McGill University

A SIMPLIFIED GRAVITY-RECOVERABLE-GOLD TEST

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In Loving Memory

André Robert Laplante (1953 – 2006)

This thesis is dedicated in loving memory to Dr. André Robert Laplante, a great man taken from us well before his time, who unfortunately could not be here with me in the final moments to see this thesis come to fruition. Words cannot begin to describe the esteem to which I hold this man: he was teacher, employer, mentor and most importantly friend, all rolled into one. The path I walk today is one he so graciously provided; any student that has ever had the privilege to study under André can attest to the lengths to which he went for his students, constantly challenging them and encouraging them to take what they've learned a step further, believing they had the potential to do great things. Such encouragement is a rare thing indeed, especially amongst an institution as vast as a university, where it is easy for students to come and go unnoticed – André not only noticed, but invested himself at every opportunity.

The name André Laplante is a well-known one, not only amongst the Metals & Materials department of McGill University, but throughout a fair portion of both the academic and industrial communities. To say André was an educated man is an understatement, he had amassed an impressive collection of achievements over the years, from PhD and P. Eng, to Associate Professor at McGill University to say the least. What is truly remarkable was that despite being a world-renown authority on gold mineral processing, André was the epitome of humility, always receptive to learning more, even from the very students he taught. The fact that an experienced man such as this was still willing and able to learn something from young men and women thirty years his junior was probably the thing that impressed me most about him.

Despite his numerous accomplishments and the highly-demanding nature of his work (both as a teacher and as an engineering consultant), André lived a balanced lifestyle most of us can only dream of. He was a serious athlete, in better shape than men half his age, and he carried a boyish nature about him, which may explain his youthful vigour and perpetual fascination with the world around him. Most importantly he was a devoted husband and father whose life was filled with joy, thanks to his wife Carol-Susan and loving children Jérémie and Amélie, whom he cherished dearly and spoke of with unwavering pride and adoration. My heart and prayers lie with you and the rest of your family, in the hopes that the pain of his sudden loss diminishes quickly in the times to come, leaving you only with the warmth of his loving memory and pride in the honourable legacy he left behind.

It was an honour to work under you sir.
May you rest in peace and tranquility, now and forever.

Sincerely,

Jason O'neil Clarke
(May 1st 2006)

Abstract

Gravity-Recoverable-Gold (GRG) is defined as gold present in a particle in sufficient quantities as to be selectively recoverable from gangue via gravity methods. The McGill standard GRG test is an ore characterization test using three stages of sequential liberation and recovery with a Knelson KC-MD3 centrifuge to determine the size distribution of GRG. This thesis describes the development and testing of two simplified versions of the GRG test, using two and one stages of recovery respectively. Both tests use a feed mass of 20 kg, as opposed to the 40 to 100 kg normally used for the standard test. Eighteen differing ore samples were processed with the simplified GRG tests. For non-abrasive ores the one-stage simplified test returns a similar GRG content and size distribution, making the two-stage test superfluous. For abrasive ores, the one-stage test returns a GRG content that can be as much as 33% relative lower than that of the standard test, with a much finer size distribution. The two-stage test exhibited similar poor performance, though to a slightly lesser degree due to and additional stage of recovery attempted prior to grinding the abrasive material. The GRG lost typically reports to size fractions coarser than 25 µm, strongly suggesting smearing onto gangue particles. Because of the lower feed mass used, both simple tests are susceptible to the nugget effect; feed representativity also becomes challenging for ore samples of a head grade of 1 g/t or less.

Résumé

On définit l'or récupérable par gravimétrie (ORG, ou GRG en anglais) comme étant présent dans une particule à une concentration telle que la densité de la particule rend sa concentration possible par gravimétrie. L'essai standard d'ORG de McGill utilise trois stades de libération et récupération séquentielles à l'aide d'un concentrateur Knelson de laboratoire (KC MD3), et permet la détermination de la quantité et de la distribution granulométrique de l'ORG. Ce mémoire décrit le développement et l'évaluation de deux versions simplifiées de l'essai ORG, une utilisant un seul stade, et l'autre deux. Les deux essais utilisent une masse initiale de minerai de 20 kilos, plutôt que le 40 à 100 kilos de l'essai standard. Un total de 18 échantillons a été utilisé pour comparer les résultats de l'essai standard et ceux des deux essais simplifiés. Pour les minerais non-abrasifs, les deux essais simplifiés donnent des résultats semblables à ceux de l'essai standard; on préfère donc dans ce cas l'essai d'un stade. Pour les minerais abrasifs, l'essai d'un stade mesure une quantité d'ORG qui peut être de 33% inférieure à celle de l'essai standard, et qui est beaucoup plus fine. L'essai a deux stades aussi mesure une quantité d'ORG réduit, mais les résultats sont un peux plus proche grace a une stade extra pour récupération. L'ORG ainsi perdu se retrouve dans la fraction granulométrique supérieure à 25 µm, ce qui laisse suggérer que l'or est enduit sur les particules abrasives plutôt que broyé finement. Parce que la masse traitée est inférieure à celle de l'essai standard, les deux essais simplifiés sont plus vulnérables aux problèmes d'effet pépite et de représentativité, surtout pour les teneurs d'alimentation égales ou inférieures à 1 g/t.

Acknowledgements

My thanks again to Professor André Laplante for providing this project and mentoring this work from conception to completion, your tireless efforts were greatly appreciated and it has been an honour to work under you.

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The author would like to acknowledge my loving parents, Carleen and Stanley Clarke, whom so painstakingly dedicated their lives to providing me every opportunity possible for education and self-improvement. Heartfelt thanks to my best friend and brother-in-arms Neelkanth Patel for his continued friendship and support.

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Thanks to my office-mates Sunil Koppalkar and Wenwu Li, I wish you the best of luck with your academic, professional and personal pursuits. Also thanks to Monique Riendeau and my buddies next door Phuong Vo, Jessica Hiscox, Ehab and Mustafa Tarkan for making grad life the most fun period of my life. Special mentions to my fellow mineral processing group members for their academic assistance and friendship: Dr. Liming Huang, Zhixian Xiao, and Jan Nesset.

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1 Introduction

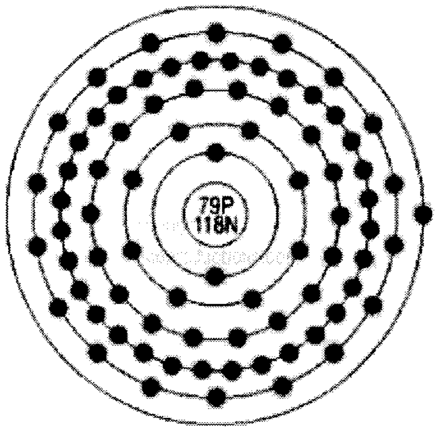
The discovery of precious metals such as gold and silver led these metals to quickly dominate as the primary means of legal tender for most civilizations over the last five millennia until the development of paper currency came about. Traces of gold currency in the form of bars and grains can be dated as far back as the Egyptians in 3000 BC (Huang 1996). Despite its fall from the primary means of currency, gold remains one of the most precious metals in this century ($515 USD/oz as of Dec 7th 2005), is currently mined worldwide, and shall continue to be so well into the future.

1.1 Introduction to gold

Gold is element number 79 on the periodic table of elements; it is categorized as a transition metal and has a face-centered cubic crystal structure. Webster's new lexicon dictionary defines gold as "a heavy, yellow, highly ductile metallic element". This concise definition summarizes most of gold's key mechanical properties that make it such a highly sought metal. Gold is indeed "heavy"; or rather gold has a high specific gravity at 19.3 g/cm^3 at 20 °C. Ductile is defined as "capable of being drawn out into threads" (Webster); from a metallurgical standpoint ductility is defined as the ability to be deformed via tensile forces without breaking. A similar term known as malleability is known as "the capability of being extended or shaped by hammering or rolling, without breaking". In this regard gold is the most ductile and malleable metal in existence: one troy ounce of gold (31.1 g) can be beaten out into a sheet 300 square feet wide (27.87 m^2) and 0.1 μm thin (Huang 1996). The primary uses for gold are jewellery, electronics (printed circuit boards), dentistry (gold teeth, gold fillings), coins, and bar hoarding (e.g. Fort Knox).

Figure 1 illustrates gold's electron shell configuration. Gold is labelled as a noble metal because it does not suffer from oxidation in an air or oxygen environment at ambient temperatures. Gold does possess one electron in the outermost ring that renders it capable of reacting with certain species under the right conditions; halogens react with gold readily, along with Aqua Regia (HNO_3 and HCl), and cyanide compounds such as NaCN (E-gold prospecting website,

http://www.e-goldprospecting.com/html/_gold_properties___gold_chemis.html).



$1s^2\ 2s^2p^6\ 3s^2p^6d^{10}\ 4s^2p^6d^{10}f^{14}\ 5s^2p^6d^{10}\ 6s^1$

Figure 1: Gold's Electron Shell configuration

1.2 How gold is formed

Gold deposits typically occur in hydrothermal veins that are formed from the remnants of igneous rocks subjected to geological events. Hypothermal deposits fall under three categories: epithermal, ranging in temperature from 50 °C to 200 °C; mesothermal, ranging from 200 °C to 300 °C; and hypothermal deposits, ranging from 300 °C to 500 °C. As the fluid rises through the near surface rocks, minerals start to precipitate inside the rock fissures. As the molten igneous rock flows and begins to cool, the silicates present tend to precipitate first out of solution. This causes the concentration of metals in the molten fluid to increase, and the molten igneous material flows through cracks and fissures in the stratum as it attempts to rise to the surface. Gold is one of the last elements to precipitate from the igneous fluid, thus it has a tendency to be found is small erratic veins dispersed throughout the surface rocks (Marsden and House, 1992). Gold is associated with several types of minerals, mostly sulphides, carbonates and silicates. The terrestrial abundance of gold is approximately 0.005 ppm (Alluvial exploration website, http://www.minelinks.com/alluvial/goldDeposits.html), and it is fairly evenly distributed throughout the globe; the primary producers of gold are South Africa, North America, China, and Australia (World Gold Council website, http://www.gold.org/value/markets/supply_demand/mine_production.html).

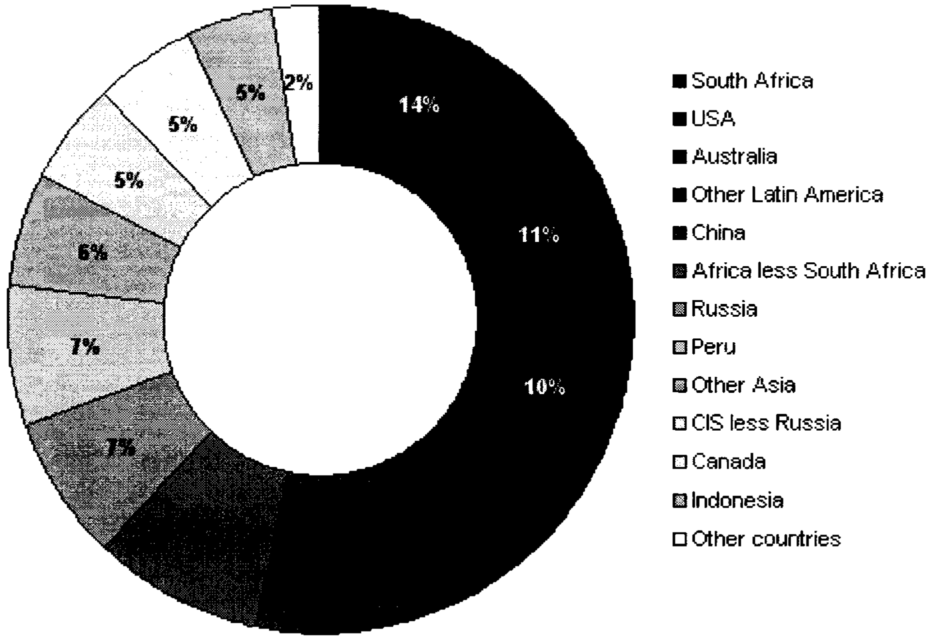


Figure 2: Top gold producing nations, 2004 (World Gold Council)

1.3 Processing options for gold-bearing ores

There exists a myriad of unit processes for gold recovery, and the selection of which process (or combination thereof) is a balance between numerous factors, such as: process economics, ore type, environmental constraints, abundance of process water, target mill tonnage, mine life, etc. The most popular unit processes for gold recovery over time have been gravity, cyanidation, and flotation. Since the focus of this thesis is on gold gravity recovery, the two other methods shall be introduced briefly for the sake of completeness in the gold recovery hierarchy, but shall not be referred to after this section. Treatment methods of refractory gold ores are beyond the scope of this work, and will not be addressed.

1.3.1 Gravity Recovery

Gravity recovery is the oldest method of gold recovery in existence. Gold's high specific gravity makes gold-bearing minerals particularly dense compared to other minerals, thus allowing them to be separated mechanically under the action of gravitational forces. Numerous gravity recovery devices have been developed through the ages: from simple tools such as pans, sluices and shaking tables, to more complex devices like spirals, jigs and centrifuges. Since gravity recovery is the focus of this thesis, the next chapter is devoted to describing such devices and providing background information on gold gravity recovery on a whole.

1.3.2 Flotation

Flotation is the process of floating a target mineral away from the rest of the gangue by creating environmental conditions that favour the target mineral's separation over other minerals present. Flotation is performed in a water-filled flotation cell under the agitation of an impellor that serves to aerate the system by supplying a steady amount of bubbles. Minerals are floated to the surface of the flotation cell as froth via the attachment of the particles to the air bubbles rising in the flotation column. The technique behind achieving an effective separation is finding a means, either chemically, physically or both, to render the target species hydrophobic (water-fearing) and the rest of the species hydrophilic (water-loving). Reverse flotation is the reverse practice of this principle in which case the undesired material is floated away and the target material is left behind in the flotation cell.

Sulphide minerals, being naturally-hydrophobic, are ideal candidates for flotation. Since gold is typically associated with sulphides it can usually be floated into the flotation concentrates of the sulphide species with minor chemical reagent addition and proper pH control.

Flotation is usually performed in a row of cells known as a *flotation bank.* One cell seldom has the residence time necessary to achieve proper reagent conditioning and selective recovery, but by putting several cells in series, very high recoveries can be achieved. Flotation is usually a closed-circuit operation between three types of cell banks: roughers, cleaners and scavengers. Roughers are an initial attempt at recovery where the easily-floatable material is recovered and passed on to the cleaning stage while the tailings are passed on to the scavenging stage. The cleaning stage involves upgrading the grade of the target species so the chemistry is boosted in favour of the target now that most of the gangue has been eliminated. The cleaner tails are typically recycled back to the roughing stage to give the particles an extra chance to be recovered. Finally the scavenger is a last-ditch effort to recover any target species left behind or that may have accidentally bypassed the rougher the first time around. The tailings of the scavenger are the final tailings, and the concentrate is recycled to the rougher (Wills, Mineral Processing Technology, 1985).

Flotation concentrates are usually smelted and the payout given in the form of a net-smelter-return (NSR). Typically, 92% to 97% of the gold present in most copper flotation concentrates is credited, which creates an incentive to recover it by gravity (whose payout typically exceeds 99%). Whenever a zinc concentrate is also produced, more selective flotation conditions are used in the copper flotation circuit, with the risk of deporting some of the gold to the zinc concentrate. Since there is no or very partial payment awarded for gold reporting to zinc concentrates, typical copper-lead-zinc circuits involve floating copper first, followed by lead and finally zinc to minimize gold reporting to the zinc concentrate. Another incentive for gravity recovery ahead of flotation is the failure of flotation to recover all the gold that would be recovered by a hypothetical gravity circuit operating upstream within the grinding circuit –i.e. a gravity circuit increases overall gold recovery.

1.3.3 Cyanidation

Cyanidation is the process of oxidizing gold in an alkaline cyanide solution to form a highly soluble aurocyanide complex that can then be separated from the gangue. A typical lixiviant used in cyanidation is sodium cyanide (NaCN), best operated in pH range 10.5 to 12. The process was first applied commercially in 1889, and the earliest patented cyanidation process was known as the Merrill Crowe process: gold-bearing minerals, after being liberated with sufficient comminution, were poured into a tank along with the lixiviant and agitated for a long residence time (up to 72 hours). The process is also known generically as *leaching*, and upon completion an aurocyanide solution could be bled away from the gangue minerals. The Merrill Crowe process added zinc dust to the clarified and de-oxygenated solution in order to precipitate the gold out of solution and recover it to be transformed into bullion.

In the seventies and eighties, discoveries in applications of activated carbon spurred the development of the *carbon-in-pulp* (CIP) and *carbon-in-leach* (CIL) processes, more effective methods of dissolved (complexed) gold recovery than Merrill-Crowe. Activated carbon grains have a very high surface area due to their intrinsically porous nature; the numerous micro- and macropores throughout the grains provide a plethora of adsorption sites for aurocyanide ions. CIP is a counter-current process where the gold is first leached from the gangue into solution, bled away from the gangue residue, and then contacted with activated carbon loaded to varying degrees. In order to maximize adsorption efficiency, the most loaded (or pregnant) gold solution is initially directed to the tank with the most coated carbon. Due to the high concentration of aurocyanide ions in solution, they will still adsorb themselves on the few adsorption sites left.

The barely-depleted aurocyanide solution is next exposed to a second tank with less-loaded carbon chips. Even more gold ions will adsorb this time since there are more available sites. This process continues thus until the final tank where the almost-barren aurocyanide solution is poured into the last tank where the carbon is least loaded. Since the number of adsorption sites greatly outweighs the number of remaining aurocyanide ions, the aurocyanide solution is thus further depleted to concentrations as low as 0.01 g/t. The CIL process is similar to CIP except that the leaching and carbon adsorption stages take place simultaneously, as the gold-bearing minerals are contacted with an alkaline cyanide solution in the presence of activated carbon, and thus the aurocyanide ions can be adsorbed as soon as they form. The loaded carbon is stripped in pressure vessels to yield a high-grade solution; this solution is usually processed by electro-winning to precipitate the gold out of solution and the anodes are melted down to produce bullion.

Even though cyanidation can achieve higher recoveries than flotation (for non-refractory ores), sometimes in excess of 95%, and activated carbon circuits are inherently more efficient than Merrill Crowe, the use of gravity recovery ahead of cyanidation typically results in higher overall recoveries of anywhere from 0.1% to 5%. Given the low capital and operating costs of gravity circuit, economic returns are very attractive.

1.4 Types of gold-bearing ores

There are numerous metallurgical categorizations of gold-bearing ores; the most common of which shall be listed below. Depending on what type of ore is present, a suitable gold recovery process must be selected to deal with the recovery challenges each ore-type present. Typical gold-bearing ores include: alluvials, oxides, free-milling, preg-robbing, and refractory ores. Readers intimately familiar with gold mineralogy may wish to proceed to the next section at this time.

1.4.1 Alluvials

Alluvials are highly-weathered/oxidized material such as sands and clays, transported hydraulically and then deposited over time via the action of water. Gold is usually found partially or completely liberated, making it fairly easy to recover. Sulphides are only present in trace amounts due to the high degree of weathering, and are therefore not an issue in this ore-type's metallurgical treatment. The weathering also has a tendency to cause the size distribution to become very fine amongst the gangue particles; therefore alluvials tend to be slimy when treated with water during processing.

1.4.2 Oxides

Oxides are similar to alluvials in that they also have been exposed to oxidizing conditions for long periods of time. The main difference is that whereas alluvials are found downstream of the deposit itself, oxides reside within the deposit itself, the upper layers of which are usually fractured. The fractured zone is often of lower grade than the non-fractured oxide zone below it. Both types of oxide zones often have very little carbonaceous matter due to the weathering; cyanide leaching is often the most effective method of gold recovery for this type of ore. Flotation does not recover the gold as well due to the oxidation of the sulphides originally present, negating their hydrophobicity. The non-fractured oxide zone requires more comminution than its fractured counterpart in order to achieve a standard target size distribution, but both respond to leaching in a very similar way.

1.4.3 Free-milling

Free-milling ores are defined as ores that are readily amenable to cyanidation with little or no preparation beforehand. The benchmark categorizing free-milling is a gold recovery of 95% via a 48-hour cyanidation leach at a grind of 80% passing 75 μm. Actual recovery can be lower because economics dictate a coarser grind size or shorter retention time. Typically free-milling ores are comprised of silicates and carbonates, sulphides can also be found in lesser proportions (1-10%); some popular sulphides include: pyrite (FeS_2), chalcopyrite ($CuFeS_2$), galena (PbS), sphalerite (ZnS), and arsenopyrite (FeAsS). Though some of these sulphides are usually present, the gold is loosely associated with them and can easily be liberated by grinding.

Flotation is a definite candidate for these ores, particularly the copper-lead-zinc trio where gold can be recovered as a by-product of the base metal flotation. Occasionally the flotation tails may be cyanided to scavenge any remaining gold.

Cyanicides are defined as compounds that consume cyanide. In the case of the gold-cyanidation process, pyrrhotite ($Fe_{1-x}S$) and stibnite (Sb_2S_3) are a few common examples; the presence of these materials is a hindrance as they consume free cyanide ions that would preferably be attacking the gold. This leads to increased reagent costs as higher cyanide addition rates are required, as more metal-cyanide complexes may be brought into solution form from the breakdown of the cyanicides.

1.4.4 Preg-robbing

Preg-robbing ores have a high concentration of carbonaceous matter that impedes cyanidation recovery. In the cyanidation process gold is dissolved by cyanide to form aurocyanide complexes ($Au(CN)_2^-$) that are separated from the rest of the material and eventually electrowon into gold bullion. However aurocyanide ions have a propensity to adsorb themselves onto carbonaceous matter, thus removing them prematurely from solution and becoming potentially lost to the cyanidation recovery process. Carbonaceous matter comes in three main forms, hydrocarbons, humic acids, and activated elemental carbon. An ore is categorized as mildly carbonaceous if its active carbon content is less than 1% carbon by weight or highly carbonaceous if the active carbon content is greater than 1%.

Preg-robbing can be a serious problem and can render a project's economics unprofitable if suitable countermeasures cannot be implemented to bolster gold recovery. Flotation is not an adequate solution, as the carbonaceous material floats along with the gold-bearing minerals and thus a separation solution must still be found to upgrade the concentrate to a smeltable grade. Gravity recovery has the potential to alleviate the burden imposed by preg-robbing carbonaceous matter. The Penjom gold mine, known as one of the worst preg-robbing ores in the world, successfully salvaged their gold operation through early gravity recovery prior to the cyanidation circuit. Up to 50% of

their gold was recovered from the grinding circuit's cyclone underflow by using a Knelson 30 and three specially developed In-Line Pressure Jigs (IPJs). The gravity concentrate is cleaned via another set of gravity devices prior to intensive cyanidation; this process change has rendered the Penjom operation profitable again.
(http://www.gekkos.com/Papers/Increased_Recovery_from_Preg-Robbing_Gold_Ore.pdf)

1.4.5 Refractory

Refractory ores have gold intimately associated with the sulphide minerals, especially pyrite and arsenopyrite. The amount of gold locked into solid solution is the recent measure of refractoriness, though gold can also be locked as isolated grains within a sulphide matrix. In either case the gold formed is inaccessible to cyanide solutions, thus rendering cyanidation useless in the ore's current form. Even flotation concentrate will suffer from the same problem of inaccessible gold, and will require the destruction of the sulphides present in order to upgrade the gold content to smeltable grades. Grinding to a finer size distribution, which usually helps increase recoverability, typically has no effect on refractory ores. Removal and/or destruction of the refractory material are the primary means to alleviate the problem. The sulphides can typically be destroyed via pre-treatment processes that fall under the general classification of roasting, pressure leaching and bio-oxidation (Marsden and House, 1992). The reaction stream is then neutralized and directed to cyanidation. It is worth noting that ores with a strong refractory component can also have an important coarse gold component, which is often the target of a gravity circuit.

1.5 Objectives of this study

The standard McGill gravity-recoverable gold (GRG) ore characterization test involves three stages of gravity recovery at progressively liberated size distributions. In order to provide statistically confident results a large sample mass, typically ranging from 60 to 100 kg, is required. Collection of a representative sample is crucial to the accuracy of the GRG test results; this issue will be discussed at length in sections 2.3 and 2.4. The standard GRG test can represent a significant effort in sample procurement, complexity, man-hours and analysis of results.

The objective of this study was to test two simplified procedures derived from the standard GRG test currently in practice. The simpler tests will be exposed to an array of 20 different ore samples in order to build a rudimentary database; the data collected can thus be compared to the vast historical database of the standard test. It is expected there will be some loss of information due to the simplified protocol; therefore a second objective of this test work is to assess the degree of information loss, and whether or not mathematical corrections can be applied to the data to extrapolate the missing information.

1.6 Thesis structure

This thesis begins with an introduction to the history of gold, gold-bearing ores and most common unit processes for gold recovery. In chapter two gravity recovery of gold is reviewed, highlighting such topics as: gravity-recoverable–gold (GRG), gold's grinding behaviour, gold sampling statistics, a review of some of the most common gravity devices and a detailed description of the Knelson concentrator. The third chapter is an in-depth walk through of the McGill standard GRG test, the two proposed simplified GRG tests, and all experimental procedures used in this research. Chapter four lists the results obtained for the samples processed with the simple GRG tests and a discussion comparing these results to those in the standard GRG test's database. Conclusions, recommendations and future work are presented in chapter five. The appendices contain a tutorial on the calculations used to generate the GRG results and a full chronological listing of experimental data.

2 GRG Overview

The concept gravity-recoverable-gold, while far from new, has often suffered from confusion/misinterpretations due to differences in nomenclature amongst the gold mineral processing community. Therefore the aim of this chapter is to provide the reader with a clear definition of gravity-recoverable-gold in order to preclude any potential misinterpretations of the subsequent results and findings. Following the definition, gold grinding and sampling theories related to gravity-recoverable-gold will be explained to educate the reader on the potential caveats of GRG processing.

