	52.17	5290	100.00	16055	100.00

APPENDIX C

TEST RESULTS AT THE NEW BRITANNIA MINE

Table C-1 to C-4: GRG Test

Table C-5 to C-10: Grinding Circuit Survey

Table C-11 to C-14: Plant SB and KC Performance

Table C-1 New Britannia Mine Rod Mill Feed, stage I, 100% -850 µm, 1.11kg/min, 4.0 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	6.19	4.92	35	2.63	2569	5.64	3.1	97.37	2575	5.64	3.2	3.59
420	11.70	9.29	40	2.94	4479	9.84	3.5	97.06	4491	9.83	3.6	7.03
300	16.23	12.89	227	21.05	4029	8.85	3.4	78.95	4045	8.86	4.3	7.62
210	14.04	11.15	248	23.93	2975	6.53	3.7	76.07	2989	6.54	4.9	6.34
150	18.01	14.30	329	30.32	3267	7.17	4.2	69.68	3285	7.19	5.9	8.51
105	17.45	13.86	469	37.18	3444	7.56	4.0	62.82	3461	7.58	6.4	9.58
75	18.01	14.30	777	44.22	4518	9.92	3.9	55.78	4536	9.93	7.0	13.80
53	10.77	8.55	1056	42.94	4499	9.88	3.4	57.06	4510	9.87	5.9	11.55
37	7.93	6.30	1585	49.24	4751	10.43	2.7	50.76	4759	10.42	5.4	11.12
25	3.41	2.71	2621	49.35	3714	8.16	2.5	50.65	3718	8.14	4.9	7.89
-25	2.18	1.73	4470	32.73	7301	16.03	2.7	67.27	7303	15.99	4.1	12.97
Total	125.92	100.00	624	34.24	45544	100.00	3.3	65.76	45670	100.00	5.0	100.00

Table C-2 New Britannia Mine Rod Mill Feed, stage II, 67% -200 mesh, 560g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
212	5.84	3.86	52	13.49	689	2.92	2.8	86.51	695	2.93	3.2	3.41
150	15.61	10.32	153	41.49	1703	7.22	2.0	58.51	1719	7.24	3.3	8.68
105	25.20	16.66	117	37.29	2253	9.55	2.2	62.71	2278	9.59	3.5	11.92
75	38.22	25.26	137	44.24	3148	13.34	2.1	55.76	3187	13.42	3.7	17.86
53	28.34	18.73	176	44.00	3326	14.09	1.9	56.00	3354	14.12	3.4	17.09
37	21.24	14.04	224	44.97	3769	15.97	1.5	55.03	3791	15.96	2.8	15.98
25	10.54	6.97	170	24.57	4120	17.46	1.3	75.43	4130	17.39	1.8	11.04
-25	6.31	4.17	235	16.01	4590	19.45	1.7	83.99	4596	19.35	2.0	14.03
Total	151.30	100.00	158	36.07	23599	100.00	1.8	63.93	23750	100.00	2.8	100.00

Table C-3 New Britannia Mine Rod Mill Feed, stage III, 82% -200 mesh, 328g/min, 2.5 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
150	3.24	2.19	29.0	31.52	163	0.74	1.3	68.48	166	0.75	1.8	0.71
105	14.25	9.65	46.3	29.76	1082	4.93	1.4	70.24	1096	4.96	2.0	5.30
75	36.49	24.71	66.4	42.13	2660	12.12	1.3	57.87	2696	12.20	2.1	13.75
53	33.41	22.62	82.1	41.47	3137	14.29	1.2	58.53	3171	14.35	2.1	15.81
37	30.68	20.77	93.6	45.38	3734	17.01	0.9	54.62	3765	17.04	1.7	15.12
25	17.69	11.98	123.7	38.59	3906	17.79	0.9	61.41	3924	17.75	1.4	13.55
-25	11.93	8.08	397.5	31.69	7271	33.12	1.4	68.31	7283	32.95	2.1	35.76
Total	147.69	100.00	106.5	37.57	21952	100.00	1.2	62.43	22100	100.00	1.9	100.00

Table C-4 New Britannia Mine Rod Mill Feed Overall Results

Size (µm)	Frist Stage: 100% -850 µm			Second Stage: 67% -75 µm			Third Stage: 82% -75 µm				Total	Total
	Stage Rec.	Dist'n	Rec. g/t	Stage Rec.	Disn't	Rec. g/t	Stage Rec.	Disn't	Rec. g/t	Losses g/t	Rec. g/t	Rec. %
600	2.63	3.59	0.0								0.0	0.1
420	2.94	7.03	0.0								0.0	0.2
300	21.05	7.62	0.1	0.00	0.00	0.0					0.1	1.8
210	23.93	6.34	0.1	13.49	3.41	0.0					0.1	1.9
150	30.32	8.51	0.1	41.49	8.68	0.1	31.52	0.71	0.0	0.0	0.2	5.1
105	37.18	9.58	0.2	37.29	11.92	0.1	29.76	5.30	0.0	0.1	0.3	7.2
75	44.22	13.80	0.3	44.24	17.86	0.2	42.13	13.75	0.1	0.1	0.6	13.8
53	42.94	11.55	0.2	44.00	17.09	0.2	41.47	15.81	0.1	0.2	0.6	12.6
37	49.24	11.12	0.3	44.97	15.98	0.2	45.38	15.12	0.1	0.2	0.6	13.1
25	49.35	7.89	0.2	24.57	11.04	0.1	38.59	13.55	0.1	0.2	0.4	8.0
-25	32.73	12.97	0.2	16.01	14.03	0.1	31.69	35.76	0.2	0.5	0.5	10.6
Total	37.5	100	1.7	34.9	100	1.0	37.6	100	0.7	1.2	3.4	74.6
O/A	37.5			21.8			15.3					
Yield	0.00395			0.00953			0.00665					
Grade	5.0	g/t		2.8			1.9					
Calc:	4.6	g/t										

Table C-5 New Britannia Mine SCOF (100% -850 µm), 216 g/min, 2.0 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
105	6.99	7.30	11.8	24.63	216	3.90	1.2	75.37	223	3.96	1.5	1.16
75	12.64	13.20	21.4	33.14	569	10.28	1.0	66.86	582	10.33	1.4	2.83
53	15.50	16.19	51.5	37.84	933	16.86	1.4	62.16	948	16.84	2.2	7.30
37	18.81	19.65	99.6	41.55	1239	22.39	2.1	58.45	1258	22.35	3.6	15.60
25	34.32	35.85	196.0	68.74	866	15.65	3.5	31.26	900	15.99	10.9	33.85
-25	7.47	7.80	474.8	31.24	1711	30.92	4.6	68.76	1719	30.53	6.6	39.27
Total	95.73	100.00	138.9	46.00	5534	100.00	2.8	54.00	5630	100.00	5.1	100.00

Table C-6 New Britannia Mine -850 µm RMDIS, 513 g/min, 3.5 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	7.57	7.08	12.0	4.03	325	5.37	6.7	95.97	333	5.40	6.8	6.63
420	8.79	8.23	13.3	5.95	443	7.32	4.2	94.05	452	7.33	4.4	5.80
300	11.72	10.97	66.9	35.75	472	7.80	3.0	64.25	484	7.86	4.5	6.46
210	9.98	9.34	63.0	26.70	403	6.66	4.3	73.30	413	6.70	5.7	6.94
150	13.25	12.40	74.2	34.01	464	7.66	4.1	65.99	477	7.74	6.1	8.51
105	14.22	13.31	113.5	47.41	484	7.99	3.7	52.59	498	8.08	6.8	10.03
75	16.50	15.44	167.7	59.58	637	10.52	2.9	40.42	653	10.60	7.1	13.67
53	10.51	9.84	178.9	51.49	654	10.80	2.7	48.51	664	10.78	5.5	10.75
37	8.31	7.78	230.8	55.60	638	10.54	2.4	44.40	646	10.49	5.3	10.16
25	3.95	3.70	376.7	53.79	497	8.21	2.6	46.21	501	8.13	5.5	8.14
-25	2.05	1.92	751.8	35.14	1037	17.13	2.7	64.86	1039	16.87	4.2	12.91
Total	106.85	100.00	129.3	40.67	6053	100.00	3.3	59.33	6160	100.00	5.5	100.00

Table C-7 New Britannia Mine PCOF (100% -850 µm), 384 g/min, 3.0 psi

Size	CONCENTRATE				TAILS				FEED			
(µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
300	13.62	12.02	10	21.99	135	1.78	3.6	78.01	149	1.93	4.2	0.18
210	7.02	6.19	42	22.60	239	3.17	4.3	77.40	247	3.21	5.3	0.38
150	6.91	6.10	164	34.09	420	5.55	5.2	65.91	427	5.56	7.8	0.96
105	8.11	7.16	475	57.30	581	7.68	4.9	42.70	589	7.68	11.4	1.95
75	15.50	13.68	857	49.29	949	12.55	14.4	50.71	965	12.56	27.9	7.83
53	18.38	16.22	1358	37.32	1225	16.19	34.2	62.68	1244	16.19	53.8	19.43
37	21.62	19.08	2035	46.67	1300	17.18	38.7	53.33	1321	17.20	71.3	27.38
25	15.06	13.29	3578	69.75	861	11.38	27.2	30.25	876	11.40	88.2	22.44
-25	7.12	6.28	6373	67.77	1856	24.53	11.6	32.23	1863	24.26	35.9	19.45
Total	113.34	100.00	1649	54.29	7567	100.00	20.8	45.71	7680	100.00	44.8	100.00