2.1 Introduction to GRG

2.1.1 GRG definition

Gravity Recoverable Gold (GRG) is defined as gold in particles whose gold content is high enough to make them sufficiently distinct from other particles present as to be recovered selectively via gravity methods in a relatively small yield, typically less than 0.1%. The term was coined in order to eliminate possible confusions that arose from the previous historical term "free gold", which was deemed too vague, since it could either stand for fully-liberated gold particles (i.e. particles free of gangue) or gold-bearing particles that could easily be recovered via gravity, or even cyanidation –i.e. the concept that a free-gold milling ore typically returns a 95% cyanidation recovery. GRG also excludes gold present in very small quantities in particles that can be separated from non-sulphide gangue by gravity, typically sulphides. These particles can also be recovered by gravity, but at much higher yields, which makes the use of semi-continuous centrifuge units impractical. These particles are often referred to as gold carriers, and they are more often recovered by flotation than gravity.

2.1.2 Marginal GRG and non-GRG

In addition to the term GRG, two other related terms have been generated to help describe the other types of gold-bearing particles recovered/rejected by the LKC: namely non-GRG and marginal-GRG. Non-GRG refers to gold-bearing particles that are not amenable to gravity recovery with the standard GRG test either due to inadequate liberation, high gangue density, or particle shape. Marginal GRG refers to gold-bearing particles that are at the threshold of selective gravity recovery, usually either almost too fine or distorted to be recovered by the LKC operated at its standard velocity. A minor amount of marginal GRG is typically recovered via a second pass through the LKC at 60Gs after the bulk of the GRG has been recovered, but it can be recovered more effectively using a centrifuge concentrator with a higher centrifugal force such as the Falcon SB40 or even a KC-MD3 rotated at high velocity. The increased centrifugal forces have proven beneficial in recovering these "difficult" particles. By convention, marginal GRG is considered part of the non-GRG, since it is not recovered by the standard GRG test. It is recovered, as GRG is, at very low weight yields, which indicates that it is not gold present in gold carriers –e.g. gold in solid solution in arsenopyrite.

2.1.3 GRG Spectrum

Other than the actual deportment in the standard GRG test, there is no clear threshold at which GRG ends and marginal-GRG begins; though the centrifuge magnifies the density differences between particles, a certain minimum density difference must already exist. A similar argument can be made for particle size: terminal settling velocities are increased in a centrifuge field, but retention times are generally in the order of one second, which limits recovery at fine size.

In order to help the reader gain a better understanding of the density difference thresholds between non-GRG, marginal GRG and GRG, Figure 3 presents a simple estimation based on density considerations only. Two cases are presented: one in which the gangue is composed of silicates or "white sands" whose s.g. is approximately 3 g/cm^3; the other using black sands such as magnetite or massive sulphide ores whose s.g. is approximately 5 g/cm^3. The gangue density can be seen to have a direct effect on the composite particle's overall s.g. that ultimately affects its gravity recoverability. All particles below are assumed to have the same volume and shape, but different gold volume fractions, from which the overall density can then be calculated via the equation:

$$\text{Overall SG} = \text{Vol fraction}_{Au} * SG_{Au} + \text{Vol fraction}_{Gangue} * SG_{Gangue}$$

One must note that shape factors and particle size can also affect a particle's gravity-recoverability, but the effect of shape on gravity-recoverability is beyond the scope of this example and this project. Shape, size or density and that Figure 3 deals with density.

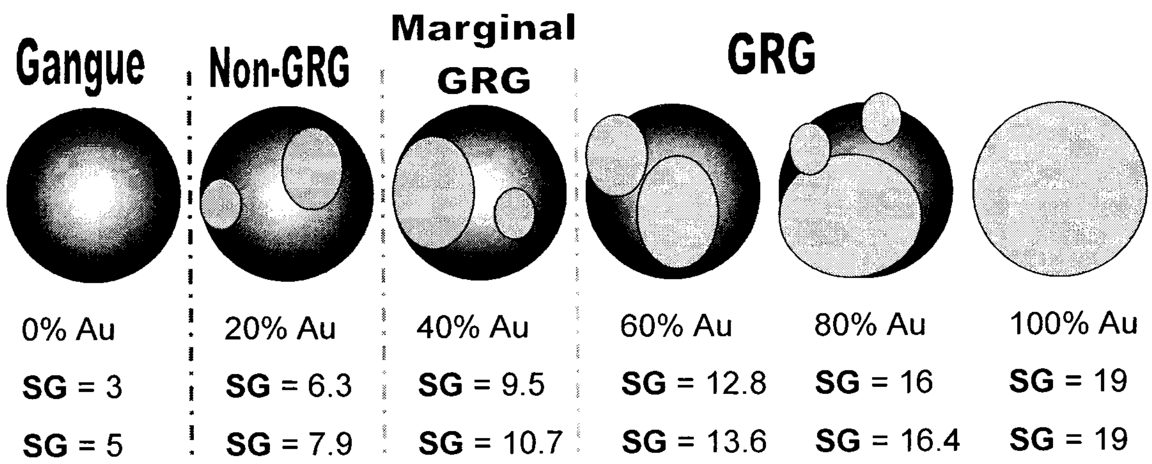


Figure 3: GRG Spectrum

Notwithstanding the above discussion, experimental results presented in chapter 4 will clearly demonstrate that particle size, rather then density, best identifies marginal GRG.

2.1.4 Why are we interested in GRG?

Generally plants are interested in recovering GRG because it represents a fraction of the incoming gold that can be recovered easily by gravity with little effort and at a lower cost than either flotation or cyanidation. By removing this easily recoverable gold from the circuit early, the rest of the circuit can be focused on the gold that is harder to recover. Also some of the GRG is sometimes difficult to recover via processes such as cyanidation (increased leach time for GRG) or flotation.

GRG tends to recycled numerous times through a grinding loop before it is of proper size and liberation; this can lead to the build up of major circulating loads of gold, often in the range of 1000 – 6000% gold. The longer this gold circulates the more likely it is to interact with grinding media, grinding equipment, gangue minerals and dissolved species in a manner that may lower its downstream recovery, therefore the inclusion of a gravity circuit to treat a bleed stream of the gold circulating load is often an effective action to minimize these interactions. The inclusion of gravity recovery can help bolster overall plant recovery: it has been shown in some cases that overall recovery can increase by 0.1-1.0% for every 10% of gravity recovery (A.R. Laplante, 2000). This reference mentions economic impact, but the 0.1-10% comes from a different source). This increase in recovery generates additional revenue that well outweighs the modest initial capital and operating costs.

The recovery of GRG from grinding circuits can also provide other numerous benefits to the overall gold recovery process. For instance, gravity recovery devices are simple to operate and can provide substantial recovery for a minimal capital investment. Gravity recovery devices do not require chemicals, which can be an asset if there are environmental restrictions on a plant; the only two inputs most gravity recovery devices require are water and electrical power, which makes them an ideal choice in areas where the two resources are abundant and inexpensive. Yet another potential benefit of a gravity pre-concentration step is a reduction in the amount of valuable material to be processed by the downstream circuit. With lower amounts of gold being fed downstream, it is likely that less/smaller equipment will be necessary to recover the remainder at comparable overall recoveries.

2.1.5 Where GRG is located in a plant

The best locations to sample GRG content are grinding and/or classification circuits. GRG is liberated by comminution steps such as crushing and grinding. Conversely, excessive grinding may render them non-gravity recoverable, some of which will be marginal GRG. Due to gold's high density, GRG largely reports to the cyclone underflow (CUF) of grinding loops, with very little GRG reporting to the COF. In order to better understand the transformations GRG undergoes during comminution, a previous investigation by Banisi on the grinding and breakage characteristics of gold will be reviewed in the following section.

2.2 Gold's grinding and breakage characteristics

Gold's malleability and softness make its grinding characteristics atypical for a mineral. Banisi (1990) conducted an in-depth investigation into gold's breakage and grinding characteristics, which shall be summarized in this section. However before that summary can be thoroughly understood, a brief review of the terminology and mathematics used to describe breakage characteristics is now presented.

2.2.1 The breakage function (b_i):

The breakage function is a measure of the amount of material broken from one size class that reports to other, finer, size classes, upon single breakage. The notation for the breakage function is b_{ij}, where the subscript i denotes the original size class of the particle being broken, and j denotes the size class of interest where the fraction of broken material reports. There is a second breakage term known as the cumulative breakage function B_{ij}. The cumulative breakage function is defined as the fraction of broken material from size class i that becomes finer than size class j upon a single breakage. The equation below gives the relationship between b_{ij} and B_{ij}:

$b_{ij} = B_{ij} - B_{i+1j}$ for $i>j$ **Equation 2: Breakage function**

It should be noted that each size class has its own distinct breakage function that is relatively independent of the comminution environment.

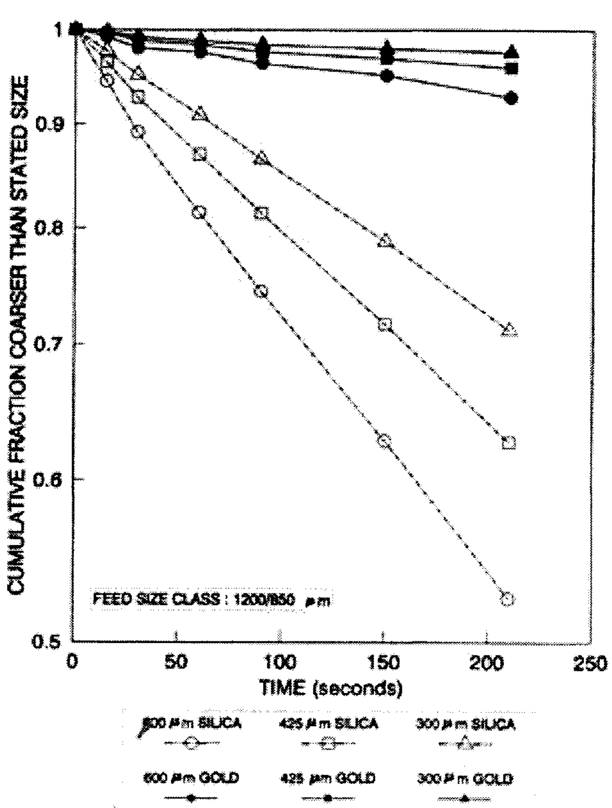


Figure 4: Breakage functions for Gold and Silica (Banisi)

2.2.2 The selection function ($S_i(t)$):

The selection function, also known as the specific rate of breakage, is a measure of the grinding kinetics of the mineral. The rate constant of the grinding process is first order for the disappearance of material from a size class due to breakage. Two terms are used to calculate the breakage rate: $M_i(t)$ which is defined as the mass remaining in size class i after a grinding time of t; and the selection function $S_i(t)$ which is the rate constant for size class i. The rate of breakage is thus defined as:

$dM_i(t) / dt = - S_i(t) * M_i(t)$ **Equation 3 : Rate of breakage**

The equation can be rearranged to solve for the amount of material left in the coarsest size class i at any time t.

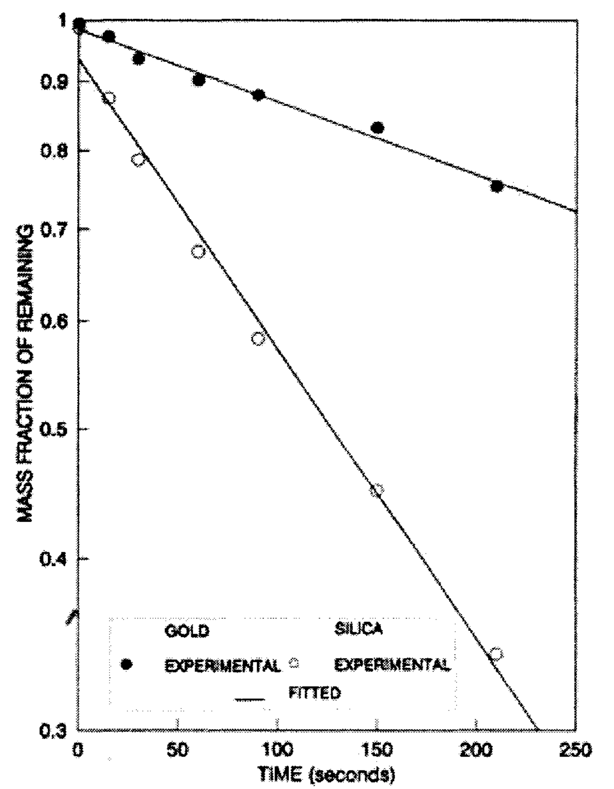


Figure 5: Selection functions for Gold and Silica (Banisi)

Expressions describing how to calculate the breakage and selection functions for several size classes at once can be found in the references, but are beyond the scope of this thesis and thus shall not be addressed here. The interested reader is encouraged to consult the reference (Banisi 1990) for further details.

2.2.3 Breakage study results

Banisi (1990) compared gold and silica's breakage characteristics using a small lab porcelain mill. The incremental grinding was stopped at regular time intervals (every 15 seconds for the first minute, then once at 90, 150 and 210 seconds) and the product screened before being returned to the mill for further grinding. Over the course of the grinding period 75% of the gold flakes remained in the coarsest size class (+850 μm), and 92.8 % either in the parent size class or the size class immediately finer (600-850 μm). Silica behaved quite differently, with only 34% remaining in the coarsest sized class and 52.6% by the second coarsest size class. Using linear regressions on the data at hand, Banisi concluded that the selection function for silica was 0.011 s^{-1}, which is roughly six times that of gold. Banisi's work also contains some useful measurements of the size distribution, number and standard deviation of the sizes of gold flakes during this grinding process. Those figures, contained in Table 1, shall be used in the results section of this thesis as a basis for assumptions of the weight of gold flakes in certain size classes.

Grinding (s)	Mean (g)	Standard Deviation (g)	Degrees of Freedom
Size Class +850 μm			
0	0.0048	0.0024	51
15	0.0045	0.0024	51
30	0.0048	0.0023	51
60	0.0049	0.0025	29
90	0.0049	0.0038	20
150	0.0049	0.0020	18
210	0.0053	0.0029	67
Size Class +600 μm			
30	0.0030	0.0013	10
60	0.0032	0.0009	11
90	0.0033	0.0015	13
150	0.0031	0.0009	17
210	0.0029	0.0011	58
Size Class +425 μm			
30	0.0010	0.0005	6
90	0.0011	0.0004	11
150	0.0016	0.0007	10
210	0.0015	0.0005	63
Size Class +300 μm			
30	0.00046	0.00019	7
60	0.00047	0.00018	6
90	0.00040	0.00011	5
150	0.00063	0.00084	6
210	0.00060	0.00062	9

Table 1: Mean and Standard Deviation of gold flake weight (Banisi, 1990)

2.2.4 Gold's deformation during the grinding process

Banisi used a scanning electron microscope (SEM) to observe the deformation of gold flakes during the grinding process, to further clarify the cause for gold's slow breakage characteristics when compared to that of silica. The study revealed that a typical gold particle would undergo a deformation cycle during grinding. Banisi identified three main shapes for gold flakes: the first category is gold flakes that are flat and round; the second is where the gold flake's shape becomes irregular; and lastly there are totally distorted gold flakes. Though the distinction between the categories is subjective since these assessments are made on SEM photographs, the basis behind the distinctions is the probability of the flake folding or breaking.

The first category of flakes (flat and round) has a much higher likelihood of folding rather than breaking, and thus will either form a spherical or cylindrical shape when folding occurs (Figure 6). Banisi dubbed these flakes "young", as their surface showed no sign of breakage. A second category had partially serrated edged with some evidence of crack propagation. These were referred to as "middle-aged" flakes, as they displayed signs of partial breakage. Finally the third category (Figure 7), dubbed "old" flakes, had highly serrated edges and heavily propagated cracks, hence a very high probability of breakage. Young flakes may continue to cyclically regenerate themselves by folding or and then flattening again, or they can start to develop cracks that advance them to middle aged flake morphology. The further crack propagation extends, the greater the probability of breakage. Occasionally partial breakage can occur where one part of the flake flattens whilst the rest breaks up into smaller flakes. It is this cycle of rejuvenation and cold-welding that causes gold to grind so much slower than other minerals.

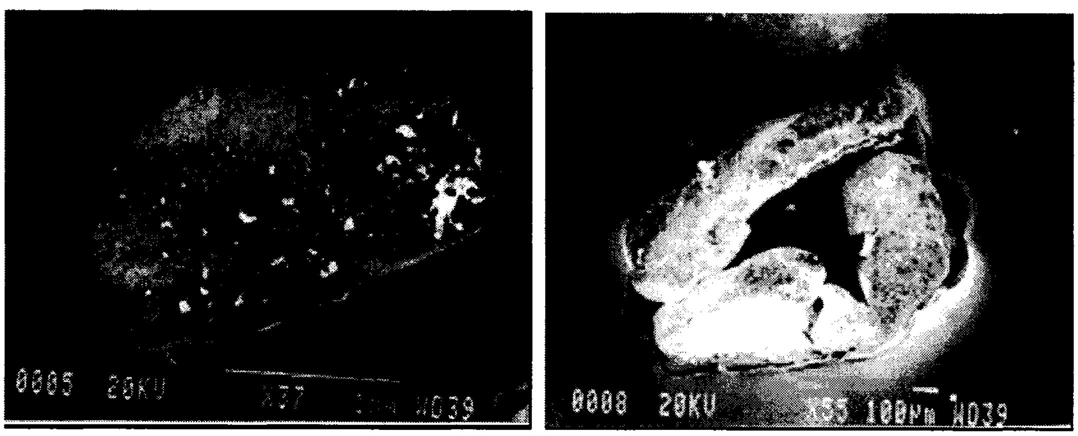


Figure 6: Young flat flake (left), Middle-aged folded flake (right) [Banisi]

The study not only addressed the deformation cycle of gold, but also two other prevalent issues involved in the grinding of gold: the smearing of gold onto other particles; and the embedding of other particles into gold's surface. During two grinding tests involving silica in isolation versus a mixture of gold and silica it was found that silica particles would embed themselves on gold particle's surface. Silica acted as a scouring agent inside of the mill given its greater hardness than gold. Gold smearing onto silica particles was also observed during the test. Both phenomena have the potential to cause mineral separation difficulties during subsequent processes depending on the extent of the interaction. In the case of particles embedding themselves in gold's surface may either lower the grade of gold recovered or even lower the recovery in the case of gold flotation. Embedding is not a significant issue for gravity recovery since the change in mass is insufficient to affect the particle's gravity recoverability. However gold smearing can pose significant problems for gravity separations if the gangue minerals on which the gold smears are low density, as these would be almost impossible to recover by gravity. Smearing on non-floatable minerals ahead of flotation could be even more serious, as overall recovery would be threatened.

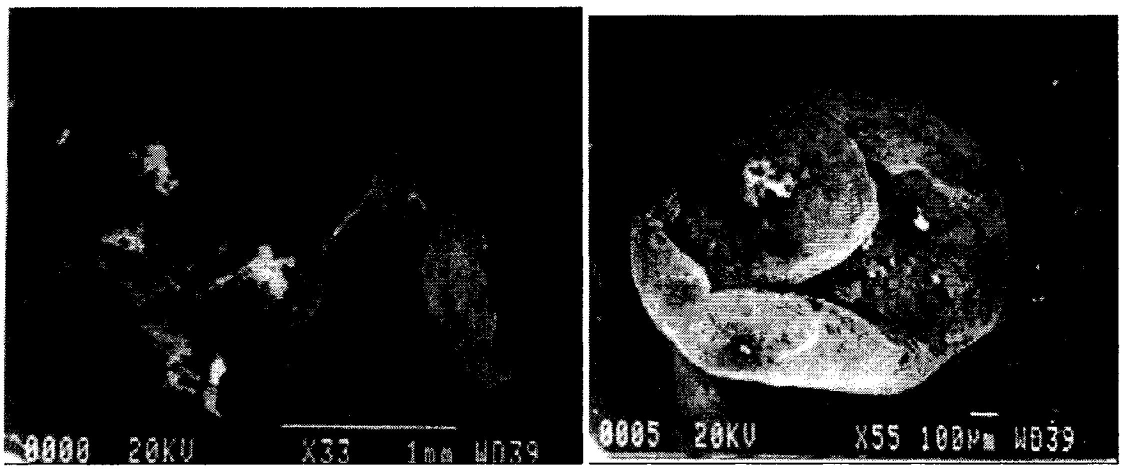


Figure 7: A totally distorted old flake (left), a newly-formed round flake (right) [Banisi]

2.3 Sampling Statistics for GRG

One of the most significant challenges of accurate GRG characterization is the procurement of a truly representative sample of the population of interest. The difficulty lies in obtaining a sample small enough in mass to be processed in a laboratory quickly yet devoid of as much sampling error as possible. Two terms used to measure the reliability of a sampling procedure are *sampling precision* and *sampling accuracy*. Sampling precision refers to the repeatability of the sampling process i.e. can similar sub-samples be obtained if the sampling procedure was attempted several times. Sampling accuracy is a measure of the lack of bias in the sampling procedure i.e. if the ore body were to be sampled using a different but reliable technique, would a similar sample still be obtained on average? Sampling accuracy is maximized by correct sampling techniques that yield a sample truly representative (unbiased) of a population. Sampling precision, which will be addressed here, is largely dictated by sample size.

Sampling errors are quantified by their variance. In the case of multi-step sampling, the total sampling variance is the sum of the individual variances of each step. It can only be minimized if measures are taken to minimize the sampling variance of each step.

The best way to minimize sampling variance is an alternation between mass reduction and size reduction. Mass reduction entails making the sample mass smaller but preserving the size distribution of the sample, whereas size reduction entails preserving the mass of the sample but reducing the size distribution in order to generate more particles. Mass reduction is necessary especially for assaying purposes since the lab cannot assay kilograms of ore. Size reduction is necessary to preserve the grade measured by the assays: by reducing the size distribution, more gold particles become liberated from each other, which increases the probability of accurate sampling. If the size reduction step is incapable of reducing the maximum amount of gold present in particles, it is totally ineffective and the subsequent mass reduction step will incur a significant sampling error. This becomes a significant problem as GRG becomes progressively liberated, because of its malleability.

In order for a sample to be deemed representative of the population from which it was extracted, it must contain similar relative proportions of all the original constituent elements present in the population. In the case of gold, this implies that the sample should contain an identical grade to the original ore body, not only overall grade, but on a size-by-size basis as well. However given gold's relatively low abundance in most ores this can become a difficult criterion to meet. A first estimate of the fundamental sampling error can be obtained from Gy's equation:

$\sigma^2_{(FE)} = CLFGD^3/M_s$ where:

- C is the composition factor, defined as the mass of ore per volume of gold (g/cm^3)
- L is the liberation factor, approximated by $L = (D_l/D)^{0.5}$ where D_l is the maximum gold grain size
- F is the particle shape factor, 1 for spheres, 0.2 for flakes, and 0.5 on average
- G is the size distribution factor, 1 for mono-sized material, 0.25 for un-sized products
- D is the maximum particle size, i.e. D_{95} (cm)
- M_s is the sample mass (g)

However Gy's equation does not work well for GRG, as it tends to overestimate the amount of gold present in discrete GRG particles. A combination of flake weights reported by Banisi and Poisson's law can help to provide a better measure of the sampling mass requirements for GRG assessment[1]. Figure 8 (Putz, 1994) illustrates the relative sampling error as a function of sample mass and grade, for two average gold weights per particle (these can be individual gold particles or the total gold content per particle).

[1] A similar approach by Clifton et al (1969) concluded that a minimum of 20 gold particles is necessary to obtain a 95% probability that the assay will return a value within ±50% of the true gold content.

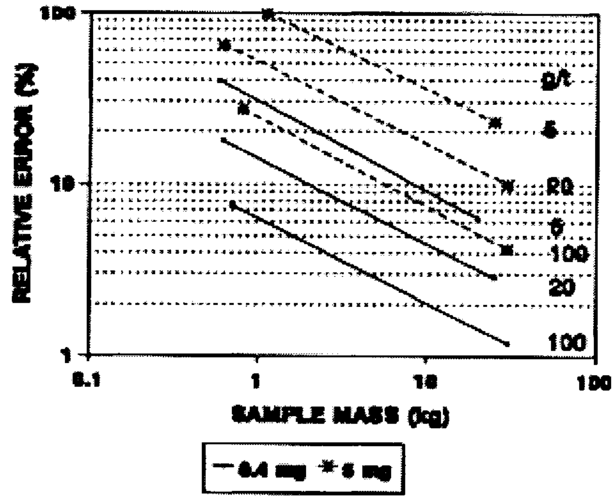


Figure 8: Relative Error vs. Sample Mass and grade (Putz, 1994)

For liberated gold particles, Table 1 shows that a 5 mg weight corresponds to the 850-1200 μm size fraction, whereas 0.4 mg corresponds to the lower limit of the 300-425 μm size fraction. As a further example, Figure 8 shows that at a grade of 5 g/t, a 20 kg sample can characterize gold up to around 300 μm without problems, but not coarser gold such as the 850-1200 μm range. The assaying stage introduces its own errors to the sampling data, but this can be minimized with size reduction or stratified sampling (which is the approach used in the GRG test).

2.4 Assaying Statistics for GRG

Due to gold's low concentration in most ores, the number of gold particles present in any given size is rather finite when compared to the number of gangue particles in the same size class. This problem was addressed in the previous section with respect to the mass of the sample used to test for GRG. It was shown that a 20 kg sample grading 5 g/t would yield adequate information for gold grains with an average weight of 0.4 g/t. With gold grains at 5 mg, a 20-kg sample was clearly inadequate. The problem becomes critical for the assaying stage, for which masses of 10 to 100 g (30 g is most common) are used, particularly if sized data are required.

Consider the 600-850 μm size class, where the average mass of a gold flake is approximately 5 mg. In order to have 20 particles for assay that is a total gold mass of 100 mg or 0.1 g. If the grade of the original sample is 1 g/t, theoretically **assaying** a 100 kg mass would be needed to obtain the 0.1 g of gold. This is clearly impractical, but the low mass (aliquot) used in assaying presents a challenge even for finer size fractions. Table 2 presents the impact of a single gold particle on the grade of aliquots of 15, 30 and 60 g for various size fractions. Assuming a feed grade of 5 g/t and a requirement of 5 grains per aliquot, the maximum contribution of a single particle should be1 g/t. Table 2 shows that with an aliquot of 30 g, all size fractions above 106 μm fail to meet the criterion. By using a gravity recovery unit to concentrate all gold particles capable of creating the Nugget Effect into a very fine mass that is fully assayed, this problem is circumvented entirely. This approach to assaying has been identified as "stratified sampling." (Cochran, 1946) Further, this approach, when used with very low weight recoveries into the concentrate, separates effectively GRG from non-GRG.

Gold Size (μm)	Mesh	Wt of 1 particle (mg)	0.5 AT 15.6 g	1 AT 31.3 g	2 AT 62.5 g	5 AT 156.3 g	32 AT 1000 g	64 AT 2000 g	320 AT 10000 g
1650	10	88	5113.9	2556.9	1278.5	511.4	50.8	40.1	7.8
833	20	11	639.2	319.6	159.8	63.9	9.88	4.94	0.87
589	28	4	232.4	116.2	58.1	23.2	6.25	1.80	0.61
295	48	0.5	29.1	14.53	7.26	2.91	0.46	0.23	0.06
208	65	0.17	9.88	4.94	5.23	0.87	0.15	0.06	0.03
147	100	0.061	3.49	1.74	0.87	0.29	0.06	0.03	0.03
104	150	0.021	1.16	0.58	0.29	0.12	0.03	0.03	0.03
74	200	0.0078	0.58	0.29	0.03	0.06	0.03	0.03	0.03
45	325	0.0017	0.12	0.06	0.03	0.03	0.03	0.03	0.03
38	400	0.001	0.06	0.03	<0.03	<0.03	<0.03	<0.03	<0.03
20		1.56E-04	0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
5		2.41E-06	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
2		1.56E-07	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03

Table 2: Nugget Effect on gold assays (Putz, 1994)

2.5 Screen Calibration

A procedure for a screen calibration test was created with the goal of assessing the effect of screening bias on the simplified test's GRG results. Since the entire database of ore samples was screened with the same Tyler stacks of concentrate and tailings screens, the same bias should be present for all samples processed. Repeated screen wear and mending of the tailings stack screens led to the selection of the concentrate stack as the more reliable set, therefore the concentrate screen results were assumed to be accurate and used as a base reference for calibrating the tailings screens. Theoretically, if provided the same sample, both the concentrate and tailings screens mass retained per size class should be identical. However the large variance of the screen manufacturer's aperture size does far from guarantee that this will be the case, as can be seen in Figure 9 where the tailings deck data points do not lie perfectly on the concentrate deck's curve.

The goal of the screen calibration is to ascertain what the actual aperture sizes are for the tailings deck, and using that information, adjust the GRG size distribution accordingly. The calibration procedure relies upon the law of conservation of mass for both the gold and the gangue, therefore the overall GRG content and feed grade are preserved, just the distribution is altered. The reason for the adjustment is so that both the concentrate and tailings values apply to the same particle size scale.

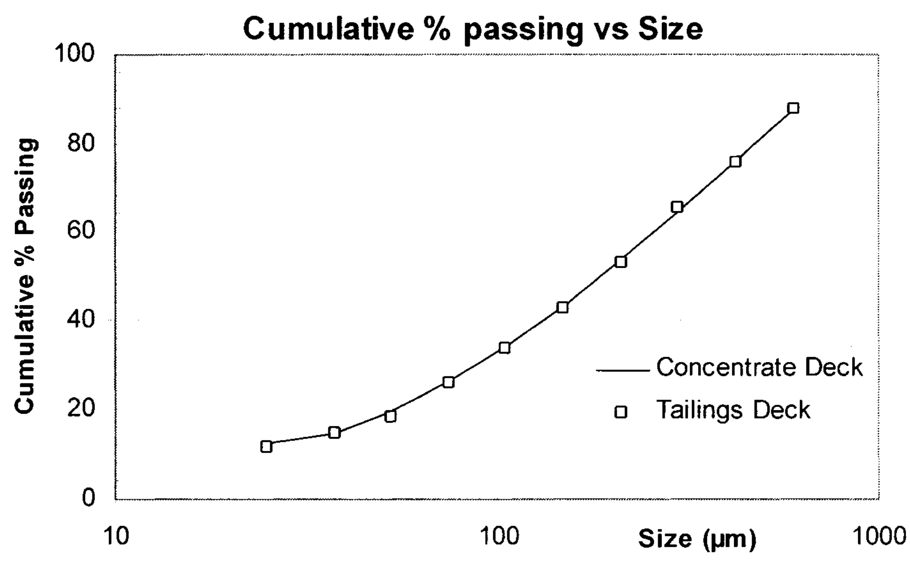


Figure 9: Difference in Cumulative % Passing for Concentrate and Tailings Decks

A reference sample with a fairly evenly distributed size distribution should be selected for this calibration. A 3 kg sample of low grade stage 1 tailings from a standard test was used since its distribution was well suited for the task. Next the sample must be split and sub-sampled into two 300 g batches for wet screening and dry screening as per the usual tailings screening procedure. A slow rotary splitting is a good method for evenly distributing the material in a representative manner, the McGill laboratory rotary splitter breaks a sample apart into 10 canisters, hence the choice for a 3 kg feed. The sample was split slowly over a period of 48 minutes, and diametrically-opposite canisters were selected as the charge for screening: the sub-sample weights were 306.5 g and 304.5 g indicating a good split, and the excess of 300 g was removed from each.

Once the wet and dry screening has been performed and the masses recorded the calibration exercise can begin. The calibration is a four step process: first the percent retained is used to obtain the actual screen apertures for the tailings screens; next the new apertures are used to recalculate the mass size distribution; after that the gold content per size class are adjusted using the new apertures; finally the adjusted grades can be calculated and the adjusted cumulative GRG distribution. The calculation steps will now be outlined in fuller detail; interested readers should consult Appendix A for a full tutorial on the various calculations used in the simplified test prior to reading this section.

Step 1) Actual Screen Apertures

The cumulative % passing is plotted versus the log of the aperture size, similar to that of the cumulative GRG plots portrayed above. Though the curves on a whole are far from linear, segments of the distribution can be approximated as log-linear between data points. The concentrate screening's % wt passing by size distribution is used as the reference point for obtaining the "actual" tailings screen apertures henceforth denoted x'. The concentrate apertures are denoted x_i where i represents the size class of interest. The linear equation describing the mass size distribution is of the form $y_i = m_i * \log(x_i) + b_i$, where y_i denotes the cumulative % passing, m_i is the slope of the line segment between the end data points of the line segment, x_i is the particle size of interest in μm, b_i is the y-intercept for the equation in %, and the subscript i is the size class range in which x resides (e.g. i = 0 for 600+ μm, i = 1 for 425+ μm to 600 μm, i = 2 for 300+ μm to -425 μm etc. consult the table from the example at the end of this section for a full listing). The cumulative % passing for the tailings screens are compared to those of the concentrate screens, one size class at a time starting with the coarsest size class (600+ μm) and working down to the finest size (-25 μm). Each tailings' y_i is checked to see what size range i it belongs to, and the corresponding "actual" size x'_i from the concentrate size distribution is read and calculated by inverting the logarithm to obtain the actual size in μm. This process is repeated for all size classes i to create a column matrix of x'_i values. These new aperture sizes correspond to the equivalent size classes the sample should have reported to had it been screened with the concentrate stack.

Step 2) Generate the new cumulative % weights y'_i using x'_i from step1

Now that the equivalent aperture sizes for the concentrate screens x'_is are known, the equivalent cumulative % passing y'_is can be read from the sample's cumulative % passing size distribution. First the line segments slopes m_i and intercepts b_i must be calculated for all size classes i. Then each y'_i can be calculated via the equation $y'_i - y_i = m_i * [\log(x'_i) - \log(x_i)]$ where y_i is the cumulative % passing for the smaller end of the size range (e.g. for i = 1, y_1 would correspond to the value at x = 425 μm); solving for y'_i yields the equation $y'_i = y_i + m_i * \log(x'_i/x_i)$. The y'_is are then converted from cumulative % passing to cumulative weight retained (g) using the screening data.

Step 3) Calculate the new cumulative % units of Au g'_i using x'_is

The units of gold per size class of the sample's tailings stream are first converted into cumulative % units Au per size class g_i. Next the line segments slopes m_i and intercepts b_i are calculated for all size classes. Similarly to step 2 above, the equivalent g'_is are calculated via $g'_i - g_i = m_i * [\log(x'_i) - \log(x_i)]$ where g_i is the cumulative % units Au for the smaller end of the size range; solving for g'_i yields the equation $g'_i = g_i + m_i * \log(x'_i/x_i)$. The g'_is are then converted from cumulative % units Au retained to cumulative units Au retained (g^2/t) using the total units Au of the sample's tailings stream.

Step 4) Calculate the calibrated GRG size distribution and feed grades

With both the adjusted mass and gold distributions, the last step that remains is to recalculate the size-by-size feed grades for the sample and ultimately the new GRG size distribution. Feed units of Au are calculated by adding the concentrate's units Au (which was the reference and hence unchanged) and the tailings' units Au (whose size distribution was adjusted to match the apertures of the concentrate screens). The feed weights by size class are calculated in the same manner using the concentrate and the new tailings mass distribution. The size-by-size feed gold distribution data can be calculated from the size-by-size feed units of Au and the total feed units Au. Lastly the total recovery of Au by size is calculated as usual by multiplying the concentrate's size-by-size recovery by the gold distribution size-by-size data, and ultimately cumulated to get the adjusted cumulative GRG size distribution.

2.6 Summary

The assaying problems posed in the previous section have been minimized in the standard GRG test by recovering the valuable material in a very small mass that is fully assayed (typical yields are less than 0.2%). The gold that is not recovered is labelled as non-GRG, which is distributed as small particles whose very fine size does not pose a sampling problem. Sequential recovery helps preserve the GRG size distribution, while minimizing assay issues.

GRG can come in various size distributions, from very coarse particles not fully liberated to very fine, fully-liberated particles. The wide swing in GRG particle shape, size and liberation can render some particles readily amenable to certain gravity recovery methods and others not, depending on the optimum range of the gravity-recovery device selected. Therefore upon gaining insight into the natural size distribution of an ore's GRG content, a gravity-recovery device of appropriate size, selectivity and efficiency must be chosen to achieve the desired level of grade/recovery. The following chapter reviews various gravity recovery devices applicable to the recovery of gold, and more importantly, GRG.

3 Gravity Recovery Devices

This chapter is intended to give the reader an overview of gravity recovery devices in general, both on an industrial and laboratory scale (where applicable). The primary gravity-recovery unit used during this research thesis is the Laboratory Knelson Concentrator, whose operation and GRG recovery mechanisms are explored in detail. The chapter begins with a description of some widely-used types of gravity units; the focus then shifts to plant-scale centrifuge units, followed by a detailed look at two laboratory gravity-recovery devices. Readers already familiar with these topics may wish to proceed directly to the next chapter in which the simplified GRG test procedure is introduced.

3.1 Typical Gravity Recovery Devices

Numerous types of gravity recovery devices have been developed throughout the history of mineral processing, yet despite the variations in throughput, size, shape and efficiency all share a common goal: to extract the mineral of interest from the gangue via exploitation of differences in their specific gravities.

3.1.1 Pans / Sluices

Panning is the simplest method of gravity recovery developed by mankind, as it only requires a pan, water and the ore to be separated. The action of shaking the ore along the pan in a controlled fashion, combined with the drag forces of water flowing out of the pan, can create a distinct separation provided the density differences between the target mineral and gangue are sufficiently large. In the case of gold, the process can operate well. However panning is a slow process yielding little concentrate and the results are rather operator-dependent. The development of increasingly sophisticated gravity recovery methods has long since rendered panning obsolete in industrial applications, but it is still widely practised by artisanal miners.

Sluices are inclined tables with riffled surfaces. Ore is slurried with water and poured down the inclined table surface: heavier particles, more prone to settling in the slurry flow, become trapped along the riffles whereas the lighter particles remain in suspension longer and are carried further downstream by the flowing water. This creates a vertical stratification down the inclined surface of the sluice, with lighter particles flowing further downstream and some ultimately discharged as tailings at the end of the table; heavier particles form the concentrate and remain deposited along the riffles, with the heaviest particles near the top riffles and the intermediately-liberated particles in the lower riffles. The lower end of the sluice where the tailings are discharged can be tapered to a certain shape: sluices with narrower discharge ends are known as pinched sluices, and are not riffled.

Sluices are still used extensively in alluvial mining, and can be effective when operated properly for placers with a gold content coarser than 100 μm.

3.1.2 Jigs

Jigs are one of the oldest methods of gravity recovery, yet their operational mechanisms remain one of the most difficult to fully explain. A jig is basically constructed of a water tank with two launders attached at different heights, one near the top, the other further down. At the bottom of the tank lies the pulsation device that generates the pulse and suction strokes of the jig. The mineral particles rest in the water atop a screen to prevent them from sinking to the bottom of the tank. The mineral bed is then projected upwards via a pulsation stroke, and as the particles begin to settle stratification occurs. The stratification mechanism is comprised of four parts, as seen in Figure 10.

The first mechanism is the differential acceleration of the particles in the mineral bed due to the pulse of water projecting them upwards. Lighter particles have less inertia to overcome and thus accelerate a little longer before achieving terminal velocity. Due to the constant influence of gravity the particles reach a maximum height at which point their upward velocity becomes zero, and they begin to fall at different initial velocities due to their density differences. Though the particles will reach the same terminal velocity, it is this differential initial velocity that produces stratification. Provided the pulp density is sufficiently high, hindered settling begins at this point where particles interact and displace one another as they fall together. The dilated bed has a natural tendency to rearrange its particles to minimize potential energy with the densest particles on the bottom of the settling bed. This natural tendency towards a lowest energy configuration is the third mechanism. Interstitial trickling is the fourth mechanism in which the fine particles nudge their way through the gaps between adjacent coarse particles to reach the lower levels of the bed.

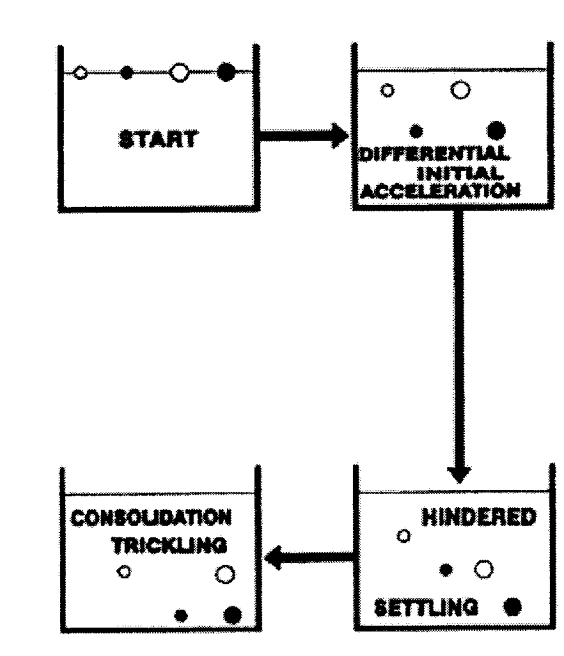


Figure 10: Jigging Process (Putz 1994)

The jigging process is a harmonic oscillation between pulsation and suction strokes, and the frequency and length of the stroke influence the performance. Pulp density is another important parameter and must be kept between roughly 30 to 50% solids. Lastly the feed rate and water addition must be constant to ensure stable performance.

3.1.3 Tables

Shaking tables are flowing-film concentrators, using a fine film of water to separate particles. They are constructed from a smooth inclined table surface; however the degree of inclination is not as severe as the sluice (a few degrees at most) since shaking tables usually treat a finer size distribution. Slurry is poured onto the shaking table, and an eccentric motor creates a harmonic oscillation to shake the table surface. The horizontal shaking, combined with the gravitational forces imposed from the degree of inclination, allows the shaking table to stratify material both horizontally and vertically across the table surface. Heavier particles tend to stratify less into a center band down the length of the table whereas intermediate and lighter particles will spread out further horizontally and wash further down the table surface. Different shapes and configurations of tables exist depending on the application and degree of separation desired. However their limited feed rate usually relegates them, for gold gravity circuits, to the domain of cleaning rather than primary recovery. Recently, there has been a significant shift to intensive cyanidation for this duty, which further marginalizes the use of tables for gold production.

3.1.4 Spirals

Spirals are helical film-type concentrators. The device earns its name from its resemblance to a coiled spring. The helical conduits are curved such that stratification of the particles fed to the spiral occurs both horizontally and vertically via different mechanisms, depicted in Figure 11. The spiral helix slopes downwards as it coils around, thus inducing a gravitational force on the particles which when combined with the slurrying water added at the top, causes the particles to flow down the coils. The three key design parameters of a spiral concentrator are its pitch, profile and radius. Each parameter has a direct effect of the recovery performance of the spiral. The pitch is the angle at which the coils tilt downward, and it affects how fast the slurry flows down the spiral. Steeper pitches are better for particles with large density differences; shallower angles are best for particles with minor density differences. Profile refers to the shape of the conduit cross section, which should be slightly curved to encourage heavies to roll back down toward the inner edge of the conduit and be collected in the trough. Radius controls the radial velocity and centrifugal forces within the conduit.

Vertical stratification occurs due to the mechanisms of hindered settling, interstitial trickling and Bagnold forces (Putz 1994). The combination of these forces causes a tendency for coarse light particles to remain on top of the slurry bed, followed by layers of fine light particles and coarse heavies, with fine dense particles resting on the bottom. Finer particles penetrate lower than their coarse counterparts due to interstitial trickling through the gaps between adjacent coarse particles.

Horizontal stratification is caused by velocity differences between the vertical layers that flow down the conduit. Since the vertical layers are comprised of different sized particles, they have slightly different average densities and hence flow at different rates. Centrifugal forces also cause the particles to shift towards the outer wall of the conduit, but the top layer particles can move more freely than the bottom layers and thus lighter particles become dispersed towards the exterior ring whilst denser particles remain near the inner wall. Depending on the difficulty of the separation a fourth design parameter, the number of turns on the helix, controls the retention time of the spiral; the greater the number of turns, the greater the residence time and the higher the chance of separation for minerals with minor density differences.

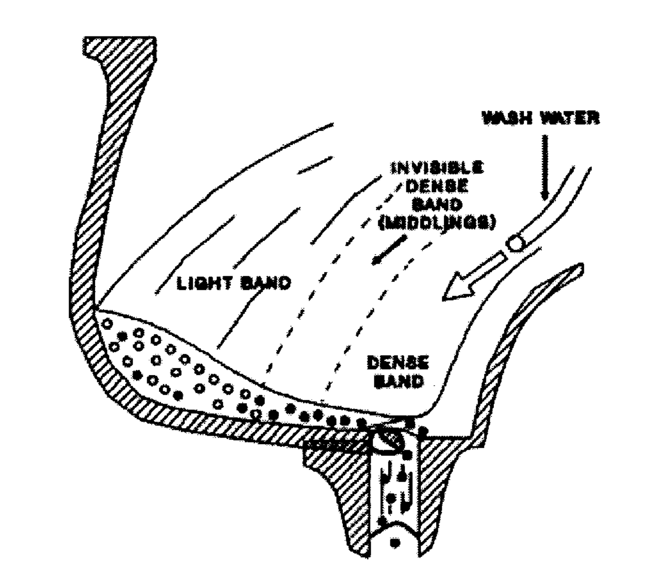


Figure 11: Spiral classification process (Putz 1994)

The spiral has several operational parameters that can be adjusted depending on the sample being processed: feed rate, feed density, yield and wash water flow rate (in the case of wash-water model spirals). Feed rate can be increased for separations with large density differences; those with smaller density differences require lower feed rates (i.e. increased retention times) to achieve an effective separation. Higher feed grades must be compensated with a lower feed rate as well. Wash water flow rate can be increased for large density range ores and should be lessened for close density range ores.

Spiral feeds must usually be diluted to prevent this problem, and as such are seldom the unit of choice for gravity recovery from circulating loads; spirals are typically used for GRG and gold-carriers recovery due to the gold high density and low abundance in ore (typical gold ore grades are less than 5 g/t).

3.1.5 Centrifuge Concentrators

Centrifuge concentrators fall under two categories: semi-continuous discharge units and continuous discharge units; in both cases the tailings are discharged continuously. Semi-continuous units are also occasionally referred to as batch units due to the fact that the concentrate is collected as one batch when the unit is shut down at the end of the processing cycle. However the time taken to flush the concentrate is so small when compared to the processing time of the centrifuge that the centrifuge can be operated nearly continuously (plant units can flush the concentrate within 60 to 120 seconds after a recovery cycle that typically extends between 30 and 120 minutes). Weight recovery to concentrate, however, is very low, typically less than 0.1%. Continuous centrifuges discharge both tailings and concentrate on a continuous basis while the centrifuge is operating. Centrifuge units have been developed for both plant-scale operation and laboratory investigations, some examples of which will now be discussed. This work will focus on semi-continuous units, which constitute more than 98% of all centrifuge applications in mining.

When compared to the longevity of gravity recovery on a whole, extensive industrial applications of centrifuge concentrators in mineral processing are relatively new. The earliest mineral processing centrifuges were patented in 1890; however they failed to become commercially applied due to the lack of wear-resistant materials and bearings (Falcon website,

http://www.concentrators.net/mining-and-fine-mineral-recovery/mining/history-of-concentrators.html).

By the late 1950s Russian and Chinese engineers began to produce workable centrifuges, but their use in western mining is virtually non-existent. The use of semi-continuous centrifuge units for gold recovery started in the early eighties and their increasing success has helped bring gravity recovery back to the forefront of industrial mineral processing over the last twenty-five years.

Semi-continuous centrifuge concentrators are constructed from a bowl mounted atop a spindle encased in a water-filled cylinder. The centrifugal force generated can be derived via Harris' Equation (1984) as:

$\mathbf{F_c = 4\pi^2 mn^2 r,}$ **Equation 1: Harris's Equation (1984)**

where F_c = centrifugal force, m = particle mass (g), n = rotational bowl speed (rpm), and r = bowl radius (m).

Typical G-forces exerted by centrifuge concentrators range in magnitude from 50 Gs to 200 Gs; generally, the finer the feed and the high-density mineral, the higher the Gs. Bowl shape, unit capacity, and motor type are all manufacturer-specific designs that will be listed for specific centrifuge units, to be mentioned later. Centrifuge concentrators require electrical power to drive the motor and large quantities of water to perform the separation.

The feed is introduced as slurry via a downcomer. The feed slurry strikes the bottom of the rotating concentrate bowl and the particles are immediately accelerated towards the inner wall of the concentrate bowl. More than 99% of the feed will be washed up the edges of the rotating bowl and discharged to the tailing launder at the top. Typically about 0.01% to 0.1% of the feed is retained in the grooves as concentrate. The very low mass recovery limits the use of these units to the recovery of very high-density (typically a s.g. of 10 or more), very low grade minerals –i.e. gold alloys and platinum group minerals.

Centrifuge units recover well from primary grinding circuits and can process various streams as shown in Figure 12. Since centrifuge units operate with slurried feeds, cyclone underflows make a logical selection to feed centrifuges and can help reduce the large water consumption required for operation. Gold grinding circuits tend to build up a significant circulating load, assaying as much as a few thousand g/t Au; treatment of a small bleed stream of the circulating load with a centrifuge concentrator can reduce the circulating load significantly and produce an primary gold concentrate for further upgrading.

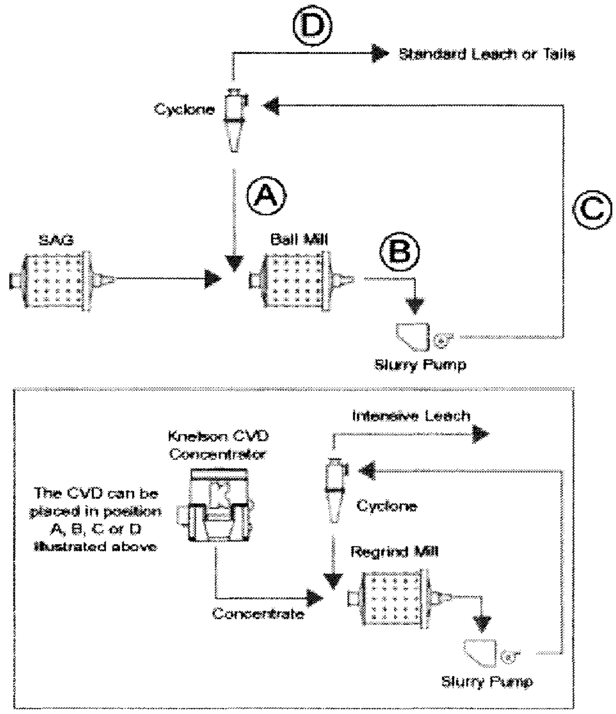


Figure 12: Centrifuge locations in grinding circuits

3.2 *Plant-scale Centrifuges*

3.2.1 Semi-continuous (Batch) Knelson

The Knelson semi-continuous concentrator series use concentrate cones with multiple rings to optimize fluidization water flow in each ring during unit operation. Though the concentrate can only be collected at the end of the unit's operation, production scale units can flush the concentrate down the launder via a patented multi-port hub in less than two minutes; this small discharge time, when compared to the typical 1-2 hours a unit will operate, is the reason why this series of Knelson centrifuges is referred to as semi-continuous rather than batch.