Table C-8 New Britannia Mine SCUF (100% -850 µm), 390 g/min, 3.0 psi

Size (µm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
300	8.58	7.08	218	40.47	264	3.82	10.4	59.53	272	3.88	17.0	0.72
210	5.22	4.30	50	5.21	545	7.90	8.7	94.79	550	7.84	9.1	0.77
150	8.28	6.83	247	25.30	993	14.39	6.1	74.70	1001	14.26	8.1	1.25
105	11.49	9.48	306	27.26	1113	16.14	8.4	72.74	1125	16.02	11.5	2.00
75	19.92	16.43	1032	40.36	1121	16.24	27.1	59.64	1141	16.25	44.7	7.89
53	21.33	17.59	1434	30.73	925	13.40	74.6	69.27	946	13.48	105.2	15.42
37	24.43	20.15	3481	46.63	827	11.99	117.7	53.37	852	12.13	214.1	28.24
25	16.14	13.31	6114	68.29	428	6.20	107.2	31.71	444	6.32	325.6	22.38
-25	5.87	4.84	18344	78.13	684	9.91	44.1	21.87	690	9.82	199.9	21.34
Total	121.26	100.00	2888	54.23	6899	100.00	42.8	45.77	7020	100.00	92.0	100.00

Table C-9 New Britannia Mine BMDIS (100% -850 µm), 300 g/min, 3.0 psi

Size (µm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
300	7.64	6.25	261	52.91	48	0.81	37.0	47.09	56	0.92	67.8	0.63
210	3.90	3.19	934	49.37	82	1.40	45.5	50.63	86	1.43	85.8	1.24
150	6.28	5.14	1886	65.09	219	3.72	29.0	34.91	225	3.74	80.9	3.07
105	9.90	8.10	1943	67.09	409	6.95	23.1	32.91	419	6.97	68.5	4.83
75	19.29	15.78	1594	42.37	900	15.28	46.5	57.63	919	15.29	79.0	12.24
53	22.84	18.68	2721	50.41	1033	17.55	59.2	49.59	1056	17.57	116.8	20.79
37	25.58	20.92	3316	57.52	1137	19.32	55.1	42.48	1163	19.35	126.8	24.86
25	19.01	15.55	4140	73.60	593	10.07	47.6	26.40	612	10.18	174.7	18.02
-25	7.81	6.39	7856	72.27	1467	24.91	16.0	27.73	1474	24.53	57.6	14.31
Total	122.25	100.00	2900	59.76	5888	100.00	40.5	40.24	6010	100.00	98.7	100.00

Table C-10 New Britannia Mine -850 µm PCUF, 517 g/min, 3.5 psi

Size (µm)	CONCENTRATE Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	TAILS Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	FEED Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	15.03	13.81	38	9.53	852	11.95	6.3	90.47	867	11.97	6.9	0.56
420	17.38	15.96	104	21.06	1135	15.92	6.0	78.94	1152	15.92	7.4	0.80
300	19.61	18.01	172	31.47	1107	15.52	6.7	68.53	1126	15.56	9.5	1.00
210	11.34	10.42	723	47.77	551	7.72	16.3	52.23	562	7.76	30.6	1.60
150	11.00	10.10	1784	51.45	381	5.34	48.6	48.55	392	5.41	97.3	3.54
105	9.86	9.06	5090	61.15	352	4.94	90.5	38.85	362	5.00	226.6	7.63
75	9.33	8.57	11904	68.09	476	6.67	109.4	31.91	485	6.70	336.3	15.16
53	5.68	5.22	25560	74.96	500	7.02	96.9	25.04	506	6.99	382.6	18.00
37	5.12	4.70	36828	76.38	583	8.17	100.1	23.62	588	8.12	420.0	22.95
25	2.70	2.48	49528	82.25	340	4.77	84.8	17.75	343	4.74	474.0	15.11
-25	1.82	1.67	68582	84.95	854	11.98	25.9	15.05	856	11.82	171.6	13.66
Total	108.87	100.00	7230	73.16	7131	100.00	40.5	26.84	7240	100.00	148.6	100.00

Table C-11 New Britannia Mine -850 µm SB-14 Feed, 400-500 g/min, 4.0 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	11.37	11.21	6	1.79	673	10.70	5.6	98.21	684	10.71	5.6	1.21
420	22.43	22.11	180	52.57	846	13.46	4.3	47.43	869	13.60	8.8	2.40
300	14.23	14.03	523	41.89	405	6.45	25.5	58.11	420	6.57	42.3	5.56
210	12.78	12.60	871	60.37	311	4.95	23.5	39.63	324	5.07	56.9	5.77
150	12.54	12.36	1727	76.14	241	3.84	28.1	23.86	254	3.98	112.0	8.91
105	7.63	7.52	3421	81.96	277	4.41	20.7	18.04	285	4.46	111.8	9.97
75	6.28	6.19	5299	79.64	501	7.97	17.0	20.36	507	7.94	82.4	13.08
53	6.56	6.47	6579	77.37	666	10.59	19.0	22.63	672	10.52	83.0	17.47
37	4.26	4.20	8306	79.53	449	7.14	20.3	20.47	453	7.09	98.2	13.93
25	1.74	1.72	7080	44.83	646	10.27	23.5	55.17	647	10.13	42.5	8.61
-25	1.63	1.61	16516	64.44	1271	20.22	11.7	35.56	1273	19.92	32.8	13.08
Total	101.45	100.00	2183	69.36	6287	100.00	15.6	30.64	6388	100.00	50.0	100.00

Table C-12 New Britannia Mine -850 µm SB-14 Tails, 400-500 g/min, 4.0 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	5.76	5.15	8	2.26	478	5.66	4.2	97.74	484	5.65	4.2	0.92
420	16.23	14.52	84	27.56	738	8.73	4.9	72.44	754	8.81	6.6	2.23
300	14.03	12.55	201	40.80	436	5.16	9.4	59.20	450	5.26	15.3	3.11
210	15.46	13.83	322	36.52	379	4.49	22.8	63.48	395	4.61	34.5	6.15
150	17.01	15.22	489	61.32	350	4.14	15.0	38.68	367	4.28	37.0	6.11
105	11.29	10.10	910	61.32	420	4.97	15.4	38.68	431	5.04	38.9	7.56
75	9.61	8.60	1216	50.31	818	9.68	14.1	49.69	828	9.67	28.1	10.48
53	10.09	9.03	1520	46.63	1098	12.99	16.0	53.37	1108	12.94	29.7	14.83
37	6.86	6.14	2651	61.65	718	8.49	15.8	38.35	724	8.46	40.7	13.30
25	2.42	2.17	2839	25.91	985	11.65	20.0	74.09	987	11.53	26.9	11.96
-25	3.00	2.68	9576	55.46	2032	24.04	11.4	44.54	2035	23.76	25.5	23.35
Total	111.76	100.00	972	48.97	8451	100.00	13.4	51.03	8563	100.00	25.9	100.00

Table C-13 New Britannia Mine -850 µm KC-12 Feed, 400-500 g/min, 4.0 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	7.19	6.65	6	1.23	721	9.97	5.0	98.77	728	9.92	5.0	1.21
420	21.98	20.33	19	6.44	1248	17.26	5.0	93.56	1270	17.31	5.2	2.21
300	17.83	16.50	96	22.02	917	12.68	6.6	77.98	935	12.74	8.3	2.60
210	13.60	12.58	294	45.64	508	7.03	9.4	54.36	522	7.11	16.8	2.93
150	13.80	12.77	517	52.69	285	3.94	22.5	47.31	299	4.07	45.3	4.52
105	8.75	8.10	1221	62.80	255	3.53	24.8	37.20	264	3.60	64.5	5.68
75	6.92	6.40	2593	59.45	444	6.14	27.6	40.55	451	6.14	66.9	10.08
53	6.55	6.06	5000	64.04	573	7.93	32.1	35.96	580	7.90	88.2	17.08
37	4.80	4.44	6761	72.12	390	5.39	32.2	27.88	394	5.38	114.1	15.03
25	1.84	1.70	6858	35.18	609	8.43	38.1	64.82	611	8.33	58.7	11.98
-25	4.83	4.47	11577	70.01	1278	17.68	18.7	29.99	1283	17.49	62.3	26.68
Total	108.09	100.00	1625	58.68	7230	99.98	17.1	41.32	7338	99.98	40.8	100.00

Table C-14 New Britannia Mine -850 µm KC-12 Tails, 400-500 g/min, 4.0 psi

	CONCENTRATE				TAILS				FEED			
Size (µm)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Rec. (%)	Weight (g)	% %Weight	Grade (g/t)	Dist'n (%)
600	6.98	6.65	3	0.99	445	5.11	4.5	99.01	452	5.13	4.5	0.71
420	15.92	15.16	24	7.08	1105	12.70	4.6	92.92	1121	12.73	4.9	1.93
300	13.36	12.72	90	17.74	953	10.95	5.8	82.26	966	10.97	7.0	2.37
210	13.48	12.84	125	23.29	652	7.49	8.5	76.71	665	7.55	10.8	2.52
150	15.04	14.32	298	43.16	412	4.74	14.3	56.84	427	4.85	24.3	3.64
105	11.30	10.76	572	39.32	397	4.56	25.1	60.68	408	4.63	40.3	5.75
75	8.75	8.33	1431	38.94	682	7.84	28.8	61.06	691	7.85	46.5	11.25
53	8.36	7.96	2925	45.78	873	10.03	33.2	54.22	881	10.01	60.6	18.69
37	5.48	5.22	4745	59.48	591	6.79	30.0	40.52	596	6.77	73.3	15.30
25	2.66	2.53	5290	29.85	880	10.11	37.6	70.15	882	10.02	53.4	16.49
-25	3.67	3.50	6197	37.24	1712	19.68	22.4	62.76	1716	19.49	35.6	21.36
Total	105.00	100.00	1086	39.89	8701	100.00	19.7	60.11	8806	100.00	32.5	100.00