Knelson's semi-continuous product series comes under three categories: an extended duty series (KC-XD) for heavy industrial applications, a center discharge series (KC-CD), and a manual discharge series (KC-MD) for laboratory testing. Unit size can range from the small lab-scale KC-MD3 (solids capacity 0-45 kg/hr, process water requirement 0.7-4.0 Lpm) to the large KC-XD70 (solids capacity 300-650 tph, process water requirement 1134-2079 Lpm). Though Knelson also manufactures variable speed centrifuges (XD series), the gravitational forces generated vary between 30 and 90 Gs. (Knelson website,

http://www.knelsongravitysolutions.com/page152.htm).

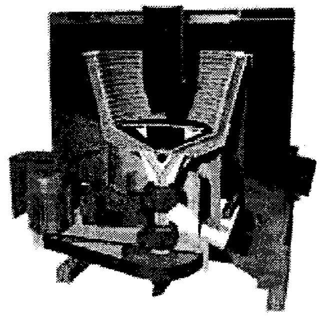


Figure 13: Semi-continuous Knelson

3.2.2 Continuous Variable Discharge Concentrator (Knelson)

The continuous discharge Knelson utilizes a series of actuated pinch valves to bleed off the concentrate at a controlled rate down a launder while the tailing stream is continuously ejected via a separate launder. Knelson manufactures designs with 1 or 2 rings for concentrate collection; the rings contain a series of fluidization water holes. Continuous discharge models range in size from the lab-scale KC-CVD6 (solids capacity 0.5-2.0 tph, process water requirement 18.9-45.4 Lpm) to the KC-CVD42 (solids capacity 40-100 tph, process water requirement 265-605 Lpm). Continuous Knelsons generate between 30 and 90 Gs of force (Knelson website). Actual industrial applications are few.

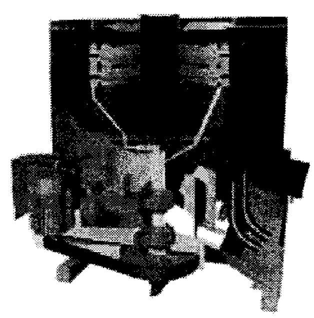


Figure 14: Continuous Variable Discharge Knelson

3.2.3 Falcon SB Series Semi-Continuous Concentrator

The Falcon semi-continuous SB operation is essentially identical to its Knelson counterpart. The main differences between the Falcon and Knelson centrifuge concentrators are their rotation speeds and their bowl designs: the Falcon concentrate bowl does not possess ridges. Falcon's SB series models range in size from the lab scale SB40 (solids capacity 0-0.25 tph, process water requirement 3.8-18.9 Lpm) to the large production scale SB5200 (solids capacity 105-330 tph, process water requirement 567-680 Lpm). The G-force range generated by this series can vary from 50 to 200 Gs (Falcon website:
http://www.concentrators.net/index.php?option=com_content&task=view&id=12&Itemid=135)

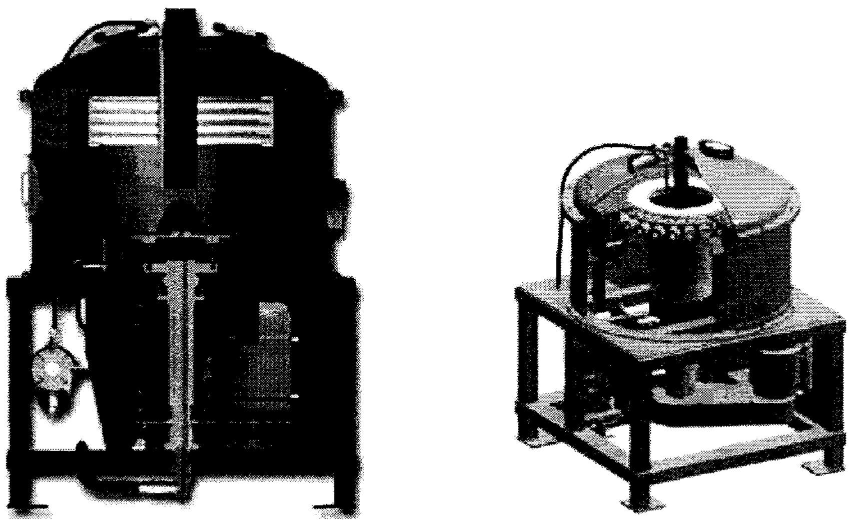


Figure 15: Falcon SB Series Concentrator (left), C Series Continuous Concentrator (right)

3.2.4 Falcon C Series Continuous Concentrator

The Falcon C series concentrators are continuous discharge units suitable for fine feed applications, the maximum particle size recommended is 1 mm. This series of units generates gravitational forces ranging from 50 to 300 Gs, allowing feed stream particles to become segregated along the smooth spinning rotor wall. The heavier particles are bled off continuously via a set of plane mass flow hoppers and patented throttling nozzles. Size ranges vary from the C400 (solids capacity 1-4.5 tph) to the C4000 (45-100 tph). The Falcon C series requires no additional process water (Falcon website). Actual industrial applications are few.

3.3 Laboratory Mozley Separator

The Mozley Laboratory Mineral Separator (henceforth denoted MLS) is strictly a device used to sort material via the mechanism of flowing film concentration. The MLS is constructed from a slightly sloped separating tray driven by an electric motor to slide back and forth horizontally to create a harmonic shaking action on the particles on its surface. A thin film of water is constantly washed down the surface of the tray to fluidize the particle bed and allow the particles to flow down to the tailing launder. Particles slide down the tray due to a slight downward tilting of the table (0 to 4 degrees, depending on the difficulty of the separation) and the film of water that allows the particles to overcome friction.

Separation occurs via horizontal stratification during the periodic shaking as the particles with higher density settle quicker and thus spread out less whereas the lighter particles spread out further and wash down the tray faster. This leads to a separation of the minerals into "bands" according to their respective densities. In the case of gold the gold band will be rather easy to distinguish visually from the rest of the material, but is generally very small, owing to the low feed mass. Once separation has occurred the unit is stopped and the unwanted bands washed down as the tailing product. The concentrate is then washed into a separate container. The MLS can yield very efficient separations and recover very fine gold particles, even below 100 μm. However the process is very time consuming, the maximum mass that can be treated in one cycle is 150 g (50 g yields an optimum separation), and the data returned are rather noisy since the separation efficiency is operator-dependent. For gold, the noise comes from the small mass treated. Efficiency can be increased by treating narrowly sized feeds, which is typical for the Mozley. These limitations restrict the range of applications for the MLS, even at laboratory scale.

3.4 Laboratory Knelson Concentrator

Though the Knelson KC-MD3 centrifugal concentrator was the primary centrifuge used for the laboratory research, the results presented in this thesis are not Knelson-specific. This is because most laboratory applications of the Knelson are aimed at characterizing ore samples, rather than predicting actual performance. This is achieved by operating the unit to maximize its recovery, rather than its production rate.

3.4.1 Construction and operating principles

The Laboratory Knelson Concentrator MD3 (henceforth referred to as LKC) is a centrifugal separation device designed to recover very dense minerals (s.g. > 9) from lighter ones (s.g. < 6). The LKC's construction is a high speed, ribbed, rotary cone powered by a drive unit. A basic diagram of the LKC is shown below, but the main innovation behind the Knelson's success is its ribbed concentrate bowl where recovery occurs.

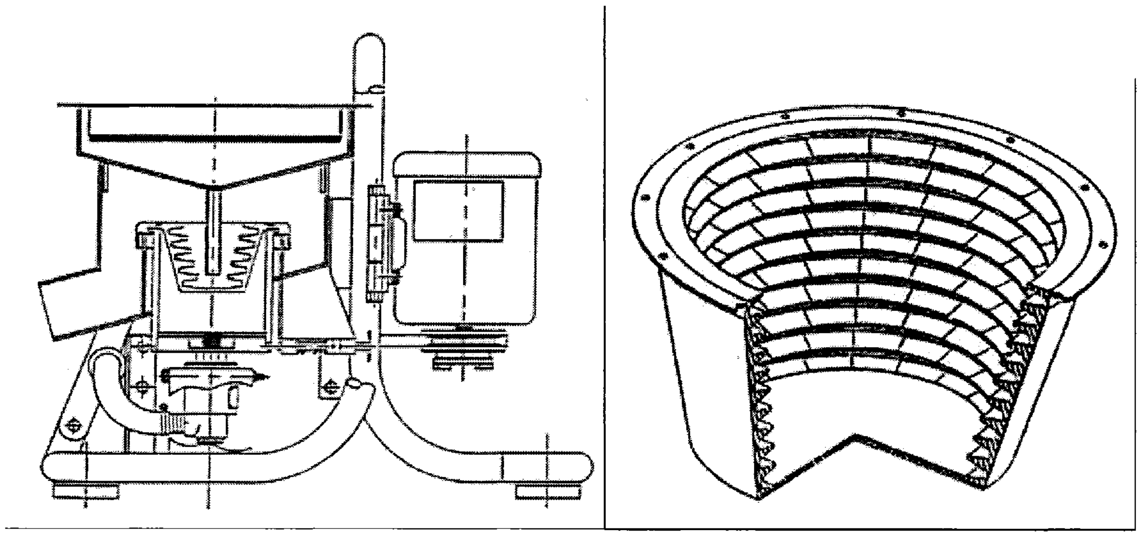


Figure 16: Knelson Design (Left), Example of a Concentrate Bowl (right)

The drive unit causes the cone to rotate at 1700 rpm to generate 60Gs of force. The LKC operates on the principles of hindered settling and centrifugal force. Slurry is fed at 20 to 70% solids into the bottom of the bowl, and centrifugal forces cause the feed to fill the ribs from the bottom up. The centrifugal force generated in the KC bed is derived from Equation 1. Heavy particles are forced against the walls and trapped between the ribs whereas lighter particles are swept to the top of the unit by the water and ejected against the outer wall of the unit (Putz 1994).

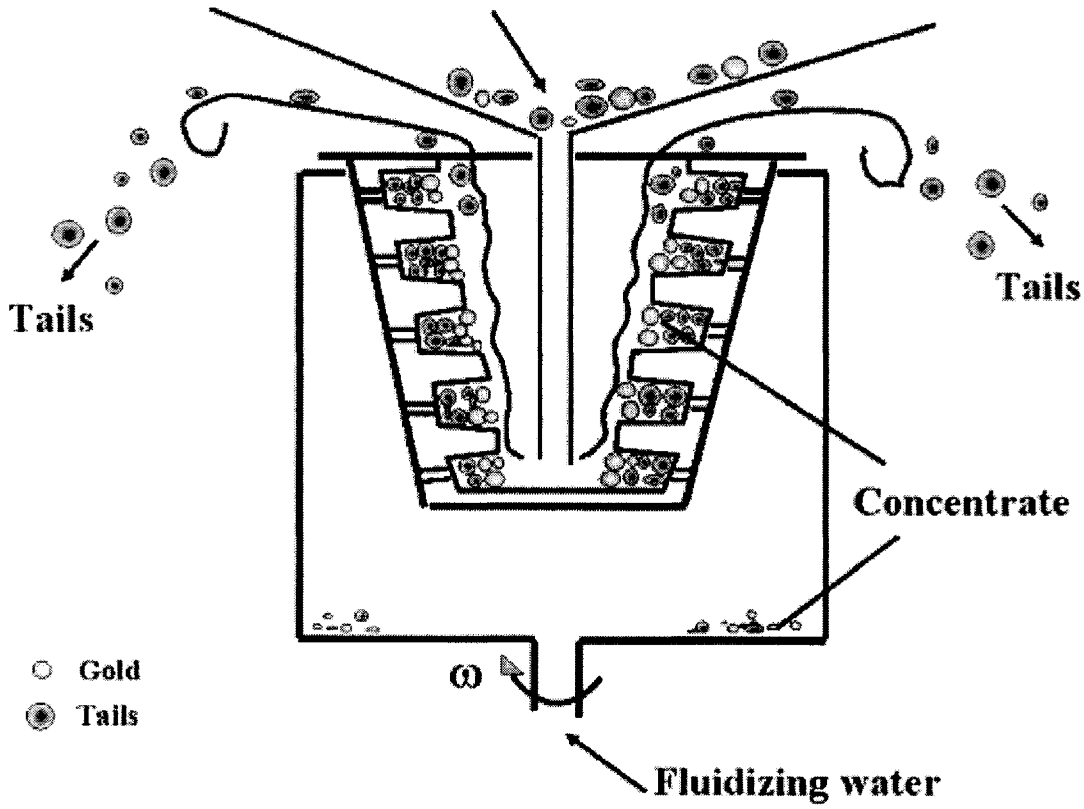


Figure 17: LKC in operation (adapted from Laplante, 2003)

The concentrate bowl is surrounded by a pressurized water jacket that forces water through the holes in the cone's rings to keep the bed of particles fluidized at the surface of the grooves (the material inside the grooves is not fluidized and serves as a dispersing medium for fluidizing water). The conflicting drag and centrifugal forces minimizes compaction of the particles accumulating in the grooves, otherwise known as the bed.

3.4.2 The three phases of Knelson recovery

Three distinctly observable phases of material are collected during the Knelson recovery cycle, and can be summarized as follows:

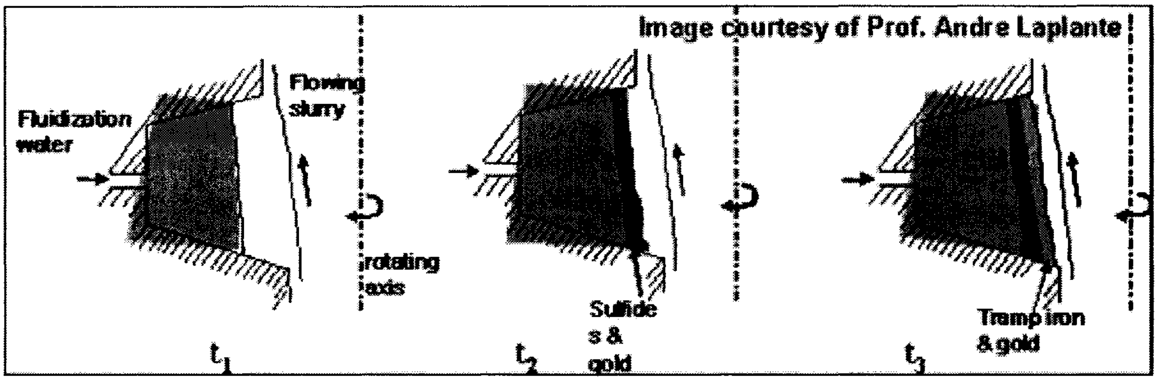


Figure 18: Three-phase LKC recovery cycle (Laplante, 2000)

In the first phase (start-up phase) the empty concentrate bowl is filled in a matter of seconds indiscriminately with gangue and dense minerals alike. The second phase is characterized by significant gold-carrier recovery, resulting in the formation of a secondary layer of concentrate (the thickness of this layer is not to scale); it is relatively short in duration in industrial applications, but can be more significant at bench scale (Laplante 2003). In the third phase, which constitutes the bulk of the recovery cycle in industrial applications, very high-density particles are recovered; most of these are typically tramp iron, with minor amounts of gold and/or platinum group minerals.

Clean fluidization water must be used to prevent blockage of the pores in the concentrate bowl (Putz 1994). Higher fluidization flow rates are necessary for coarser size distributions since larger particles are harder to fluidize. Optimum backwater pressure also increases as the specific gangue density increases (Ounpuu, 1992). Shear forces created by the fluidization water favour the KC for the recovery of fine dense particles; Banisi (1990) states that the mobility of the particles within the bed causes light particles to be continuously replaced by incoming heavier ones until the heaviest particles in the feed are retained. This phenomenon is now known to take place at the surface of the concentrate bed. Fine dense particle trickling is favouring by the dispersive forces created by the shear rate induced by the bowl rotation. This phenomenon is known as the **Bagnold effect** (1954). Density allows coarse dense particles to force their way into the concentrate bed via displacement, finer particles pass through the flowing slurry via interstitial trickling, and particles of intermediate size can experience a lower trickling rate than even finer particles.

3.4.3 Improvements in bowl design

Earlier bowl models were cylindrical, but then the G forces were not uniform throughout the bowl (47 Gs at bottom, 59 Gs at top). Gravitational force also varied widely within each ring (from 47 to 68 G). Using conical bowls resolved these issues, and wedge profiles were added to reduce variation within grooves (Banisi, 1990). These various improvements in design have helped to bolster the reliability of the LKC's performance.

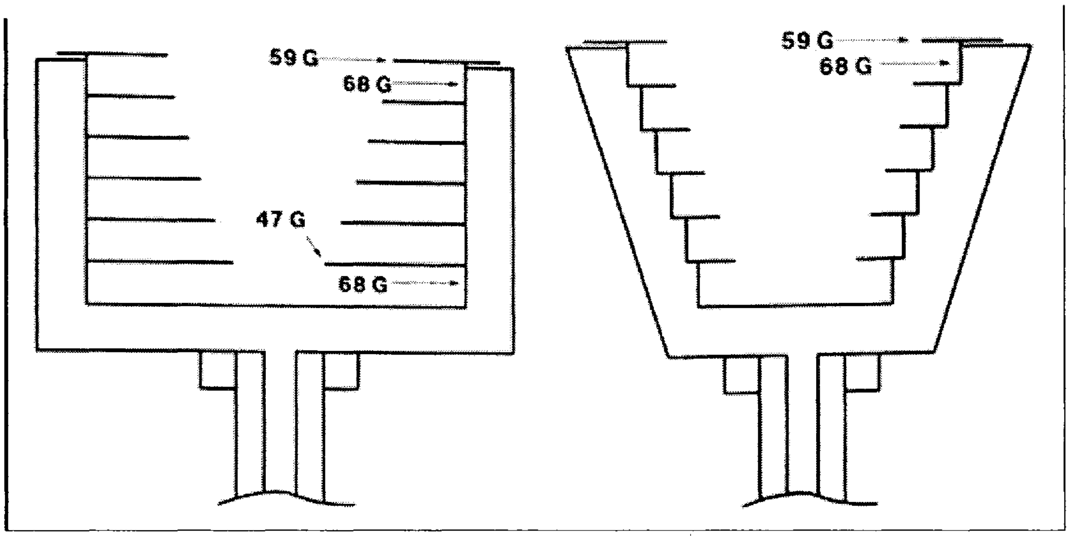


Figure 19: Cylindrical versus Conical Bowl Designs

3.4.4 Reliability of the Laboratory Knelson Concentrator

Previous investigative work has been conducted to assess the ability of the LKC to correctly measure then amount of GRG. One such study (Banisi 1990) compared the size-by-size and total amounts of gold recovered in the lab amalgamation tailings and LKC tailings, and found that amalgamation tailings grades were only slightly lower than the LKC's. The tests were performed on COF and CUF samples. While the two types of tests returned similar gold contents, the type of gold being recovered was theorized to be quite different: amalgamation only recovers fully liberated, clean and/or flat gold flakes, whereas the LKC will recover gold particles of intermediate liberation. Therefore the gold gravity potential reported from the amalgamation results may not necessarily be an accurate predictor of plant gravity recovery since most particles recovered via gravity are not fully liberated.

Banisi also compared the LKC to the MLS on primary and secondary cyclone overflow samples. The comparison made between the two was imperfect since gold particles found in overflows are generally either very fine or flaky, and this is the type of material for which the LKC is known to have the poorest performance. Therefore it was expected that the MLS achieved better recoveries for these samples; however, Banisi noted that the LKC would have been the superior candidate had the feed sample been what is typically fed to gravity recovery units (i.e. cyclone underflows, mill discharges, etc.). Another limitation of this work was the use of much higher yields for the MLS, which resulted in the recovery of gold associated with sulphides.

A second study of amalgamation versus the LKC (Putz 1994) confirms that the two methods return very similar gold grades for all size classes measured (150 μm down to –38 μm), as well as similar gold distributions. However the size-by-size recoveries differed greatly in their respective tailings as well as the total free gold content of the tailings. Putz concurs with Banisi that these differences can be attributed to the type of gold particles being recovered. These comparisons, along with the large amount of research performed with the LKC over the last 15 years (Banisi 1990, Putz 1994, Woodcock 1994, Huang 1996, Xiao 1999) establish the LKC as an efficient and reliable predictor of GRG content, whose results serve as an upper threshold for plant-scale KC performance. Any centrifuge concentrator could have been used and achieved similar results provided sufficient feed and fluidization flow rates were observed.

3.5 Summary

This chapter outlined the types of gravity recovery devices available, along with their potential applications and/or drawbacks. In plants where gravity recovery is a viable option, it is important to size the gravity units accordingly to optimize performance while minimizing capital costs. Further, a prediction or estimate of gravity recovery must be generated to assess its potential economic impact. In order to select the appropriate recovery device and size, and predict circuit performance, the GRG content and size distribution of the ore to be processed must be known. This need for detailed size-by-size information has led to the development of a stratified sampling process using the LKC: the standard GRG test.

It should be noted that the GRG test characterizes an ore rather than a recovery unit, therefore results can be applied to all of the equipment presented in this chapter. While centrifuges are the recovery unit used in this research, the results presented are not centrifuge-specific, and different equipment will operate on different size ranges of GRG with varying performance. The following chapter will now describe the GRG test protocol in greater detail.

4 The Simplified GRG Test

4.1 Introduction to the Standard GRG Test

The standard GRG test was first developed at McGill University in 1990. It is a three-stage GRG ore characterization test performed at progressively finer size distributions in order to give the LKC the best recovery possible at gold's natural size distribution. The test conditions in the lab test are set to be as conducive to maximum LKC recovery as possible; as such the GRG content returned from the test results usually represent the maximum gravity recovery possible; actual plant recovery will usually be somewhat lower, typically one third to two-thirds of the GRG content. Figure 20 is a schematic flow diagram of the standard GRG test:

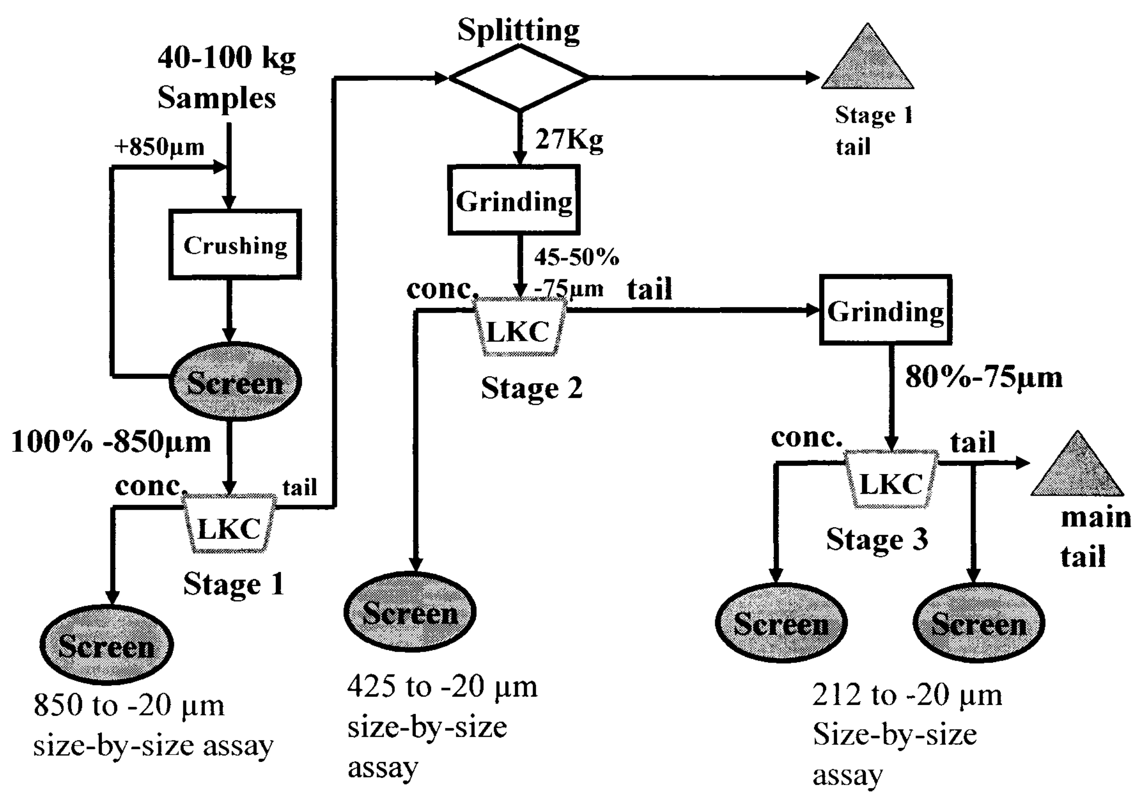


Figure 20: The standard GRG test (Laplante, 2003)

In order for the test results to be accurate, the sample used must be as representative as possible of the original ore body from which it was extracted. Ideal samples for GRG tests are either drill core rejects or feed samples composited over a long period of time, typically a month. Given the potential for sampling errors previously mentioned in chapter 2, at least 50 kg of ore is necessary in order to have an acceptable level of statistical confidence in the test's results, unless head grade is particularly high. Stage 1 entails crushing the ore sample to 100% passing 850 μm so that the GRG can be recovered efficiently by the LKC. The entire concentrate is screened down to 20 μm, a 600g sub-sample of the tailings is screened down to 20 μm and 27 kg of the tailings are sub-sampled for the next stage. Stage 2 consists of grinding a 27-kg sub-sample of the stage 1 tailings to 45-55% passing 75 μm and processing with the LKC. The concentrate and tailings sub-samples are extracted and screened as per stage 1, and the remaining tailings, typically around 24 kg, are passed on to the final stage. Stage 3 grinds the ore to 80% passing 75 μm prior to processing a final time through the LKC. The concentrate and tailings sub-sampled are screened and assayed in the same fashion as stages 1 and 2. Operating conditions are adjusted as feed coarseness decreases from stages 1 to 3, from 1000-1200 g/min to 270-350 g/min. Corresponding fluidization water flows range from 7.0 L/min (stage 1) to 5.0 L/min (stage 3).