APPENDIX D

TEST RESULTS FOR THE 4-IN SUPERBOWL MODEL

Table D-1 to D-5: Coarse Silica Test

Table D-6 to D-10: Fine Silica Test

Table D-11 to D-16: Fine Magnetite Test

Table D-1 Coarse silica-tungsten feed

Slurring water flowrate (L/min): 1.5 Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 10.6 Feed Solids (%): 40.0

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
600	47.26	13.06	13.38	28.31		1504	15.04	13.43	0.89	8.95
425	48.80	13.49	18.44	37.79		1494	14.94	19.62	1.31	13.08
300	47.54	13.14	22.42	47.16		1145	11.45	20.76	1.81	13.84
212	32.01	8.85	15.88	49.61		844	8.44	16.43	1.95	10.95
150	37.01	10.23	19.14	51.72		807	8.07	19.05	2.36	12.70
106	27.99	7.74	14.24	50.88		626	6.26	13.98	2.23	9.32
75	26.52	7.33	12.29	46.34		634	6.34	12.30	1.94	8.20
53	19.64	5.43	8.78	44.70		445	4.45	8.61	1.93	5.74
37	16.90	4.67	6.11	36.15		489	4.89	6.56	1.34	4.37
25	18.86	5.21	5.37	28.47		421	4.21	3.98	0.94	2.65
-25	39.32	10.87	12.93	32.88		1591	15.91	15.30	0.96	10.20
Total	361.85	100.00	148.98	41.17	99.32	10000	100.00	150.00	1.50	100.00

Table D-2 Coarse silica-tungsten feed

Slurring water flowrate (L/min): 1.5 Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 15.0 Feed Solids (%): 40.0

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
600	38.75	12.27	13.35	34.45		1504	15.04	13.43	0.89	8.95
425	41.52	13.15	18.80	45.28		1494	14.94	19.62	1.31	13.08
300	41.26	13.06	22.06	53.47		1145	11.45	20.76	1.81	13.84
212	29.03	9.19	15.78	54.36		844	8.44	16.43	1.95	10.95
150	34.47	10.91	18.95	54.98		807	8.07	19.05	2.36	12.70
106	26.53	8.40	14.42	54.35		626	6.26	13.98	2.23	9.32
75	23.86	7.55	11.79	49.41		634	6.34	12.30	1.94	8.20
53	18.38	5.82	9.10	49.51		445	4.45	8.61	1.93	5.74
37	14.78	4.68	6.06	41.00		489	4.89	6.56	1.34	4.37
25	16.07	5.09	5.40	33.60		421	4.21	3.98	0.94	2.65
-25	31.18	9.87	12.81	41.08		1591	15.91	15.30	0.96	10.20
Total	315.83	100.00	148.52	47.03	99.01	10000	100.00	150.00	1.50	100.00

Table D-3 Coarse silica-tungsten feed

Slurring water flowrate (L/min): 1.5 Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 20.7 Feed Solids (%): 40.0

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
600	33.61	11.63	13.25	39.42		1504	15.04	13.43	0.89	8.95
425	36.50	12.64	18.34	50.25		1494	14.94	19.62	1.31	13.08
300	37.72	13.06	22.11	58.62		1145	11.45	20.76	1.81	13.84
212	26.61	9.21	15.74	59.15		844	8.44	16.43	1.95	10.95
150	30.89	10.69	18.81	60.89		807	8.07	19.05	2.36	12.70
106	23.82	8.25	14.30	60.03		626	6.26	13.98	2.23	9.32
75	22.01	7.62	11.93	54.20		634	6.34	12.30	1.94	8.20
53	16.75	5.80	8.93	53.31		445	4.45	8.61	1.93	5.74
37	13.86	4.80	5.98	43.15		489	4.89	6.56	1.34	4.37
25	15.98	5.53	5.56	34.79		421	4.21	3.98	0.94	2.65
-25	31.13	10.78	12.02	38.61		1591	15.91	15.30	0.96	10.20
Total	288.88	100.00	146.97	50.88	97.98	10000	100.00	150.00	1.50	100.00

Table D-4 Coarse silica-tungsten feed

Slurring water flowrate (L/min): 3.0 | Feed rate (kg/min): 2.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
600	38.05	12.78	13.33	35.03		1504	15.04	13.43	0.89	8.95
425	39.22	13.18	18.20	46.40		1494	14.94	19.62	1.31	13.08
300	39.73	13.35	22.54	56.73		1145	11.45	20.76	1.81	13.84
212	27.82	9.35	15.93	57.26		844	8.44	16.43	1.95	10.95
150	33.40	11.22	19.21	57.51		807	8.07	19.05	2.36	12.70
106	25.37	8.52	14.48	57.08		626	6.26	13.98	2.23	9.32
75	22.50	7.56	11.81	52.49		634	6.34	12.30	1.94	8.20
53	17.74	5.96	9.20	51.86		445	4.45	8.61	1.93	5.74
37	13.11	4.40	6.10	46.53		489	4.89	6.56	1.34	4.37
25	13.40	4.50	5.20	38.81		421	4.21	3.98	0.94	2.65
-25	27.32	9.18	12.60	46.12		1591	15.91	15.30	0.96	10.20
Total	297.66	100.00	148.60	49.92	99.07	10000	100.00	150.00	1.50	100.00

Table D-5 Coarse silica-tungsten feed

Slurring water flowrate (L/min): 5.4 | Feed rate (kg/min): 5.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 48.1

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
600	36.59	12.42	13.26	36.24		1504	15.04	13.43	0.89	8.95
425	38.31	13.00	18.09	47.22		1494	14.94	19.62	1.31	13.08
300	39.37	13.36	22.57	57.33		1145	11.45	20.76	1.81	13.84
212	27.86	9.45	16.11	57.82		844	8.44	16.43	1.95	10.95
150	33.13	11.24	19.17	57.86		807	8.07	19.05	2.36	12.70
106	25.40	8.62	14.53	57.20		626	6.26	13.98	2.23	9.32
75	23.26	7.89	11.96	51.42		634	6.34	12.30	1.94	8.20
53	16.72	5.67	8.77	52.45		445	4.45	8.61	1.93	5.74
37	13.69	4.65	6.24	45.58		489	4.89	6.56	1.34	4.37
25	13.62	4.62	5.08	37.30		421	4.21	3.98	0.94	2.65
-25	26.71	9.06	11.94	44.70		1591	15.91	15.30	0.96	10.20
Total	294.66	100.00	147.72	50.13	98.48	10000	100.00	150.00	1.50	100.00

Table D-6 Fine silica-tungsten feed

Slurring water flowrate (L/min): 1.5 | Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 10.7 | Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	23.96	8.90	16.80	70.12		372	3.72	15.83	4.25	10.55
212	25.97	9.65	17.37	66.88		762	7.62	18.50	2.43	12.33
150	38.09	14.15	24.32	63.85		1410	14.10	24.53	1.74	16.35
106	34.80	12.93	21.89	62.90		1515	15.15	21.72	1.43	14.48
75	29.85	11.09	16.53	55.38		1344	13.44	17.10	1.27	11.40
53	23.25	8.64	12.31	52.95		1116	11.16	12.02	1.08	8.01
37	25.19	9.36	13.38	53.12		1148	11.48	14.78	1.29	9.85
25	33.12	12.30	15.25	46.04		1233	12.33	13.50	1.09	9.00
-25	34.96	12.99	9.89	28.29		1100	11.00	12.06	1.10	8.04
Total	269.19	100.00	147.74	54.88	98.49	10000	100.00	150.00	1.50	100.00

Table D-7 Fine silica-tungsten feed

Slurring water flowrate (L/min): 1.5 | Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	22.08	9.63	16.63	75.32		372	3.72	15.83	4.25	10.55
212	23.58	10.28	17.13	72.65		762	7.62	18.50	2.43	12.33
150	33.48	14.60	23.84	71.21		1410	14.10	24.53	1.74	16.35
106	30.58	13.33	21.50	70.31		1515	15.15	21.72	1.43	14.48
75	25.23	11.00	16.48	65.32		1344	13.44	17.10	1.27	11.40
53	20.37	8.88	12.81	62.89		1116	11.16	12.02	1.08	8.01
37	20.97	9.14	13.50	64.38		1148	11.48	14.78	1.29	9.85
25	26.93	11.74	15.12	56.15		1233	12.33	13.50	1.09	9.00
-25	26.16	11.40	9.85	37.65		1100	11.00	12.06	1.10	8.04
Total	229.38	100.00	146.86	64.02	97.91	10000	100.00	150.00	1.50	100.00

Table D-8 Fine silica-tungsten feed

Slurring water flowrate (L/min): 1.5 | Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 20.7 | Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	20.43	10.39	16.77	82.09		372	3.72	15.83	4.25	10.55
212	21.09	10.73	17.21	81.60		762	7.62	18.50	2.43	12.33
150	29.01	14.75	23.48	80.94		1410	14.10	24.53	1.74	16.35
106	26.33	13.39	21.09	80.10		1515	15.15	21.72	1.43	14.48
75	22.41	11.40	16.82	75.06		1344	13.44	17.10	1.27	11.40
53	16.34	8.31	12.02	73.56		1116	11.16	12.02	1.08	8.01
37	18.22	9.27	13.41	73.60		1148	11.48	14.78	1.29	9.85
25	23.19	11.79	15.25	65.76		1233	12.33	13.50	1.09	9.00
-25	19.61	9.97	9.57	48.80		1100	11.00	12.06	1.10	8.04
Total	196.63	100.00	145.62	74.06	97.08	10000	100.00	150.00	1.50	100.00