The resulting size-classified mass and assay data make it possible to calculate size-by-size gold recoveries and the overall GRG content. A powerful tool for such a mass balance is an Excel worksheet for each stage, followed by a worksheet to calculate the overall gold and GRG balance. An explanation of the calculation process is provided in Appendix A.

Due to the numerous stages involved the standard GRG test can at times be a lengthy procedure, which may render its overall cost prohibitive to potential clients due to the number of man-hours, assays, and large sample size required. Whilst this may not be the case when a single test is required for circuit design, a simpler test becomes attractive for benchmarking many samples and for routine testing. Early investigations performed by Woodcock (1994) attempted to discern whether three stages were redundant and a shorter test used in its stead. A few tests were conducted but no concrete conclusions were reached at that time. This thesis is a continuation of that work; an introduction to the two simplified GRG tests developed will now be given.

4.2 The two Simplified GRG Tests

Two simplified tests were used in the research work of this thesis, a two-stage and a one-stage test. The two-stage test begins with stage 1 as per the standard test, but the tailings are bypassed directly to the stage-3 size distribution. The one-stage test grinds the ore sample directly to the stage-3 size distribution prior to one round of processing with the LKC. Both tests use 20 kg ore samples rather than the 50-100 kg of the standard test. Their respective flowsheets are shown in Figure 21 and Figure 22.

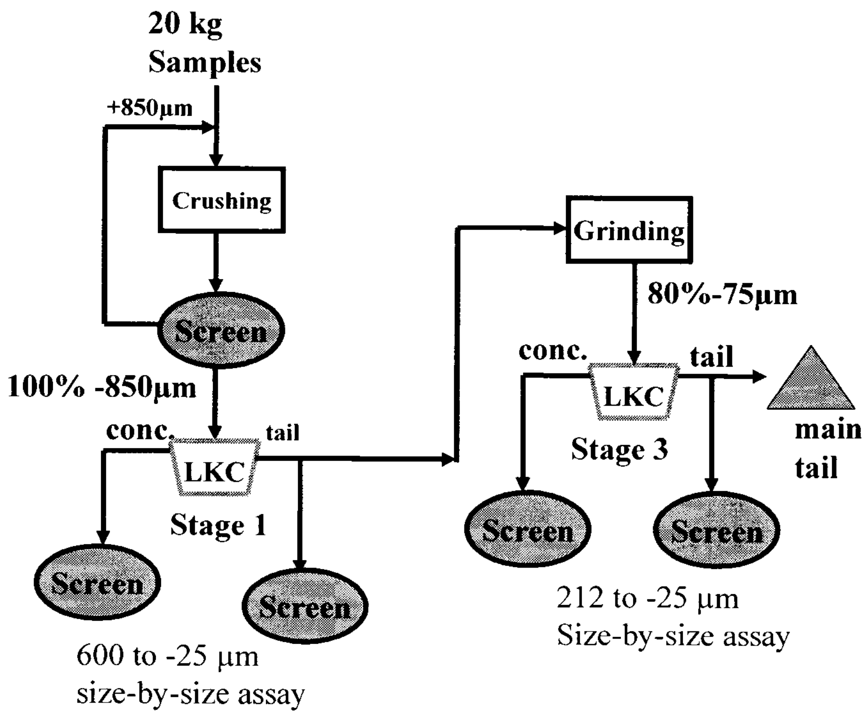


Figure 21: The two-stage GRG test

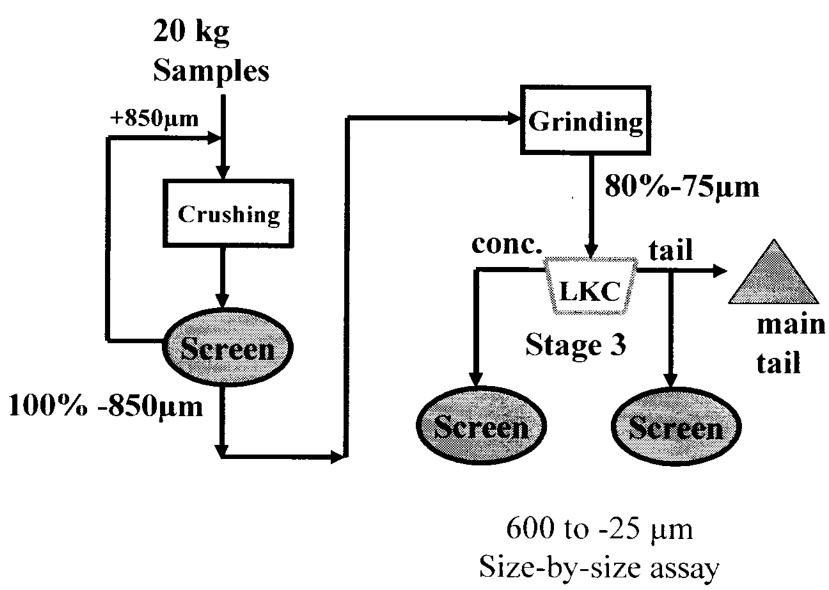


Figure 22: The one-stage GRG test

4.3 Experimental procedure

The experimental procedure consists of several steps derived from the flowsheets depicted previously. The major stages involved in the simplified GRG test are the following:

i) Crushing/Screening
ii) Grinding/Screening
iii) Separation via the LKC
iv) Dewatering/Drying
v) Concentrate Preparation/Screening
vi) Tailings Sub-sampling
vii) Tailings Wet & Dry Screening
viii) Tailings preparation for second stage

i) Crushing

The LKC has been found to operate better when the size distribution is below 850 μm (Woodcock, 1994). Crushing is usually done in a laboratory cone crusher, provided the feed particles are smaller than 1.25cm in diameter (if not, a jaw crusher is used to obtain the desired size distribution). The crusher product is passed over a vibrating screen at 850 μm. The undersize is removed from the crushing circuit and the oversize recycled to the cone crusher. This process is repeated as often as necessary until approximately 1 kg of oversize remains, at which point this remaining oversized material is pulverized for one to two minutes and screened again.

A modest grade discrepancy between one-stage and two-stage tests run on the same ore sample was observed in the first few simple tests (most notably the GU series in Chapter 4), even though they were split from the same source. Due to the low mass used for the simplified GRG tests, it is crucial that each sample processed be representative of the ore from which it was extracted; when a sample in excess of the 20 kg is provided (e.g. 40 kg for two tests), a one-pass sub-sampling of the desired mass cannot guarantee the homogeneity of the sub-sample. Extensive mixing must be done prior to the splitting step in order to generate splits that are as identical as possible to the original; therefore an improved mixing protocol was implemented for all samples processed after the Guyana Quartzite (GU2) series.

The protocol modification was simple: samples to be split are poured into the riffler and split and recombined repeatedly. In the case where adequate sample mass is provided for both the standard and simplified tests, this usually leads to four to six 15 kg pails of material which are grouped into pairs of left-side and right-side splits. In order to minimize bias, the mixing step uses a left-side split with a right-side split from a different pair. This process is repeated for each of the remaining left and right pails. By choosing a pail from a different pair's split blending is occurring, and the more times this process is repeated, the more thorough the blending. Given time constraints, usually two rounds of left-side and right-side split mixing is enough to deem the sample adequately homogenous. No additional large grade discrepancies have been observed for the rest of the simple GRG tests processed in this manner.

ii) Grinding

In the case of the two-stage simple test, the grinding step is deferred until after the product of step 1 has been processed with the LKC once (stage 1). However the 1-stage test requires that the crusher product be ground directly to the stage-3 size distribution of 80% passing 75μm before processing. Samples are ground using laboratory rod mills. The mills are typically fed with 2 kg batches of material per grind. For this work, the 20 kg sample is broken up into 9 equal parts (usually 2240 g) to complete grinding in three rounds using three 40 cm length by 20 cm inside diameter rod mills, each with a charge of ~20 rods varying between 0.8 cm and 2.5 cm in diameter. The initial grind time, which varies between 60 and 180 minutes, must be inferred from how the sample responded to crushing. Each mill product from the first round is matted and a sub-sample extracted using a spatula on equidistant points of the matte. The three sub-samples are then mixed, matted and sub-sampled down to 100 g in a tared lab pan. This 100 g sample is then wet-screened at 75 μm, the undersize discarded, and the oversize dried and weighed. The mass of the oversize is then used to calculate the grind time of the second round.

If the first round's size distribution is within ±10% of the target (80% -75 μm) the sample is retained as is and the remaining two grind times are adjusted to compensate so that the average of 80% is reached; if the sample is too coarse (below 70% passing 75 μm) it must be put aside and reground to target. If the sample grind time was severely overestimated and the grind product too fine (above 90% passing), the remaining batches are underground slightly from target, followed by extensive riffling to blend the mismatched material as evenly as possible. Similar adjustments are done with the second and third batches of grinds if they are still off target.

iii) LKC

The third step entails processing the sample with the 3" Laboratory Knelson Concentrator (LKC). The sample is tared on the large scale and then scooped/poured into the LKC feed hopper in close proximity to a dust hood. Two to three tailings barrels (each approximately 150 L capacity placed on rollers for easy transport) are positioned near the LKC, one directly underneath the LKC tailings tube for tailings collection. Next the fluidization water hose is connected and the LKC is switched on. The fluidization water valve is opened halfway before the water is turned on in order to prevent water build-up and possible damage to the LKC. Water flow is set to maximum for reproducibility purposes (the LKC fluidization valve then can be set to a known value of 17.2 kPa (2.5 psi), which yields a fluidization flow rate of 5 L/min). Slurrying water flow rate is adjusted to approximately 2 to 2.5 L/min. Actual flow rates are then measured using a tared bucket and stopwatch recorded over an interval of 30 seconds.

The sample feed rate is calibrated to the appropriate target (1000 g/min and 7 L/min fluidization for stage 1, 300-350 g/min and 5 L/min for stage 3). The vibratory feeder is allowed to run for 2 to 3 minutes to build a steady feed stream at an initial vibratory setting, and then adjusted to the desired rate, using measured feed rates over 1-minute.

A stopwatch is used to record the duration of the LKC processing time (time starts when the feed first falls into the LKC and time is stopped when the vibratory feeder is empty), to confirm the actual feed rate after the test is complete. Since the tailings discharge rate varies between 7 to 9 L/min on average, the tailings barrels must be changed every 18 to 25 minutes. The feed in the hopper must be stirred semi-continuously (every 2 minutes) to prevent the solids from "rat-holing" and to keep the feed rate as constant as possible. The downcomer seldom plugs unless the slurrying water rate is not high enough or the vibratory feeder discharge is incorrectly positioned over the slurrying water jets. At the end of the test, the vibratory feeder is turned off, as is the slurrying water flow, the fluidization flow is halved, and finally the remaining fluidization flow and the LKC are simultaneously shut down. This procedure minimizes the risk of accidental concentrate ejection at the end of the test.

Concentrate is collected by disassembling the LKC and removing the concentrate bowl, the contents of which are washed out into a pan. The concentrate bowl is then further washed by dunking it vigorously in three water bowls to dislodge any remaining solids. Typically the first bowl wash contains most of the remaining concentrate and the second wash the rest, the third should be clear. Any material remaining in the concentrate bowl holder or the inner walls of the LKC is also treated as concentrate and is thus washed into a bucket, decanted and added to the concentrate pan and placed in the oven to dry. The tailings barrels are wheeled aside near a floor drain and the solids are allowed to settle for a minimum of 3 hours before the dewatering phase can begin.

iv) Dewatering

The excess water is siphoned out until the discharge becomes slightly cloudy. The tailings bed is then scooped into large oven pans, labelled and placed in the oven to dry at approximately 80 to 95°C. The remaining tailings are washed into buckets and pressure filtered, labelled and dried; each tailing barrel is siphoned and washed out in this manner.

v) Concentrate Preparation/Screening

The concentrate, once dried and cooled, is weighed and spread onto a piece of matting paper. A hand magnet is used to remove tramp iron particles present in the concentrate. These are weighed, examined and recombined with the concentrate if smeared gold is detected or if magnetic minerals are present. A designated stack of 8" Tyler screens, ranging in aperture from 600 μm down to 25 μm, is brush-cleaned and optically inspected for tears using a magnifying lens before each screening. The concentrate is poured in and the stack placed in a Ro-Tap machine for 20 minutes. Sample bags (4 oz. capacity) are labelled with a code for each size class. The concentrate is weighed and bagged per size class. All gold assays were performed at Laboratoire Bourlamaque, Val d'Or, QC.

The fire assaying method used to ascertain the gold grades of the concentrate and tailings samples screened in the simplified GRG test imposes minimum mass constraints for accurate assays. In the case of concentrate the minimum mass required for each size class is approximately 0.2 g whereas tailings require at least 5 to 10 g. These are the guidelines used for the third stage of the standard test, and at first it was decided to apply them to the 1-stage test. However, gold's slow grinding kinetics normally means that the minimum assay requirements (in terms of gold mass) can be met at a coarser size class during stage-1. The original protocol required the concentrates to be cumulated down to the coarsest size class that matches the minimal mass requirements of the tailings, causing a loss of GRG information at coarser size classes. This procedure limits the range of the GRG size distribution graph for the simplified test, making it harder to visually assess its trending of the standard test results.

The solution to this problem was simple: coarser concentrate size fractions are now assayed even if the there is not enough mass in the corresponding tailing size fraction. The selection of the coarsest size fraction for tailing assays remains unchanged. The modified protocol does not alter any of the grades for finer size classes but makes it possible to extend the GRG size distribution at coarse size, typically to 300 to 425 μm, rather than the original 150 μm to 212 μm.

vi) Tailings Sub-Sampling

Typically 18 of the original 20 kg remains as tailings after the dewatering/drying stage; the cooled tailings cakes are broken down manually over the vibrating sieve (850 μm setting). When a continuous screen is used, the oversize discharge is obstructed to achieve a sufficient retention time on the screen. A tared pail is placed at both the undersize and the oversize discharge points. The screen is operated in the presence of two dust hoods (one over the undersize discharge, the other over the screen itself). Once the cake has been fully broken, the oversize barrier is removed and the oversize collected; the screen is then shut down and cleaned.

Riffling is performed using a lab-scale Jones riffle and several tared riffling pans. In order to prevent riffling bias the sample is fed slowly back and forth from one end of the riffler to the other. One side pan is retained for further sub-sampling; the other pan's contents are returned to storage. The selection of which side is discarded (left or right) is alternated each pass to further minimize the chance of sub-sampling bias. This process continues until approximately 600 g (ideally no more than 650 g) remains in one pan. The pans contents are then matted and scooped into two equal portions of 300 g with a spatula.

vii) Tailings Wet & Dry Screening

A vibrating wet-screening deck is placed over a fitted pail to collect the undersize material. A filter paper is labelled and its weight recorded, then inserted into the pressure filter to await the undersize slurry. The first 300 g tailings sample is wetted in the pan and the screen wetted. The screen is then inserted into the vibrating deck, the apparatus turned on, and the tailings are gradually poured into the screen. The wet-screening should be done in four intervals, washing approximately a quarter of the solids into the screen at a time, washing the oversize several times and draining thoroughly before more solids are added. By adding the solids in this gradual fashion, most of the undersize material (minus 25 μm) will be washed through the deck, and there should be very little undersize left in the pan after dry screening. The oversize material is labelled and placed in the oven to dry. The same process is repeated for the second 300 g sample. Finally the undersize slurry is pressure filtered and dried. The two oversize products are dry screened using the procedure described above for the concentrate.

Each screen oversize is weighed and bagged individually similar to the concentrate screening procedure, the only modification being that a minimum of 2 g is needed in the coarsest bag to be assayed (typically this occurs by the 150 μm screen for the one-stage test). The second round of oversize screening is identical to the first, and each size class is weighed and added to the previous round's matching size class (the purpose of the repeat is to check for consistency and to generate enough assaying mass). Finally the pan contents are mixed with the wet screen undersize and sub-sampled with a spatula. Since the assay lab typically requires no more than 30 g for tailing samples, any tailings size class that contains more than 40g is sub-sampled on matting paper to the target of 30g, and the excesses are saved individually. For the minus 25 μm size class, which is assayed twice, approximately 60 g is sub-sampled for assaying. Any samples coarser than 150 μm and in excess of 5 g are pulverized.

viii) Tailings Preparation for the next stage

The one-stage test requires no further tailings handling, and the tailing product is retained in a labelled bucket until assays are available and checked for discrepancies. The sample can then be discarded or stored for further investigation if so desired. In the case of the two-stage test, the sample is split into the nine batches for grinding and the process repeated starting at step 2.

4.4 Marginal-GRG Assessment

Occasionally gold particles are liberated or near-liberated, but because of particle size or shape, cannot be recovered at the rotation velocity of the regular GRG tests (i.e. a theoretical 60 Gs). This gold, when recovered at high rotation velocities and very low yield, can be defined as *marginal GRG.* The LKC has a limited ability to recover fine GRG below 25 μm effectively; this problem becomes less significant with finer feeds, and previous experiments conducted by Woodcock (1994), Huang (1996) and Rowland show that at for a feed P_{80} of 75 μm, the problem is generally limited to the minus 20 μm fraction. A few samples were chosen during this research project for further investigation into their marginal-GRG content. The tailings from simple GRG tests were fed to a laboratory Falcon SB40 at a controlled rate to determine the amount and size distribution of marginal-GRG left behind.

The laboratory Falcon's operation is virtually identical to the LKC, but generates in excess of 180 Gs of centrifugal force. This additional force increases the capability of the Falcon to recover very fine dense particles, such as fine marginal-GRG, and possibly incompletely liberated gold particles of relatively fine size. The concentrate can provide additional insight into the nature of the marginal-GRG recovered, whilst the calculated feed grade and size distribution of the Falcon test can be compared to the tailings data of the parent test to verify the integrity of the sample. Results from the Falcon experiments shall also be presented in the following chapter.

5 Test Results and Discussion

The results in this chapter are presented in chronological order of sample processing. A full explanation on the procedure used to calculate and plot GRG content is provided in Appendix A. The complete data for each test sample can be found in the same order in Appendix B. The samples have been given generic labels to preserve the anonymity of the providers; for the sake of continuity with the upcoming publications of this research (Laplante and Clarke, 2006), the same sample abbreviations have been used.

5.1 Nevada Au-Cu (NV)

The first sample examined was a coarse, high-grade sample of known high GRG content. This sample was a test run for the simplified GRG test procedure, and as such the sample consisted of the leftover stage 1 tailings of the standard GRG test. The standard test had returned an overall GRG content of 91%, with 82.2% of the GRG content recovered in stage 1 alone. The simplified sample was ground to the stage 3 target size distribution under standard test conditions with an average grind time of 120 minutes per batch, the results of which can be seen in Table 3 and Figure 23 below.

Particle Size (μm)	STD stage 1 Cum Rec. (%)	STD overall Cum Rec. (%)	2-Stg overall Cum Rec. (%)
600+	43.9	44.22	43.38
425+	56.6	56.98	55.90
300+	63.42	64.14	62.64
212+	66.11	67.08	65.30
150+	68.41	69.87	67.57
106+	71.42	73.45	72.09
75+	75.81	78.73	77.39
53+	78.30	82.27	80.83
38+	80.03	85.13	83.80
25+	81.03	88.10	86.33
Pan	81.64	91.04	90.55

Table 3: NV Cumulative Recoveries

Due to the large GRG recovery in stage 1, there is very little GRG left for recovery in stages 2 and 3, therefore the GRG distributions in Figure 23 below hardly differ. Similarly the grades are in good accord: the standard test returned a head feed grade of 11.6 g/t and the simplified test yielded 11.8 g/t, resulting in a relative difference of less than 2%. Figure 24 indicates that the standard test recovered more fine marginal-

GRG than the 1-stage test; in this case the increased recovery can be attributed to the extra stage of recovery in the standard test, allowing it to recover additional fines with which the LKC is known to have difficulty.

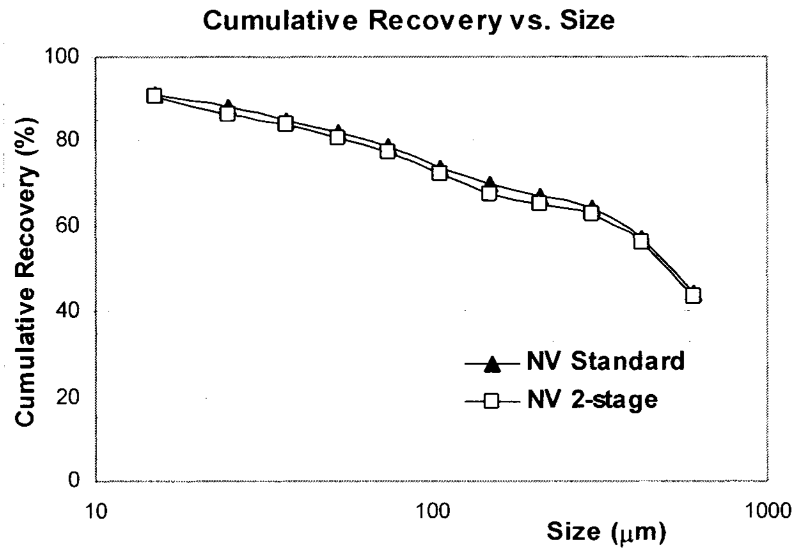


Figure 23: NV GRG size distribution

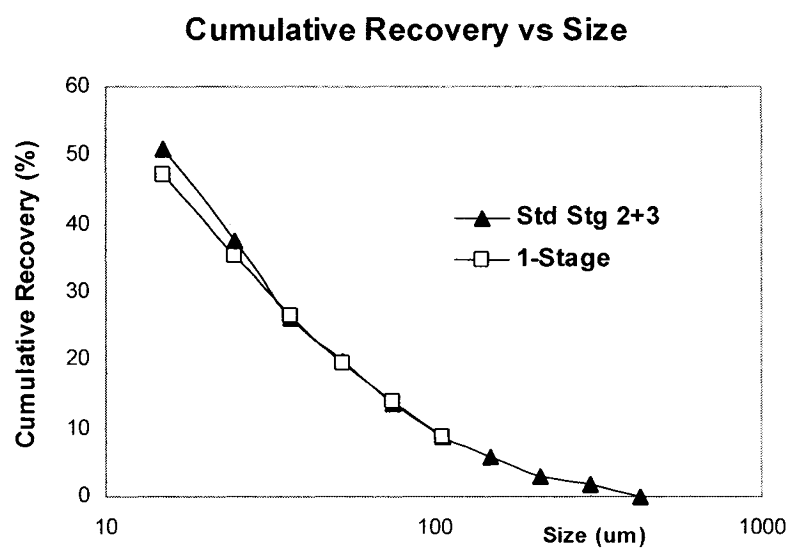


Figure 24: NV stage-by-stage recovery

5.2 Quebec Cu-Au I (QC1)

The first Quebec Cu-Au sample (henceforth referred to as QC1) consisted of 22.3 kg processed via the two-stage test and 22.15 kg via the one-stage test, both under normal operating conditions. The sample was softer than most, with an average grinding time of 65 minutes per batch to reach stage 3 size distribution (the average grind time for most samples studied was between 90 and 120 minutes per batch). The summary of results is displayed in Table 4:

Particle Size (μm)	Standard Cum Rec. (%)	Corrected Std. Cum Rec. (%)	2-Stage overall Cum Rec. (%)	1-Stage overall Cum Rec. (%)
600+	0.13	0.22	0.24	0.00
425+	0.14	0.50	0.54	0.00
300+	0.70	5.37	5.38	0.00
212+	1.88	8.65	8.07	0.00
150+	4.19	12.62	12.41	0.00
106+	7.79	16.74	16.47	9.72
75+	11.23	21.38	22.64	14.83
53+	14.67	25.43	25.73	19.52
38+	18.94	29.30	29.56	20.43
25+	23.68	34.43	34.66	30.33
Pan	35.62	45.10	42.37	40.10

Table 4: QC1 Cumulative Recoveries

Stage 1 of the standard test returned a recovery of only 1.7%, which contradicted results of the two-stage simplified test. This was also in stark contrast to recoveries obtained in stages 2 and 3, 19.4% and 14.5%, respectively. The head feed grades returned by the three tests further supported the hypothesis that stage 1 was in error since the overall head feed grades were in accord: the two-stage and one-stage tests were 0.52 g/t and 0.46 g/t respectively versus the standard's 0.48 g/t. The cause of the problem was not identified; furthermore, the head grade of the sample was well below the expected grade of 1 g/t; it was therefore decided to process a second, more representative sample. This repeat test, Quebec Cu-Au II, will be described in section 5.3.

Figure 25 shows that the GRG size distribution of the one-stage test is finer than that of the two-stage test, although the overall GRG content is somewhat similar. This could be explained by the additional grinding GRG recovered in the first stage of the two-stage test is subjected to for the one-stage test.

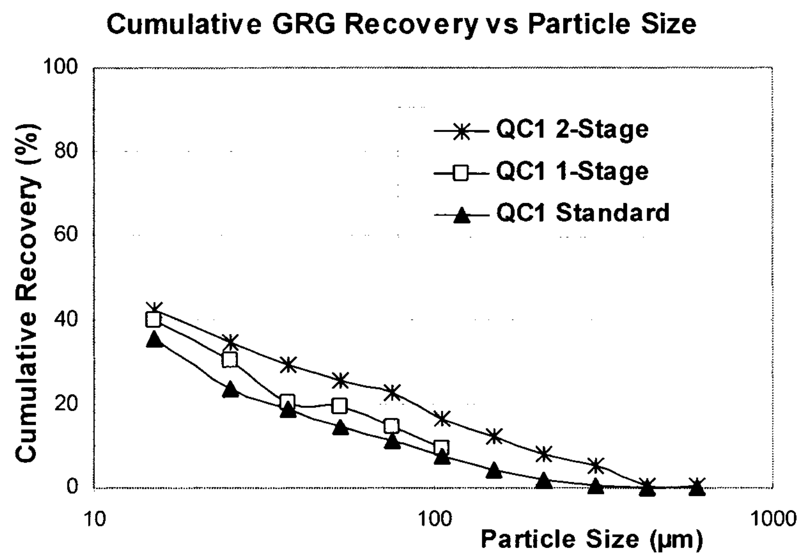


Figure 25: QC1 GRG size distribution

5.3 Quebec Cu-Au II (QC2)

A full series of repeat tests (standard, two-stage and one-stage) were performed on a second Quebec Cu-Au sample (QC2) supplied from the same ore body but sampled in the pit more carefully. Theoretically the results returned should have been identical to its QC1 predecessor if the sample was truly representative and processed correctly. The results shown in Table 5 and Figure 26 clearly demonstrate that this was not the case.