Table D-9 Fine silica-tungsten feed

Slurring water flowrate (L/min): 3.0 | Feed rate (kg/min): 2.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	21.48	9.70	16.58	77.19		372	3.72	15.83	4.25	10.55
212	23.22	10.49	17.24	74.25		762	7.62	18.50	2.43	12.33
150	33.20	15.00	23.69	71.36		1410	14.10	24.53	1.74	16.35
106	30.75	13.89	21.56	70.11		1515	15.15	21.72	1.43	14.48
75	25.94	11.72	16.64	64.15		1344	13.44	17.10	1.27	11.40
53	20.11	9.09	12.80	63.65		1116	11.16	12.02	1.08	8.01
37	20.61	9.31	13.61	66.04		1148	11.48	14.78	1.29	9.85
25	24.95	11.27	14.84	59.48		1233	12.33	13.50	1.09	9.00
-25	21.09	9.53	9.69	45.95		1100	11.00	12.06	1.10	8.04
Total	221.35	100.00	146.65	66.25	97.77	10000	100.00	150.00	1.50	100.00

Table D-10 Fine silica-tungsten feed

Slurring water flowrate (L/min): 5.4 | Feed rate (kg/min): 4.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 42.5

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	22.43	9.46	16.45	73.34		372	3.72	15.83	4.25	10.55
212	23.98	10.12	17.06	71.14		762	7.62	18.50	2.43	12.33
150	34.74	14.65	23.55	67.79		1410	14.10	24.53	1.74	16.35
106	32.30	13.63	21.44	66.38		1515	15.15	21.72	1.43	14.48
75	27.45	11.58	16.00	58.29		1344	13.44	17.10	1.27	11.40
53	21.94	9.26	12.56	57.25		1116	11.16	12.02	1.08	8.01
37	23.11	9.75	13.60	58.85		1148	11.48	14.78	1.29	9.85
25	29.43	12.41	15.19	51.61		1233	12.33	13.50	1.09	9.00
-25	21.68	9.15	8.95	41.28		1100	11.00	12.06	1.10	8.04
Total	237.06	100.00	144.80	61.08	96.53	10000	100.00	150.00	1.50	100.00

Table D-11 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 1.5 | Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 10.0 | Feed Solids (%): 40.0

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	33.57	5.88	17.36	51.71		372	3.72	15.83	4.25	10.55
212	44.40	7.77	15.60	35.14		762	7.62	18.50	2.43	12.33
150	79.88	13.98	20.91	26.18		1410	14.10	24.53	1.74	16.35
106	74.60	13.06	18.69	25.05		1515	15.15	21.72	1.43	14.48
75	63.93	11.19	15.35	24.01		1344	13.44	17.10	1.27	11.40
53	55.55	9.72	10.98	19.77		1116	11.16	12.02	1.08	8.01
37	67.04	11.73	14.03	20.93		1148	11.48	14.78	1.29	9.85
25	44.84	7.85	10.43	23.26		1233	12.33	13.50	1.09	9.00
-25	107.49	18.81	12.10	11.26		1100	11.00	12.06	1.10	8.04
Total	571.30	100.00	135.45	23.71	90.30	10000	100.00	150.00	1.50	100.00

Table D-12 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 1.5 | Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 15.0 | Feed Solids (%): 40.0

Size	SB Concentrate					Feed				
(μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%) *	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	37.49	6.72	17.58	46.89		372	3.72	15.83	4.25	10.55
212	47.94	8.59	17.21	35.90	100.00	762	7.62	18.50	2.43	12.33
150	78.76	14.11	23.60	29.96	98.08	1410	14.10	24.53	1.74	16.35
106	71.72	12.85	20.71	28.88	95.35	1515	15.15	21.72	1.43	14.48
75	60.62	10.86	16.05	26.48	93.86	1344	13.44	17.10	1.27	11.40
53	56.41	10.11	11.83	20.97	93.51	1116	11.16	12.02	1.08	8.01
37	60.02	10.75	13.23	22.04	93.51	1148	11.48	14.78	1.29	9.85
25	38.61	6.92	8.92	23.10	92.21	1233	12.33	13.50	1.09	9.00
-25	106.56	19.09	14.65	13.75		1100	11.00	12.06	1.10	8.04
Total	558.13	100.00	143.78	25.76	95.85	10000	100.00	150.00	1.50	100.00

* The coarsest and finest two fractions are combined separately to calculate the recoveries; an average recovery is used for −53 and +37 μm fractions; the excess tungsten in +212 μm fraction is added to the next fraction. Calculation is same for the following tables.

Table D-13 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 1.5 — Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 20.0 — Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	38.34	7.31	17.43	45.46		372	3.72	15.83	4.25	10.55
212	46.52	8.87	17.16	36.89	100.00	762	7.62	18.50	2.43	12.33
150	73.19	13.95	23.49	32.09	96.82	1410	14.10	24.53	1.74	16.35
106	67.38	12.84	20.73	30.77	95.44	1515	15.15	21.72	1.43	14.48
75	58.37	11.12	16.25	27.84	95.03	1344	13.44	17.10	1.27	11.40
53	49.80	9.49	11.67	23.43	91.46	1116	11.16	12.02	1.08	8.01
37	53.36	10.17	12.84	24.06	91.46	1148	11.48	14.78	1.29	9.85
25	37.03	7.06	9.47	25.57	93.04	1233	12.33	13.50	1.09	9.00
-25	100.74	19.20	14.31	14.20		1100	11.00	12.06	1.10	8.04
Total	524.73	100.00	143.35	27.32	95.57	10000	100.00	150.00	1.50	100.00

Table D-14 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 1.5 — Feed rate (kg/min): 1.0
Fluidization water flowrate (L/min): 25.0 — Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	31.26	7.02	16.68	53.36		372	3.72	15.83	4.25	10.55
212	38.75	8.70	16.52	42.63	96.71	762	7.62	18.50	2.43	12.33
150	64.00	14.37	21.93	34.27	89.40	1410	14.10	24.53	1.74	16.35
106	59.91	13.45	18.83	31.43	86.69	1515	15.15	21.72	1.43	14.48
75	49.69	11.16	14.59	29.36	85.32	1344	13.44	17.10	1.27	11.40
53	44.34	9.96	11.28	25.44	87.09	1116	11.16	12.02	1.08	8.01
37	47.65	10.70	12.06	25.31	87.09	1148	11.48	14.78	1.29	9.85
25	33.47	7.52	9.77	29.19	89.63	1233	12.33	13.50	1.09	9.00
-25	76.28	17.13	13.14	17.23		1100	11.00	12.06	1.10	8.04
Total	445.35	100.00	134.80	30.27	89.87	10000	100.00	150.00	1.50	100.00

Table D-15 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 3.0 — Feed rate (kg/min): 2.0
Fluidization water flowrate (L/min): 15.0 — Feed Solids (%): 40.0

	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	34.85	7.20	17.03	48.87		372	3.72	15.83	4.25	10.55
212	45.23	9.35	16.66	36.83	98.14	762	7.62	18.50	2.43	12.33
150	75.55	15.61	22.43	29.69	91.44	1410	14.10	24.53	1.74	16.35
106	68.10	14.07	20.22	29.69	93.09	1515	15.15	21.72	1.43	14.48
75	55.86	11.54	15.59	27.91	91.17	1344	13.44	17.10	1.27	11.40
53	47.13	9.74	11.80	25.04	90.11	1116	11.16	12.02	1.08	8.01
37	45.92	9.49	12.35	26.89	90.11	1148	11.48	14.78	1.29	9.85
25	31.45	6.50	9.20	29.25	94.56	1233	12.33	13.50	1.09	9.00
-25	79.79	16.49	14.97	18.76		1100	11.00	12.06	1.10	8.04
Total	483.88	100.00	140.25	28.98	93.50	10000	100.00	150.00	1.50	100.00

Table D-16 Fine magnetite-tungsten feed

Slurring water flowrate (L/min): 5.4
Fluidization water flowrate (L/min): 15.0
Feed rate (kg/min): 5.0
Feed Solids (%): 48.0

Size	SB Concentrate					Feed				
Size (μm)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Recovery (%)	Mass (g)	Wt. (%)	Tungsten (g)	Grade (%)	Dist'n (%)
300	33.69	7.20	16.53	49.07		372	3.72	15.83	4.25	10.55
212	42.88	9.17	16.60	38.71	96.50	762	7.62	18.50	2.43	12.33
150	69.96	14.95	22.48	32.13	91.64	1410	14.10	24.53	1.74	16.35
106	61.87	13.22	19.83	32.05	91.30	1515	15.15	21.72	1.43	14.48
75	51.24	10.95	15.25	29.76	89.18	1344	13.44	17.10	1.27	11.40
53	46.52	9.94	11.67	25.09	88.43	1116	11.16	12.02	1.08	8.01
37	47.46	10.14	12.03	25.35	88.43	1148	11.48	14.78	1.29	9.85
25	33.55	7.17	9.45	28.17	92.33	1233	12.33	13.50	1.09	9.00
-25	80.69	17.25	14.15	17.54		1100	11.00	12.06	1.10	8.04
Total	467.86	100.00	137.99	29.49	91.99	10000	100.00	150.00	1.50	100.00