Particle Size (µm)	Standard Cum Rec. (%)	2-Stage overall Cum Rec. (%)	1-Stage overall Cum Rec. (%)
600+	0.21	0.80	0.00
425+	1.03	1.27	0.00
300+	3.83	2.29	0.00
212+	7.56	4.74	0.00
150+	12.51	10.12	8.69
106+	18.67	17.42	15.01
75+	24.68	25.24	23.68
53+	30.30	31.00	28.44
38+	35.92	38.44	36.45
25+	41.50	44.00	42.56
Pan	52.85	56.50	55.48

Table 5: QC2 Cumulative Recoveries

The head feed grades returned for the tests were 1.09 g/t, 0.83 g/t and 0.80 g/t for the standard, two-stage and one-stage tests, respectively. These grades are in line with typical head grades for the mill, although the difference between the head grade of the standard test and that of the two simple tests is high. The cumulative GRG size distributions are in good accord as shown in Figure 26.

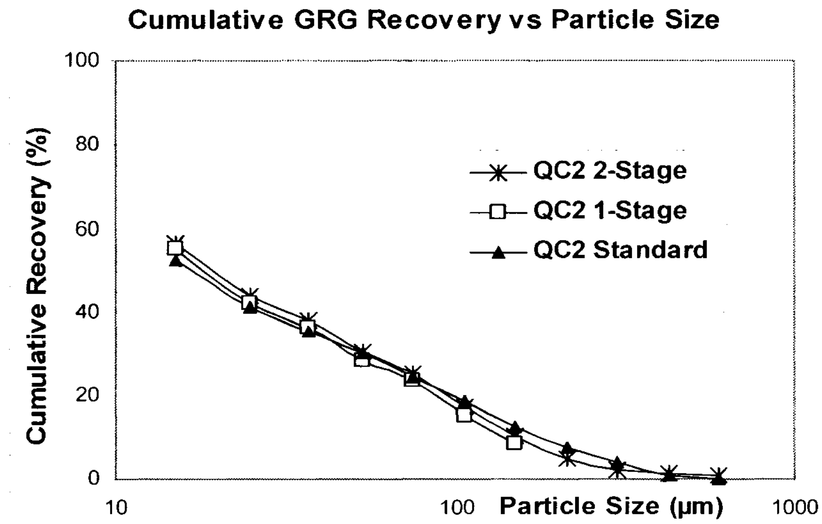


Figure 26: QC2 GRG size distribution

The higher head grade of the second sample, which is in line with the estimated grade of the pit, confirms pit sampling can be considerably more challenging than drill core or finely crushed ore sampling. Figure 27 compares GRG of the two QC samples. Results suggest that for this ore, GRG content is correlated with head grade.

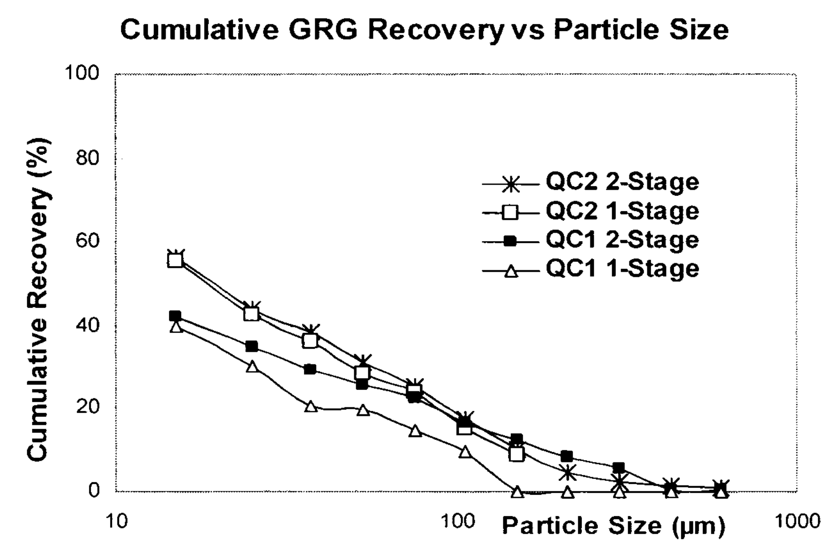


Figure 27: QC tests comparison

Figure 26 illustrates that the one-stage simple test would be a very suitable replacement for the full test for this ore, as it estimates well both the GRG content and its size distribution.

5.4 Guyana Saprolite (GU1)

The Guyanese Saprolite sample was weathered alluvial tailings at a naturally fine size distribution of approximately 74% passing 75 μm before processing. This caused modifications to both the standard and simplified GRG test procedures for this particular sample: the standard test was shortened to two stages, the first at the natural size distribution, followed by stage 3 at 90% passing 75 μm. Since the standard test had become a two-stage test, a two-stage simplified test would have proved redundant; therefore two one-stage tests were conducted at a target size distribution of 90% passing 75 μm but at a different fluidization water flow rate in order to study its effect on Knelson recovery. The one-stage test at a low fluidization water rate (2.9 Lpm) was dubbed test A, and the other at high fluidization rate (5 Lpm) test B.

Particle Size (μm)	Standard Cum Rec. (%)	1-Stage test A Cum Rec. (%)	1-Stage test B Cum Rec. (%)
600+	4.70	0.00	0.00
425+	8.73	0.00	0.00
300+	12.10	0.00	0.00
212+	13.95	9.64	0.00
150+	17.74	15.47	20.49
106+	23.08	22.48	25.84
75+	27.74	34.22	37.72
53+	32.01	39.55	42.63
38+	35.70	43.84	46.59
25+	40.93	46.65	49.23
Pan	48.04	52.48	54.12

Table 6: GU1 Cumulative Recoveries

The weathered saprolite ground easily; the average grinding time to reach 90% passing 75 μm was under 40 minutes. The head grade of the three tests is in agreement as well at 1.01 g/t for the standard test, 0.90 g/t for one-stage test A, and 0.95 g/t for one-stage test B. No appreciable differences in GRG recovery were observed between tests A, 52.5% and B, 54.1%, thus demonstrating that fluidization water has very little effect on the laboratory Knelson's performance provided a minimum threshold is obeyed to safeguard the slurry flowing in the bowl from compaction. If anything, the slightly higher recovery of Test B validates the original fluidization flow chosen for this stage, 5 L/min. The standard test returned a slightly lower recovery, 48%.

The GRG distribution shown in Figure 28 illustrates that the GRG is almost evenly distributed amongst the size classes for the standard test. The two one-stage simple tests show the same increase in GRG fineness observable only above 53 µm observed for its Quartzite sister-sample. GRG information at sizes coarser than 150 µm for test A and 106 µm for test B is lost, as the small mass reporting to the concentrate and tailing of the these coarser size fractions made assaying unreliable. Recognizing that the mass of gold present in the concentrate fractions may be significant, it was decided, for subsequent tests, to assay these fractions provided mass was sufficient, as described in Chapter 4, section 4.3-v.

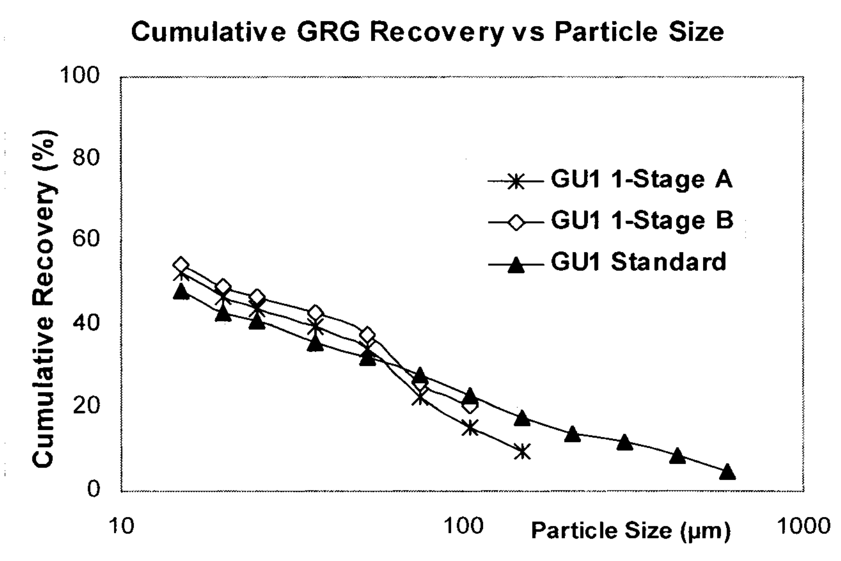


Figure 28: GU1 GRG size distribution

5.5 *Guyana Quartzite (GU2)*

The second sample investigated was a Guyanese Quartzite. Sufficient mass was provided for both the two-stage and one-stage test (20 kg for the two-stage, 18.9 kg for the one-stage). The initial size distribution of the crushed sample was fairly coarse at 35% passing 75 µm. The sample ground with an average grinding time of 110 minutes per batch to reach stage-3 size distribution. The standard test had yielded a GRG content of 80.4% in a 2.85 g/t feed. The two simple tests returned a very similar GRG content, 83.3% for the two-stage test and 80.6% for the one-stage test.

Particle Size (μm)	Standard Cum Rec. (%)	2-Stage overall Cum Rec. (%)	1-Stage overall Cum Rec. (%)
600+	1.76	1.97	0.00
425+	3.68	6.16	0.00
300+	7.20	13.10	0.00
212+	12.15	18.66	0.00
150+	19.01	23.05	0.00
106+	25.86	31.75	15.60
75+	33.27	39.05	22.02
53+	41.23	46.48	47.98
38+	46.86	55.25	55.79
25+	59.95	63.76	64.48
Pan	80.44	83.32	80.58

Table 7: GU2 Cumulative Recoveries

The head feed grades for the standard, two-stage and one-stage tests were 2.85 g/t, 2.79 g/t and 3.42 g/t respectively. The one-stage head grade differs from the others, but apparently this has no impact on the GRG content. The difference in head grade could be attributed to inadequate sample mixing after grinding, and changes were made to the simplified GRG test procedure to correct for this in future samples; these changes were addressed in Chapter 4, section 4.3-i. All subsequent samples were processed using the improved feed preparation protocol.

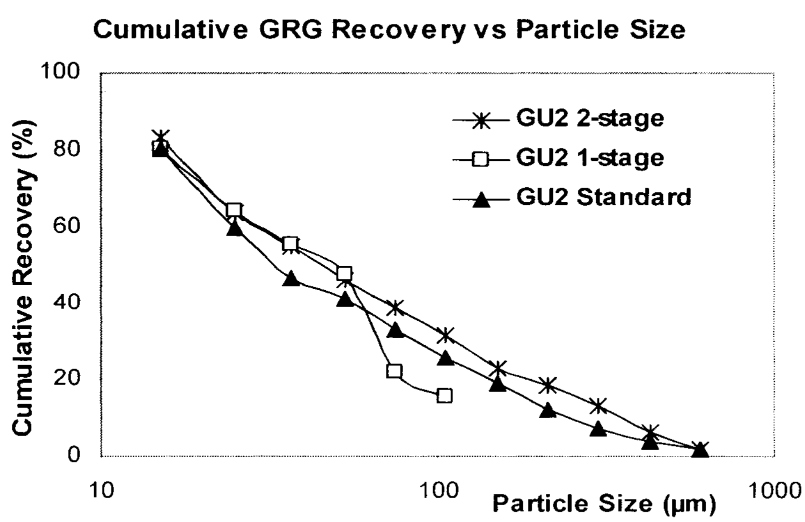


Figure 29: GU2 size distribution

Figure 29 compares the size distribution of the GRG for the three tests. Between 53 µm and 75 µm, there is a sharp step-like increase for the 1-stage test's GRG distribution, also observed in the Guyanese Saprolite sister-sample GU1. This implies that the additional grinding in the one-stage test only affects its GRG size distribution above 53 µm.

5.6 Mexican Sulphide (ME)

The Mexican sample was a massive sulphide copper-lead-zinc volcanogenic ore, assaying 29.5% Fe, 6.4% Zn, 1.8% Pb, and 0.3% Cu. The ore body also contains 294 g/t Ag and 2.5 g/t Au for precious metals recovery (Espinosa-Gomez, 2005). Forty kilograms of material was provided for the two simplified GRG tests. The samples averaged a grind time of 90 minutes per batch, a yielded an unusually high mass of concentrate due to the higher proportion of sulphides present (concentrates weighed between 150 g to 190 g per stage, as compared to the average 80-120 g for most samples). The various test results can be seen in the following table:

Particle Size (µm)	Standard Cum Rec. (%)	2-Stage overall Cum Rec. (%)	1-Stage overall Cum Rec. (%)
600+	0.56	1.23	0.18
425+	0.76	2.05	0.56
300+	2.09	3.19	1.43
212+	6.64	6.92	3.26
150+	13.82	12.92	7.20
106+	21.97	21.46	15.79
75+	28.94	29.99	24.77
53+	35.85	37.94	33.16
38+	39.98	42.95	38.38
25+	43.02	45.38	40.99
Pan	43.59	47.12	42.79

Table 8: ME Cumulative Recoveries

The head feed grades returned are in good accord at 2.89 g/t for the standard, 2.94 g/t for the two-stage and 2.91 g/t for the one-stage test. The GRG size distributions exhibited an excellent correlation as well, as shown in Figure 30. The two-stage test slightly overestimates GRG content. The new concentrate assaying protocol (Chapter 4, section 4.3-v) successfully restored the full GRG curve, which confirms that some additional grinding of GRG and loss of the coarser component takes place above 53 µm.

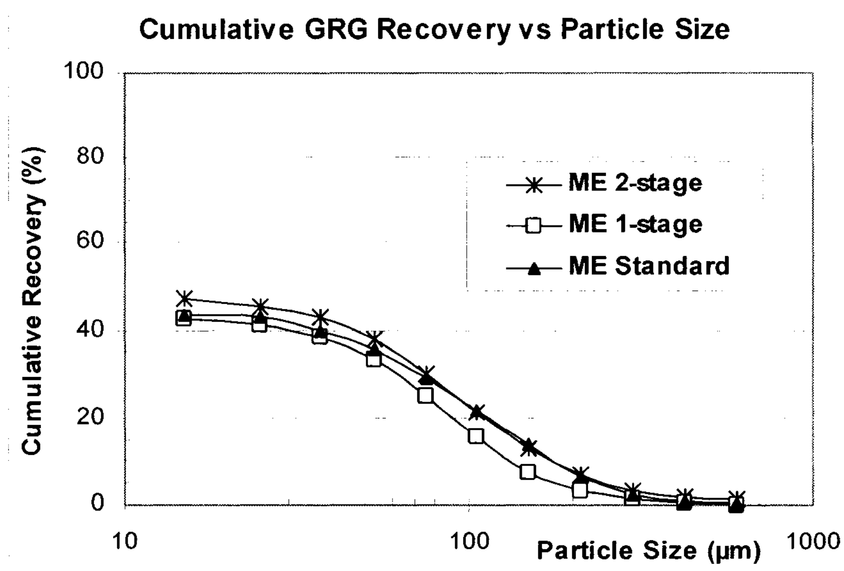


Figure 30: ME GRG size distribution

5.7 Ghanaian Composite (GH)

Four African ore samples were processed with the standard GRG test, named Kenyase-East (KE4), Kenyase Central (KC4), Teekyere (T39) and Bosumkese (B41). Their head feed grades ranged between 1 and 4 g/t, and the samples were hard, with a mean grind time of 120 minutes on average. A summary of their standard and simple GRG test results are listed in Table 9 and Table 10 respectively.

Particle Size (µm)	Standard KE4 Cum Rec. (%)	Standard KC4 Cum Rec. (%)	Standard T39 Cum Rec. (%)	Standard B41 Cum Rec. (%)
600+	0.21	0.46	0.14	0.29
425+	0.45	1.03	0.34	0.73
300+	0.98	2.66	1.50	1.77
212+	2.13	5.53	3.16	3.52
150+	4.18	11.52	5.13	6.77
106+	6.23	16.61	7.62	10.27
75+	7.96	21.26	10.12	13.44
53+	9.78	25.09	13.03	16.60
38+	11.23	26.99	15.88	19.31
25+	13.20	29.09	19.36	22.54
Pan	20.08	35.19	28.56	32.60

Table 9: GH Standard Cumulative Recoveries

Unfortunately insufficient mass (<11 kg) was available to perform simplified tests on each sample, therefore a composite sample was created using equal 10 kg remnants of each ore type after crushing. This 40 kg composite sample was then mixed thoroughly via repeated riffling and split into two identical 20 kg samples to be processed with the two simplified GRG tests. The goal of this experiment was to see if the simplified tests would report a GRG content equal to the mathematical average of the GRG contents reported from the four standard GRG tests.

Particle Size (μm)	Ave. Standard Cum Rec. (%)	2-Stage Cum Rec. (%)	1-Stage Cum Rec. (%)
600+	0.29	0.45	0.00
425+	0.68	1.09	0.00
300+	1.91	1.88	0.00
212+	3.97	2.84	0.03
150+	7.67	3.57	0.35
106+	11.23	5.77	2.36
75+	14.54	10.35	6.32
53+	17.67	13.77	9.81
38+	19.94	16.23	12.18
25+	22.63	18.40	14.15
Pan	30.42	25.76	20.60
Grade (g/t)	2.70	2.77	2.86

Table 10: GH Simple Cumulative Recoveries

The calculated head grade of the overall sample, 2.70 g/t, compared well with that of the two simple tests, 2.77 g/t for two-stage test and 2.86 g/t for the one stage-test. GRG content, however, was considerably lower for the simple tests, 25.8% for the two-stage test and 20.6% for the one-stage test, compared with the 30.4% of the standard test. The relative drop in GRG content is 15.3% for the two-stage test and 32.3% for the one-stage test. A comparison of the GRG size distributions is provided in Figure 31.

The hard and abrasive nature of the GH ores has transformed the GRG particles into non-GRG during the grinding process. The coarser GRG has disappeared, being ground into either finer GRG or non-GRG. Since no GRG had been detected in the stage 1 concentrate fractions of the standard test (the plus 150 μm fractions are upgraded and examined using an optical microscope), the nature of the GRG remained uncertain. A further investigation was made using 12 kg of stage 1 tailings from the standard GRG test (the highest-grade KC4 sample): the sample was ground to stage 3 size distribution and reprocessed with the LKC. The concentrate was screened into separate size classes and the intermediate size classes (106-150 μm, 75-106 μm, and 53-75 μm) were each separated on a MLS and inspected visually for gold, but none was found.

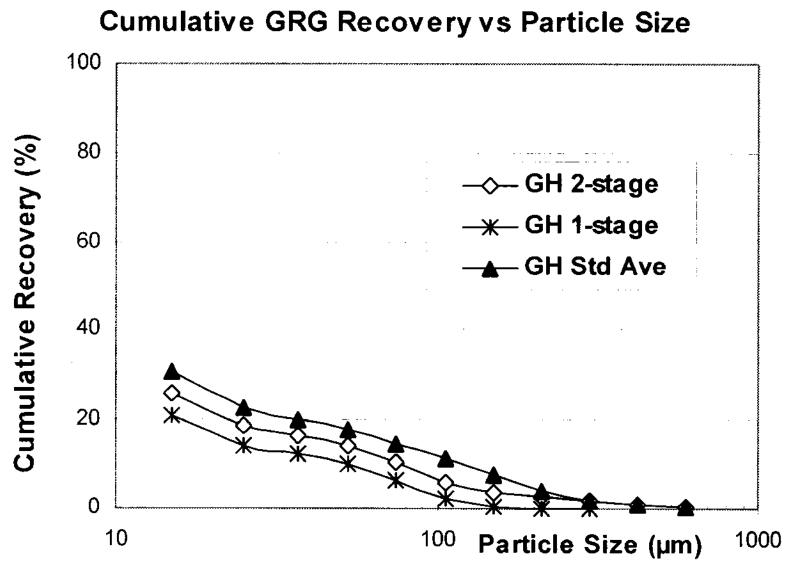


Figure 31: GH GRG size distributions

5.8 *Western Australia Cu-Au series (AU1-AU3)*

The Western Australia Cu-Au test series (AU) was comprised of three different ore types: a southern diorite (AU1), a central diorite average (AU2), and a central diorite lower (AU3) sample. These samples were but three of several zones of a massive low-grade gold-copper bedrock source with an average grade of 0.9 g/t. The gold is associated with silver as electrum and native bismuth. Chalcopyrite is one of the primary sulphides present, along with pyrrhotite and pyrite to lesser extents (mineralogy provided by the client). These copper-gold samples were low-grade samples ranging from 0.7 g/t to 1.4 g/t. Twenty kilograms of each ore type were available for processing with the one-stage simplified test. Average grinding times ranged from 80 minutes to 100 minutes per batch.

Table 11 and Table 12 show the head grade and GRG content of the standard and one-stage tests for the three ore samples. Agreement in head grade is only fair, despite using the modified blending and splitting protocol. This may identify a limit of the simple tests: at these low grades, 20 kg may be an inadequate mass to split for adequate sample representativity. The reported GRG contents returned by the simplified tests were lower than that of the standard by 10% for the AU1 and AU2 samples, but for the AU3 sample the simple test GRG content was 5% higher than that of the standard. The cumulative GRG size distributions are displayed in Figure 32, Figure 33, and Figure 34.

Particle Size (μm)	Standard AU1 Cum Rec. (%)	Standard AU2 Cum Rec. (%)	Standard AU3 Cum Rec. (%)
600+	0.26	0.15	0.17
425+	0.96	0.32	1.20
300+	3.18	0.78	1.64
212+	4.75	1.66	2.92
150+	7.80	3.13	4.30
106+	11.45	5.94	5.99
75+	15.70	9.38	7.35
53+	21.82	13.81	9.85
38+	27.51	18.28	12.40
25+	34.90	23.64	15.96
Pan	50.58	36.20	25.05
Grade (g/t)	1.45	0.68	0.70

Table 11: AU Series Standard Cumulative Recoveries

Particle Size (μm)	1-Stage AU1 Cum Rec. (%)	1-Stage AU2 Cum Rec. (%)	1-Stage AU3 Cum Rec. (%)
300+	0	0	0
212+	2.69	0.01	4.49
150+	4.72	1.01	5.58
106+	7.48	2.46	7.47
75+	11.31	5.13	9.48
53+	15.92	8.36	11.68
38+	21.22	12.00	14.93
25+	25.80	15.33	18.39
Pan	40.13	25.41	30.28
Grade (g/t)	1.24	0.78	0.55

Table 12: AU Series Simple Cumulative Recoveries

Though the GRG curves in the three figures track their respective standard GRG distributions, samples AU1 and AU2 clearly demonstrate a drop in GRG content. In both cases the 1-Stage GRG curves diverge slightly from the original, and the GRG is not recovered by the final size class (minus 25 μm). Sample AU3 does not seem to suffer the same problem as the simple 1-Stage test appeared to have slightly higher coarse GRG content (212 μm), which carried through to the finest size classes, creating a slight over-prediction in cumulative GRG content.

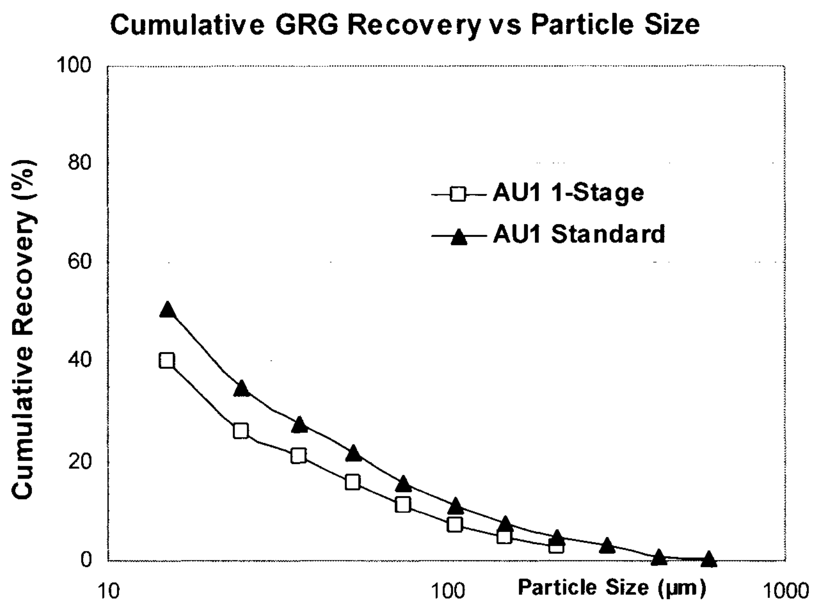


Figure 32: AU1 GRG size distribution

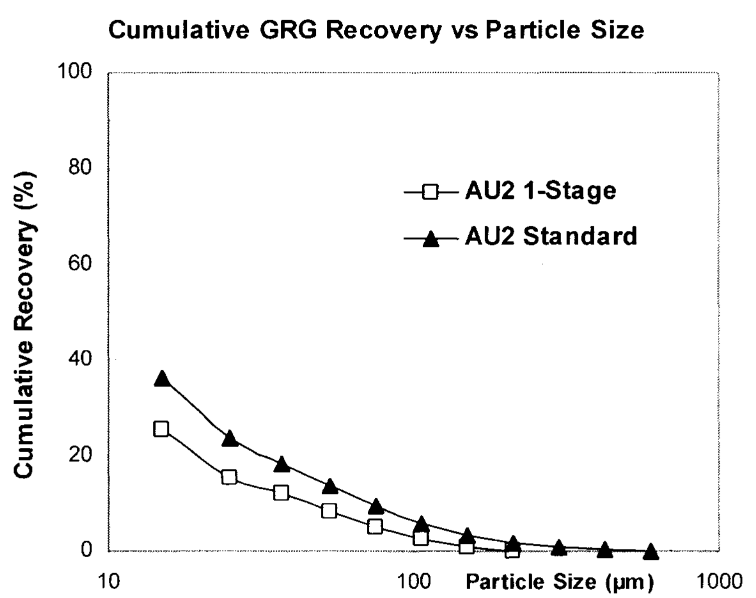


Figure 33: AU2 GRG size distribution

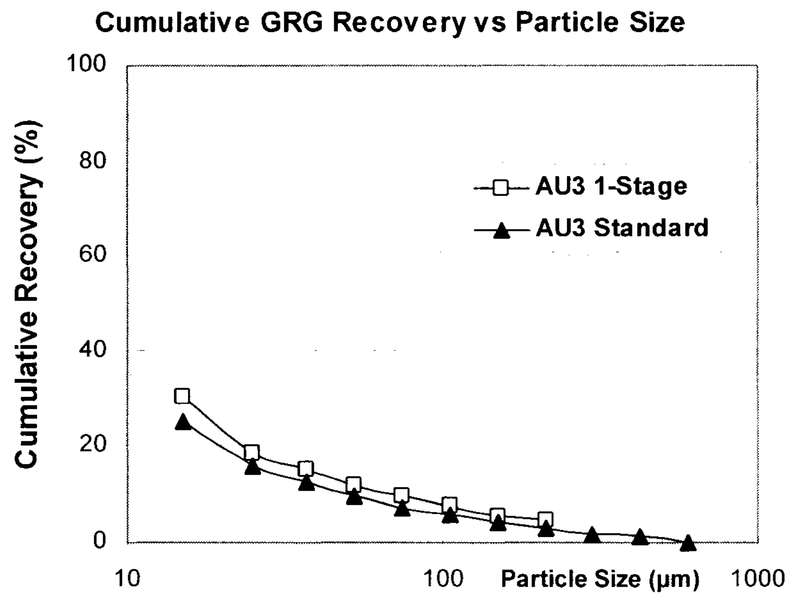


Figure 34: AU3 GRG size distribution

The 1-stage test GRG contents of the three samples varied between 10% less and 5% more than that of the standard test. The calculated head feed grades of the standard and 1-stage tests also fluctuated significantly, up to 21% relative (AU3). This suggests that low-grade (<1 g/t) ores may prove difficult to evaluate with the 1-stage test, even when the GRG is not particularly coarse –i.e. not significant amounts of GRG above 300 μm. On average, the three one-stage tests yielded a GRG content that was 5.3% lower in absolute value and 11.1% lower in relative value. It was decided to investigate the nature of the gold present in the non-GRG fraction for both the standard and one-stage tests. The rationale is that differences may shed light into the fate of the GRG of the standard and simple tests. This will now be discussed.

5.8.1 AU Falcon Tests

The objective of this work is to compare the non-GRG of the standard and one-stage tests. To gain added insight into its nature, the non-GRG is further separated into marginal GRG and a final gravity tailing.

Equal masses (10 kg) of the three standard ore sample stage 3 tailings were collected and mixed thoroughly by repeated riffling using the even and odd method. Similarly 10 kg of the tailings of each simplified test sample was extracted and mixed to create a second 30 kg composite sample. Each composite sample was processed with a lab-scale Falcon SB40 which possesses a higher rotational velocity than the lab Knelson MD3, thus allowing it to recover finer GRG particles as previously described in chapter two. The separation products were screened down to 20 μm versus the usual 25 μm in order to gain more information on the fine end of the size distribution.

The simplified composite's head feed grade of 0.61 g/t agreed with the simplified averaged tailings grade of 0.57 g/t, and the standard composite's head feed grade 0.56 g/t matched the standard averaged tailings grade exactly. Full results can be found in Table 13.

Figure 35 shows the cumulative retained marginal GRG curves for the two non-GRG samples (these curves are strictly analogous to the GRG curves already presented in this chapter). Virtually all of the marginal-GRG recovered in both samples was below 20 μm, the region where KC-MD3 recovery is known to be poor. The marginal-GRG contents reported were 13.6% for the one-stage tailings composite and 17.5% for the standard composite. The differences in marginal GRG content between the two tests fell within the boundaries of experimental and assaying errors given the low grade of the samples processed. If anything, the hypothesis that more marginal GRG would be present in the non-GRG component of the one-stage test is refuted.

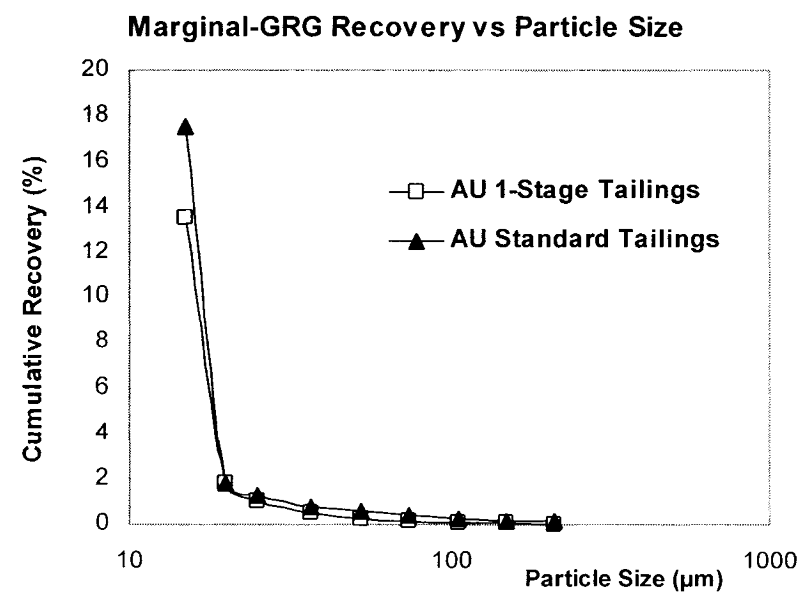


Figure 35: AU Falcon Tests Marginal GRG size distribution

Figure 36 cumulates the non-Gravity component (i.e. the tailing of the SB tests) in a similar manner to the GRG size distributions. As expected, the non-gravity component is higher for the one-stage test. Furthermore the losses increase significantly in the intermediate (minus 106 μm to plus 38 μm) size range, identifying gold that is coated or smeared on the surface of gangue particles, rather than gold that is too fine to report to the marginal GRG component. This insight is significant, but ideally this result should be confirmed with different ore types and higher-grade material (to reduce the impact of sampling and assaying errors).

Particle Size (μm)	FSB Simple Cum Rec. (%)	FSB Standard Cum Rec. (%)
300+	0	0
212+	0.04	0.13
150+	0.05	0.15
106+	0.11	0.27
75+	0.13	0.43
53+	0.27	0.59
38+	0.50	0.80
25+	1.04	1.27
20+	1.82	1.84
Pan	13.57	17.53
Grade (g/t)	0.61	0.56

Table 13: AU Falcon GRG results

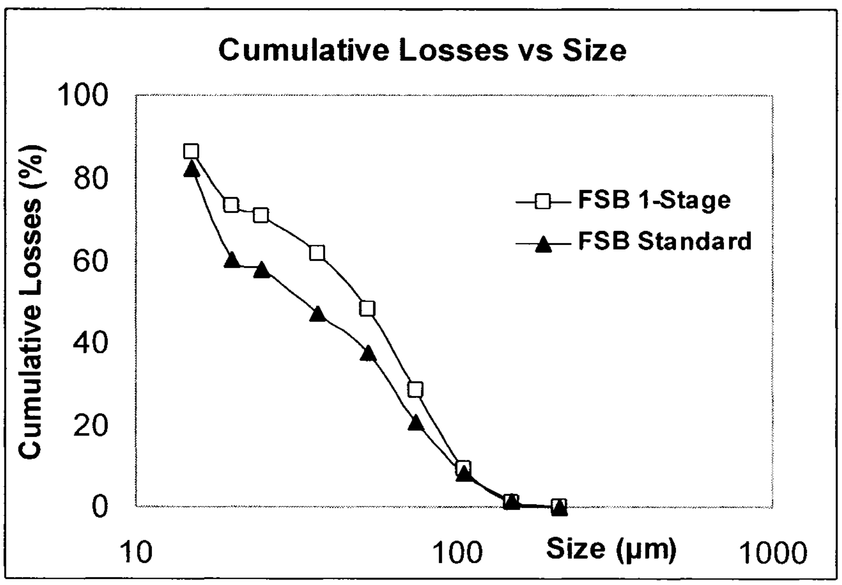


Figure 36: AU Falcon Cumulative Losses

5.9 QC Gold Series (QC3-QC5)

The QC Gold series consisted of three different ore bodies: a sediment sample dubbed QC3, a low-grade sample QC4 and finally a quartz sample QC5. Samples of each type were provided in sufficient amounts for both simplified tests; however, whereas the feed for the standard tests were drill core rejects of high representativity, the feed for the simple tests from bulk samples extracted from underground workings. Therefore it was known beforehand that head grades could differ, which may have an impact on the measured GRG content.

The QC3 sample was the softest with an average grind time of 75 minutes per batch, the QC4 sample ground at 125 minutes on average per batch, and lastly the QC5 sample was particularly hard with an average grind time of 170 minutes per batch. The second sample QC4 experienced a tangle during the one-stage test's grinding phase that caused a slightly coarser size distribution in one of the batches. The other batches were ground longer to achieve a finer size distribution to offset the tangled batch, resulting in the average stage3 target of 80% passing 75 μm.

Particle Size (μm)	Standard QC3 Cum Rec. (%)	2-Stage QC3 Cum Rec. (%)	1-Stage QC3 Cum Rec. (%)
600+	0.09	0.44	0
425+	0.15	1.04	0
300+	0.96	2.39	0
212+	2.30	3.45	0.95
150+	5.14	6.29	3.66
106+	8.95	10.46	7.23
75+	13.75	15.25	12.53
53+	20.51	20.89	17.66
38+	26.38	25.51	22.25
25+	32.50	29.44	25.87
Pan	43.49	37.77	34.67
Grade (g/t)	4.28	1.02	0.95

Table 14: QC3 Cumulative Recoveries

The grades and GRG contents for the QC3, QC4 and QC5 samples can be found in Table 14, Table 15, and Table 16 respectively. The head feed grades reported from this test confirm the ores samples used for the standard and simplified tests were significantly different; head grade was far more variable, from 1 to 25 g/t, compared to 4 to 8 g/t for the standard tests. Head grades for the two simple tests are in good agreement for all three samples. The QC4 underground sample has a much higher grade than that of the drill core, whereas the two other underground samples have a much lower grade than their more representative counterparts. The differences in head grade are significant enough to affect GRG content and size distribution.

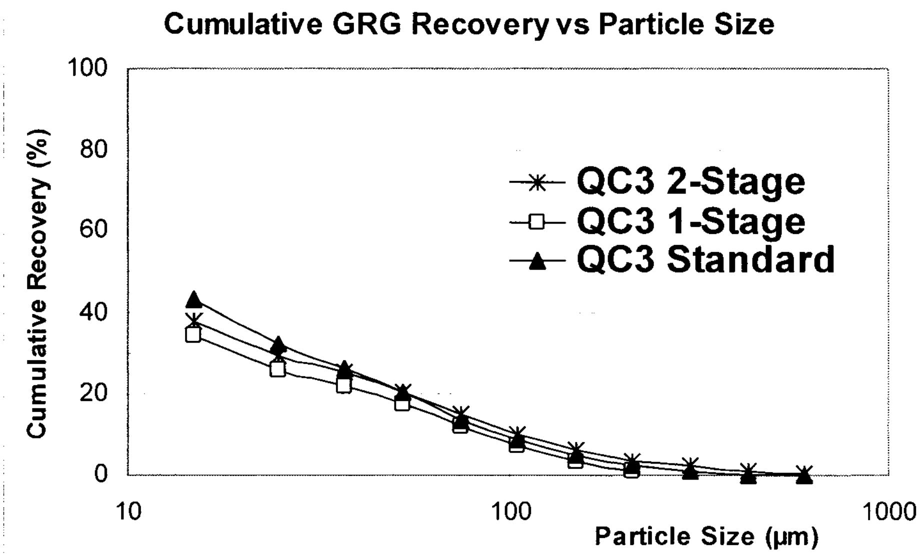


Figure 37: QC3 GRG size distribution

Figure 37 shows that despite the much higher head grade of the standard test, 4 g/t vs. 1 g/t, the 2-Stage and 1-Stage simplified tests return GRG content very similar to that of the standard test for sample QC3.

Particle Size (µm)	Standard QC4 Cum Rec. (%)	2-Stage QC4 Cum Rec. (%)	1-Stage QC4 Cum Rec. (%)
600+	0.15	0.25	0.23
425+	0.94	0.98	0.59
300+	3.33	1.38	1.14
212+	6.67	3.69	2.63
150+	14.02	8.42	6.20
106+	24.48	17.39	14.94
75+	34.70	30.23	26.72
53+	51.92	47.44	42.23
38+	62.01	61.99	56.82
25+	70.60	71.62	66.23
Pan	80.12	85.95	79.62
Grade (g/t)	8.11	24.77	25.03

Table 15: QC4 Cumulative Recoveries

Similarly Figure 38 shows that the large difference in head grade for the QC4 sample, 8 g/t vs. 25 g/t, does not appear to affect GRG content significantly, though the GRG size distributions indicate appreciable comminution of coarse GRG above 106 μm.

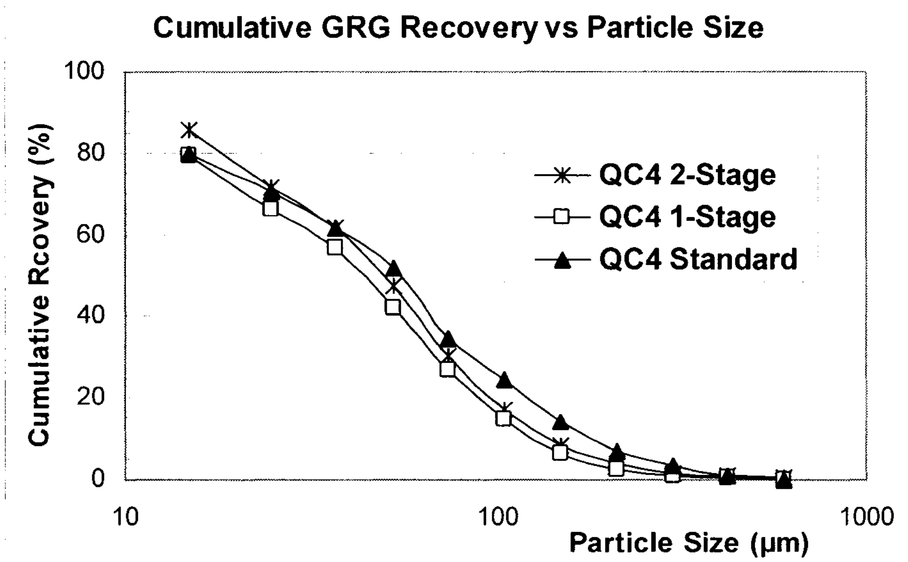


Figure 38: QC4 GRG size distribution

The QC5 sample achieved comparable GRG contents with the exception of the one-stage test whose results were questionably lower when compared to samples QC3 and QC4, for which the fit was very good.

Particle Size (μm)	Standard QC5 Cum Rec. (%)	2-Stage QC5 Cum Rec. (%)	1-Stage QC5 Cum Rec. (%)	1-Stage QC5R Cum Rec. (%)
600+	0.58	0.53	0	0
425+	1.89	1.53	0	0
300+	3.19	2.70	0	0
212+	5.57	4.61	0.57	0.76
150+	10.31	8.16	2.75	3.67
106+	18.67	15.42	8.56	11.43
75+	28.39	27.78	18.18	24.28
53+	42.51	42.55	29.89	39.93
38+	55.05	53.95	39.16	52.31
25+	66.09	61.59	45.14	60.29
Pan	79.02	79.25	54.31	72.53
Grade (g/t)	8.69	1.78	2.17	1.63

Table 16: QC5 Cumulative Recoveries

The difference in head feed grade for the one-stage test was also somewhat larger than normal, therefore a second 600 g of one-stage tailings was sub-sampled, screened and sent for assay (QC5R). The re-screening data were in better accord with the two-stage and standard tests, as was the feed grade. Possible sample contamination during screening may have been the cause of the discrepancy.

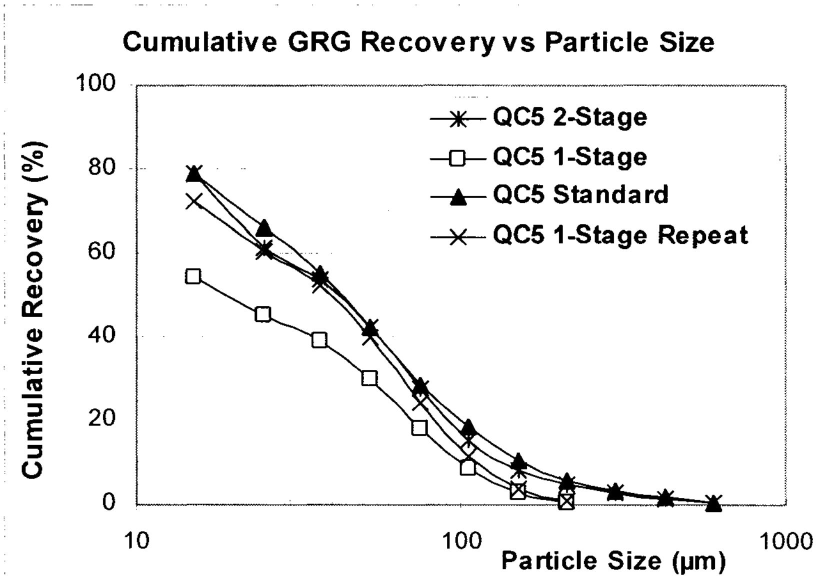


Figure 39: QC5 GRG size distribution

The effect of head grade on GRG content for this sample series can be estimated from the observation that the standard test's results are generally intermediate between those of the two-stage and one-stage tests. Inspection of Figure 37, Figure 38 and Figure 39 would therefore suggest that the impact of the grade difference for the sample QC3 is 7%, 3% for the QC4 and 3% for the QC5R sample. The sign of the differences is consistent with the generally observed fact that GRG content increases with increasing head grade, but in this case, given the small differences in GRG content vis-à-vis the significant swings in head grade, the effect is slight.

5.10 Peru Sample (PE)

The Peruvian sample was processed with the one-stage simplified test only. Average grinding time per batch was 160 minutes due to the samples high hardness. The sample also possessed high magnetite content, with approximately 73 g of the total concentrate yield of 113 g being magnetically-separable via a hand magnet. Full GRG results and grades are provided in Table 17.

Particle Size (μm)	Standard PE Cum Rec. (%)	1-Stage PE Cum Rec. (%)
600+	0.05	0
425+	0.34	0
300+	1.39	0
212+	2.01	1.26
150+	3.51	2.42
106+	6.07	4.64
75+	8.78	7.63
53+	13.47	12.25
38+	19.43	18.15
25+	29.21	25.14
Pan	56.31	47.87
Grade (g/t)	2.19	2.26

Table 17: PE Cumulative Recoveries

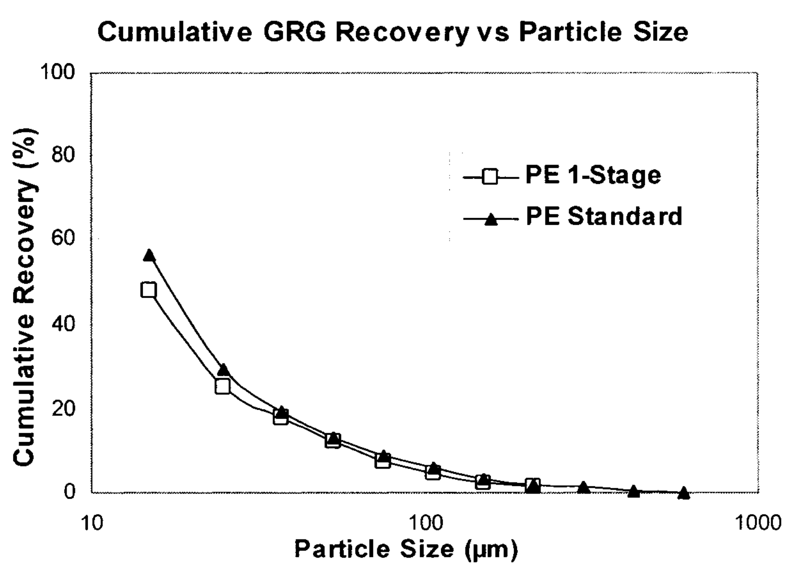


Figure 40: PE GRG size distribution

Figure 40 shows that the GRG content returned by the two tests was very close, as was the fit of the GRG size distributions, albeit the one-stage test had less recovery in the finer size classes (< 38 µm) leading to an overall reported GRG content 7% lower than that of the standard. This lower recovery can be attributed to smearing of GRG in the intermediate size classes (between 106 µm and 38 µm), as can be seen in Figure 41. The graph demonstrates an increasing divergence in losses down to the 38 µm size class, at which point the 1-stage test seems to possess less fine non-GRG than the standard because the two tests' overall losses converge somewhat by the finest size fraction. Feed grades for the two tests were in good agreement, 2.19 g/t vs. 2.26 g/t.

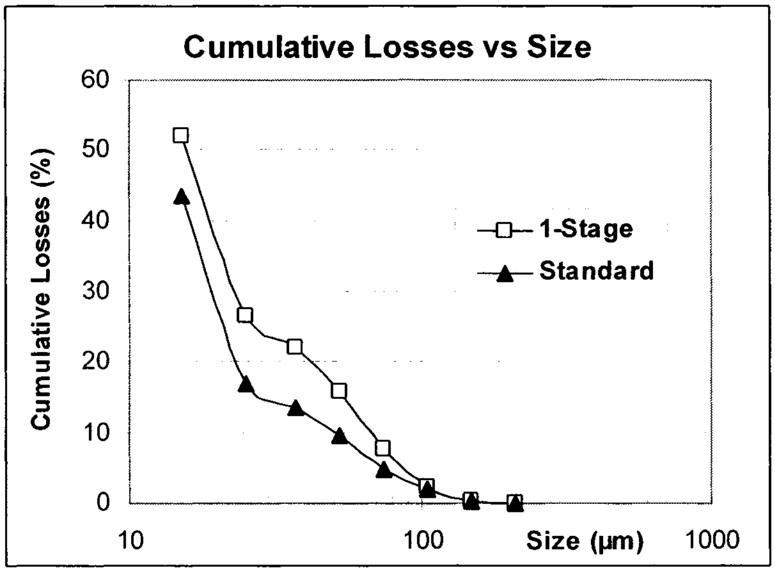


Figure 41: PE cumulative losses

5.11 British Columbia (BC)

The BC sample was from a base metal volcanogenic ore containing about 20% sulphides. Twenty kilograms was processed with the one-stage test under standard operating conditions. The ore sample was relatively easy to grind, averaging 80 minutes per batch. The cumulative GRG recovery size distributions of the two tests run parallel but are offset by a large margin (11% absolute, 33% relative to the one-stage test). Also the head feed grades fail to agree, the standard test being almost 0.5 g/t higher than the one-stage. Figure 42 shows that the missing GRG component in the one-stage test is located at coarse size – above 600 µm.

Particle Size (μm)	Standard BC Cum Rec. (%)	1-Stage BC Cum Rec. (%)
600+	15.89	0
425+	17.09	0
300+	20.32	0
212+	22.82	6.56
150+	25.44	12.25
106+	27.90	15.03
75+	30.25	17.99
53+	33.10	21.35
38+	36.10	24.86
25+	39.83	27.90
Pan	43.53	32.03
Grade (g/t)	2.12	1.77

Table 18: BC Cumulative Recoveries

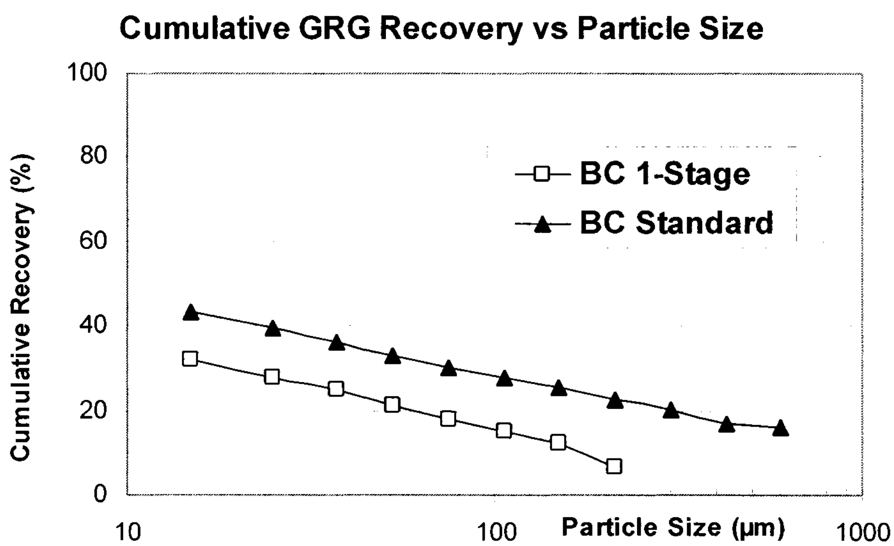


Figure 42: BC GRG size distribution

Optical microscope examination of the five coarsest size fractions of stage 1 (standard test) detected two large gold flakes in the coarsest size class (600 μm). The two large flakes totalled 22.5 mg in the 107 g of concentrate recovered in stage 1, contributing 0.338 g/t of the overall sample feed grade (2.12 g/t). Assuming that the two gold particles observed were the only ones present in the 600+ μm concentrate and that the 66 kg of ore used for the standard test can be treated as the population for that ore type, then theoretically the average number of similar gold particles observed in 20 kg should be equal to 20 kg/ 66 kg* 2 particles = 0.6 particle. Assuming a Poisson distribution, the probability of having zero particles present is 55%, that of one particle 30%, and 15% for 2 or more particles, as shown in Figure 43.

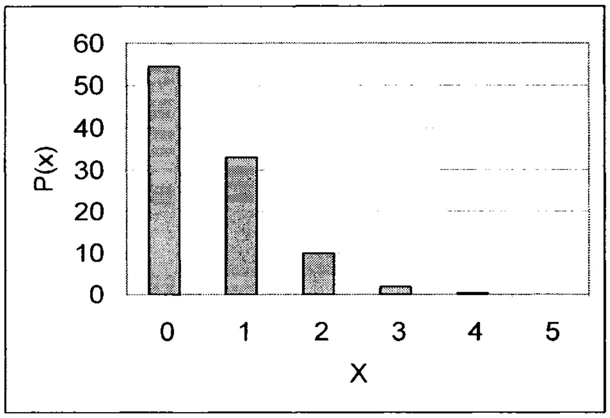


Figure 43: Poisson distribution for number of particle (x) (mean = 0.6)

Given that the most likely outcome is no particle, the standard test's recovery in the 600 μm size class was set to zero to observe the effect on the overall GRG content: it dropped to 32.9% with a head feed grade of 1.79 g/t, values very similar to the one-stage test results, 32.0% and 1.77 g/t. What is clear is that even if one or more +600 μm particles had been included in the 20-kg sample of the simple test, results would still not match the standard test, as the head grade and GRG content would be higher due to the lower mass used. It is, however, possible to match the two results if the number of gold particles in the **standard** test is varied from 0 to 5, assuming that each flake contains 11.25 mg of gold (as measured for the two flakes of the standard test), the effects which are shown in Figure 44 for head grade and in Figure 45 for recovery. Clearly, in the absence of coarse gold particle, the standard test results align with those of the simple test.

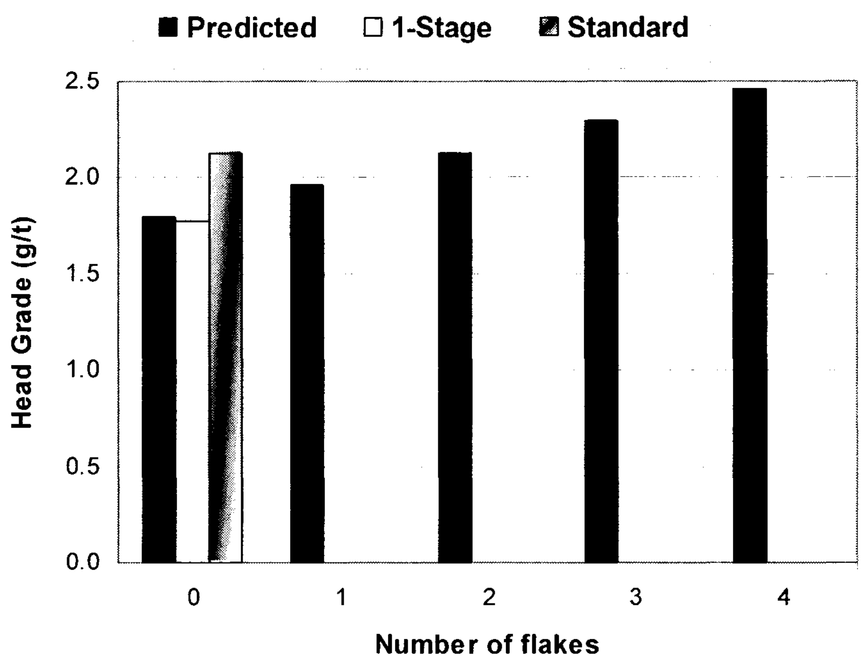


Figure 44: BC predicted head grades assuming the number of coarse gold flakes in the standard test feed

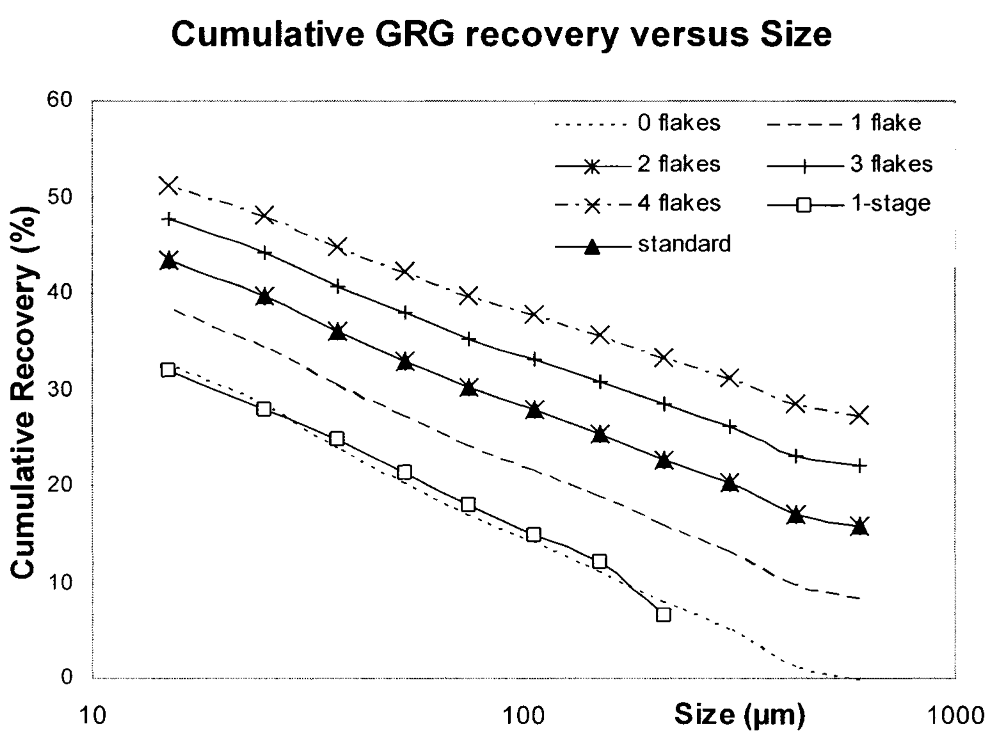


Figure 45: BC GRG cumulative content in the standard test as a function of the number of gold flakes (each gold flake assumed to contain 11.25 mg of gold).

Though it is impossible to confirm the number of 600 μm particles in the one-stage test feed, test results (head grade and GRG content) strongly support the absence of coarse gold. Susceptibility to the Nugget effect is an important caveat to the simple test, and the numerical correction method outlined above is one way in which to reconcile the data, but only when both the standard and simple test are performed, and the coarse concentrate of the standard microscopically examined to determine the number of coarse gold particles. Such a practice should be systematic for the standard GRG test.

5.12 Quebec Sample 6 (QC6)

The QC6 sample originates from an Archean vein type copper-gold deposit. It was provided in sufficient quantity to be processed with both the two-stage and one-stage tests. The sample was of moderate hardness, with an average grind time of 90 minutes per batch in order to reach the target size distribution. The recovered concentrates had minor magnetic content (< 1% by weight) in the form of long (~5 mm), thin magnetite particles that were easily separable with a hand magnet.

Particle Size (μm)	Standard QC6 Cum Rec. (%)	2-Stage QC6 Cum Rec. (%)	1-Stage QC6 Cum Rec. (%)
600+	3.38	2.54	0
425+	5.02	4.04	0
300+	7.43	6.04	0
212+	10.52	9.29	5.20
150+	14.62	14.08	10.10
106+	19.96	19.44	15.19
75+	25.54	25.04	20.99
53+	32.07	31.45	27.56
38+	38.09	37.81	34.06
25+	43.28	42.74	39.26
Pan	53.85	52.63	48.99
Grade (g/t)	9.79	9.92	9.50

Table 19: QC6 Cumulative Recoveries

The cumulative GRG contents and head feed grades reported in Table 19 for all three tests are in good accord within the bounds of experimental error. The one-stage test's recovery is slightly lower than the other two tests but this trend is normal for the database of samples studied up to this point. Figure 46 shows that the cumulative GRG recovery curve for the one-stage test is nearly log-linear with particle size at an R^2 of 0.998. The same trend is displayed for the standard and two-stage tests below 212 μm, but not at the coarse end. The MC sample was the only one in the target database for which this log-linear trend was observed.

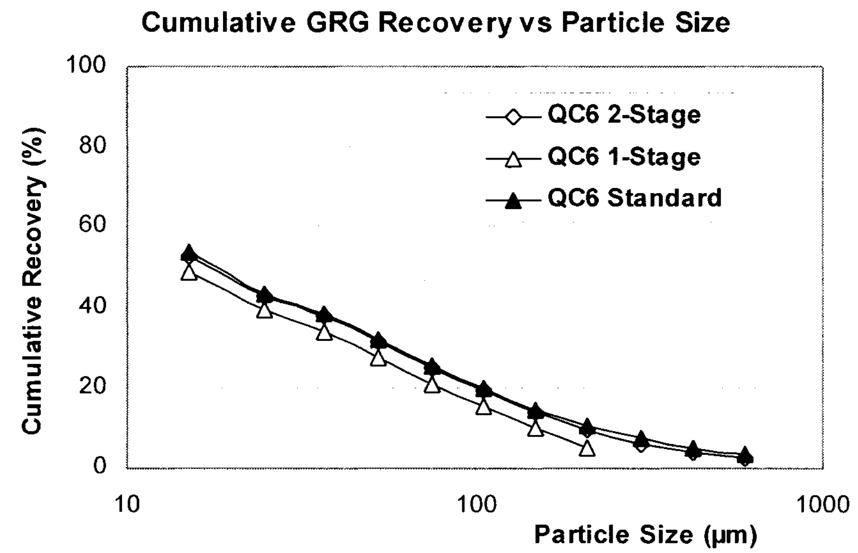


Figure 46: QC6 GRG size distribution

5.13 Saudi Series (SA1-SA2)

Two Saudi Arabian ore samples of known high GRG content were provided for processing with the one-stage test, one average grade sample dubbed Saudi Low (SA1) and a very high-grade sample, Saudi High (SA2). Twenty kilograms of each sample was used for the one-stage test, and a second twenty kilograms of SA2 was kept aside for a repeat one-stage test (SA2B). Due to the high hardness of the Saudi samples, the tailings of the SA2 sample from both the standard and simplified tests were processed with the Falcon SB40 to assess their marginal GRG contents in the same manner as the Western Australia (AU) series.

5.13.1 Saudi Low (SA1)

The SA1 sample required an average grind time of 145 minutes per batch. One of the batches leaked from the mill during the grinding process near the end of the grind cycle. The cause of the spill was incomplete tightening of the mill cover, which gradually loosened itself during the grinding cycle. The sample was recovered; though there was slightly more sample mass recovered than the original mill charge, the threat of contamination was deemed sufficiently low given the known high GRG content of the ore and the fact that the only two kilograms of the total twenty were affected. The sample was re-riffled several times to ensure adequate mixing.

Particle Size (μm)	Standard SA1 Cum Rec. (%)	1-Stage SA1 Cum Rec. (%)
600+	0.8	0
425+	2.1	0
300+	4.3	0
212+	8.6	4.86
150+	14.6	9.29
106+	20.6	14.03
75+	26.3	21.45
53+	33.4	29.40
38+	38.6	35.71
25+	43.9	40.29
Pan	52.8	50.05
Grade (g/t)	3.43	3.17

Table 20: SA1 Cumulative Recoveries

Figure 47 shows that both tests are in accord despite the aforementioned grinding issue. The cumulative GRG size distributions of the two tests are similar albeit the one-stage test is shifted slightly lower, as usual, with a relative drop of 5% in recovery. It should be noted that this drop is fairly benign as compared to other hard ore types that have suffered relative drops as high as 33% (AU series). Inspection of the cumulative losses of the two tests in Figure 48 unveils an increase in losses for the intermediate size classes (25 μm < x < 75 μm) after which the distributions converge somewhat down to -25 μm. This implies that the losses are attributable to gold smearing onto the hard and abrasive gangue.

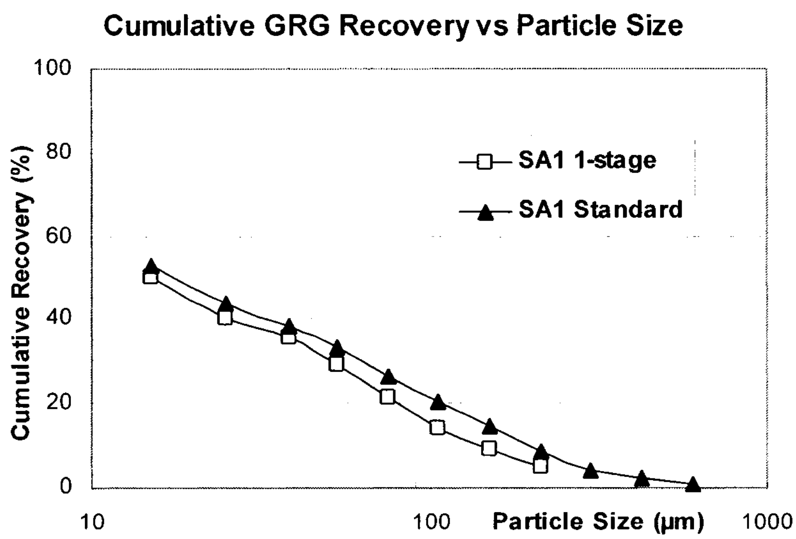


Figure 47: SA1 GRG size distribution

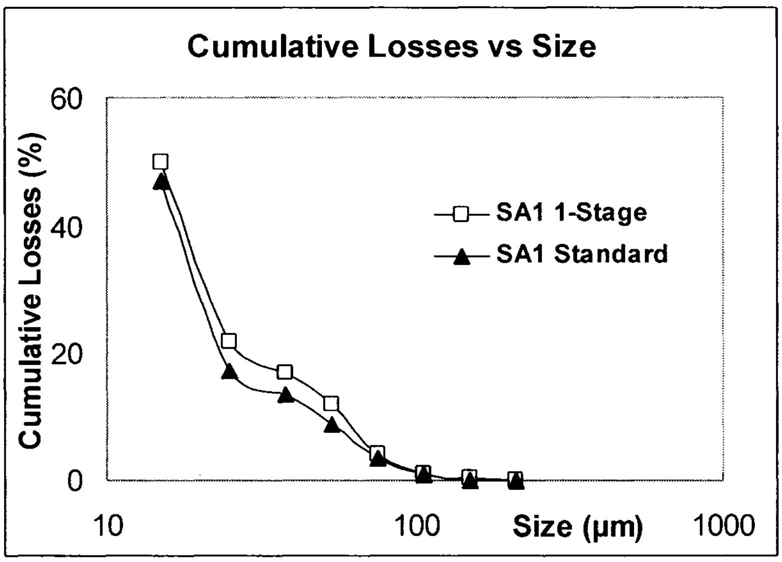


Figure 48: SA1 Cumulative Losses

5.13.2 Saudi High (SA2)

The high grade Saudi sample was even harder than its SA1 counterpart, with an average grind time of 155 minutes per batch. The head feed grades yielded by the standard and one-stage tests were 44.4 and 45.0 g/t respectively, which is in very good agreement. The cumulative GRG size distributions are of similar shape; however the one-stage test's curve is shifted 6% lower than the standard.

Particle Size (μm)	Standard SA2 Cum Rec. (%)	1-Stage SA2 Cum Rec. (%)	1-Stage SA2B Cum Rec. (%)
600+	0.5	0	0
425+	1.1	0	0
300+	4.2	0	0
212+	8.0	2.93	3.08
150+	17.9	9.09	9.50
106+	28.4	19.34	18.25
75+	38.9	31.18	30.71
53+	51.4	45.49	44.85
38+	62.4	57.46	56.60
25+	72.1	64.90	64.27
Pan	82.1	76.34	75.71
Grade (g/t)	44.43	45.03	45.14

Table 21: SA2 Cumulative Recoveries

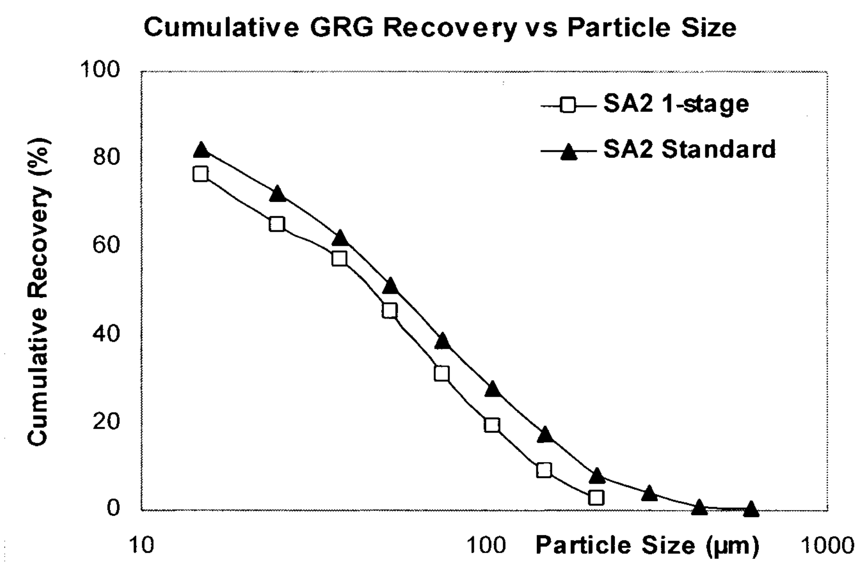


Figure 49: SA2 GRG size distribution

To estimate experimental error, a repeat one-stage test (SA2B) was conducted under identical lab conditions. The results of the SA2 and SA2B test work are show in Figure 50; reproducibility is excellent.

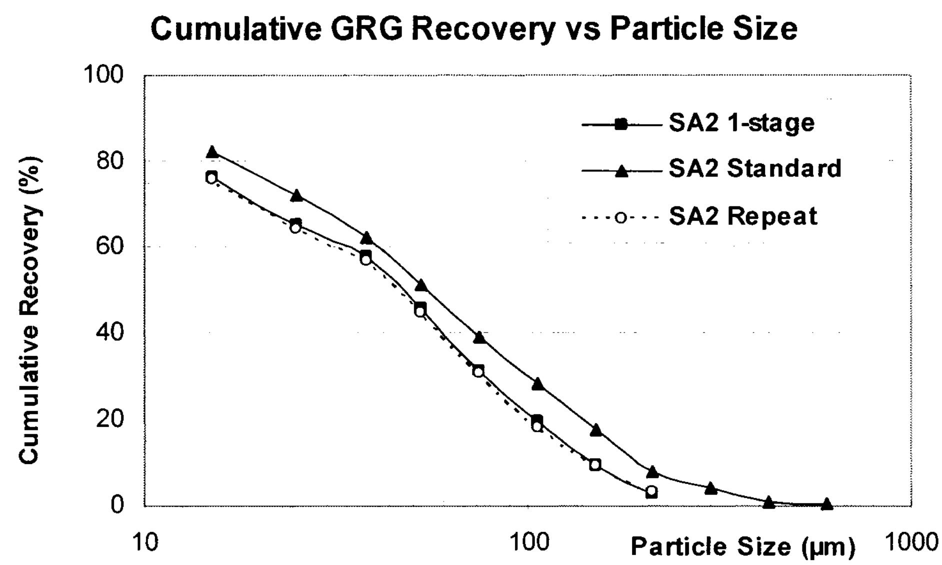


Figure 50: SA2B GRG size distribution

There is a missing coarse GRG component above 150 µm: the standard GRG test has a cumulative GRG content of 18% above 150µm, whereas the average one-stage test has 9%. The overall GRG difference drops from 9% to 6% by the final size class, which implies that 3% of the missing 9% GRG at coarse size was recovered as finer GRG; the remainder must have been transformed into marginal GRG and non-GRG. Figure 51 shows that the cumulative tailings distribution for the repeat test is slightly different from that of the original one, with more non-GRG above 25 µm and less below –i.e. more smearing and less actual grinding.

Figure 51 presents the cumulative retained distribution of the non-GRG for the two tests. Of the 6.1% difference, 3.1% occurs above 25 µm and 3% below. This is an indication that smearing alone, which takes place largely above 25 µm, cannot account for the full difference in GRG content. The other logical cause of GRG loss in the 1-stage test is a transformation of the intermediate and fine GRG into non-GRG and possibly very fine, marginal GRG.

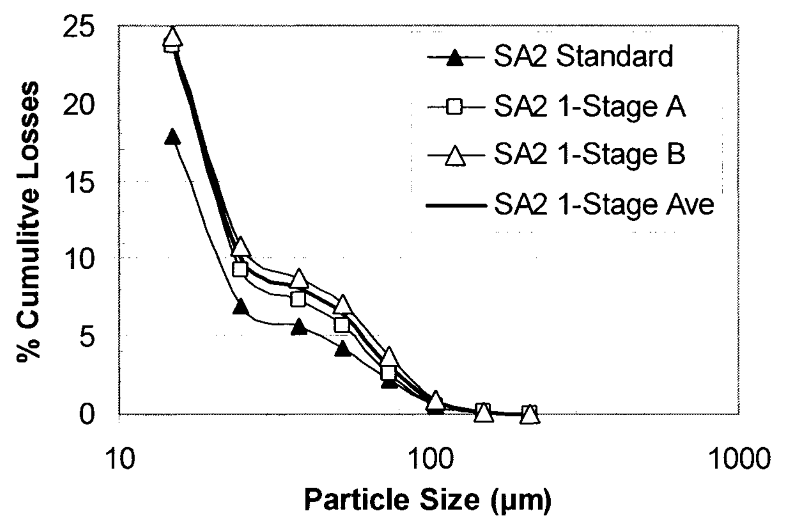


Figure 51: SA2 and SA2B cumulative losses

5.13.3 SA2 Falcon Work

Since the head feed grade of the SA2 sample was far higher than that of the previous AU series, only 20 kg of tailings was used from the standard test and blended tailings of the one-stage test. The procedure is described in section 4.4.

The Falcon test with the standard GRG test tailing yielded a recovery of 31.4% with a head grade of 8.7 g/t (it had been measured at 8.0 g/t in the standard test). For the simple test, head grade was 10.5 g/t (as opposed to 10.7-11.0 g/t for the tailing of the two simple tests), with a recovery of 36.7%. As for the previous Falcon tests, the gold thus recovered is described as marginal GRG, which is a subset of the non-GRG. The marginal GRG for the standard test is 5.9% of the total gold in the standard sample (scaling down the 100-kg feed mass to match the 20-kg of the 1-stage test); this value is lower than the 8.2% for the 1-stage test, and the difference lies in the amount of additional marginal GRG produced by the 1-stage protocol.

Figure 52 shows the cumulative marginal GRG of the standard and one-stage tests (100% is the total gold in the ore). In both cases, the bulk of the marginal GRG is finer than 20 µm, 80% for the standard test and 72% for the 1-stage test. The long grinding time of the 1-stage test prior to any GRG recovery has transformed GRG particles into marginal GRG particles that are too fine (below 20 µm) or too flaky (mostly above 20 µm) to be recovered efficiently by a LKC at 60 Gs.

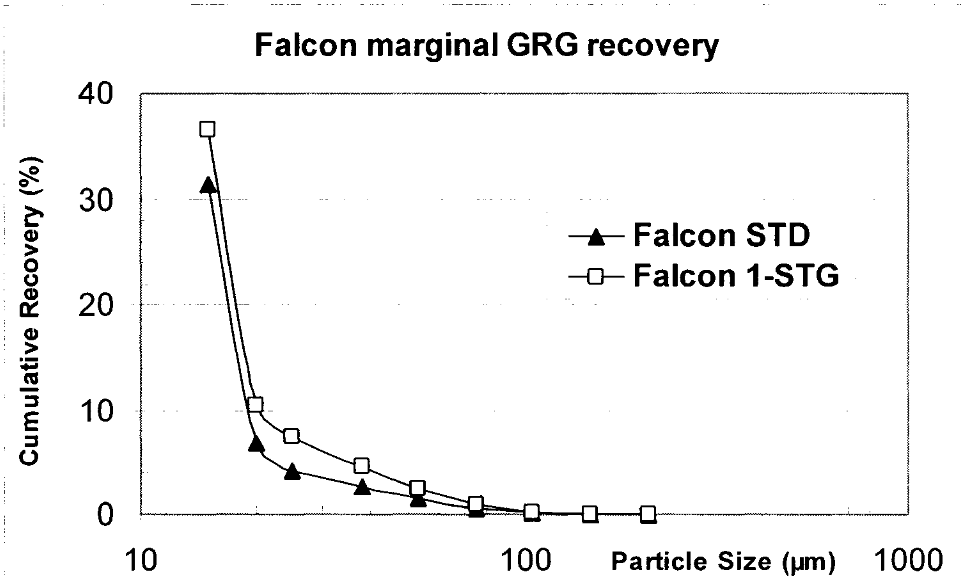


Figure 52: SA2 Falcon Tests Marginal GRG size distribution

The presence of marginal GRG above 20 μm was not observed with the Western Australia (AU) series graphs, possibly due to the low grade of the sample (1 g/t versus the 44 g/t in SA2). The high-grade SA2 has helped add an additional level of insight into the distributions of marginal GRG and non-GRG, as displayed in Figure 53. The AU gangue is also known to be highly abrasive, which may have increased smearing to the detriment of fine/flake free gold production –i.e. marginal GRG.

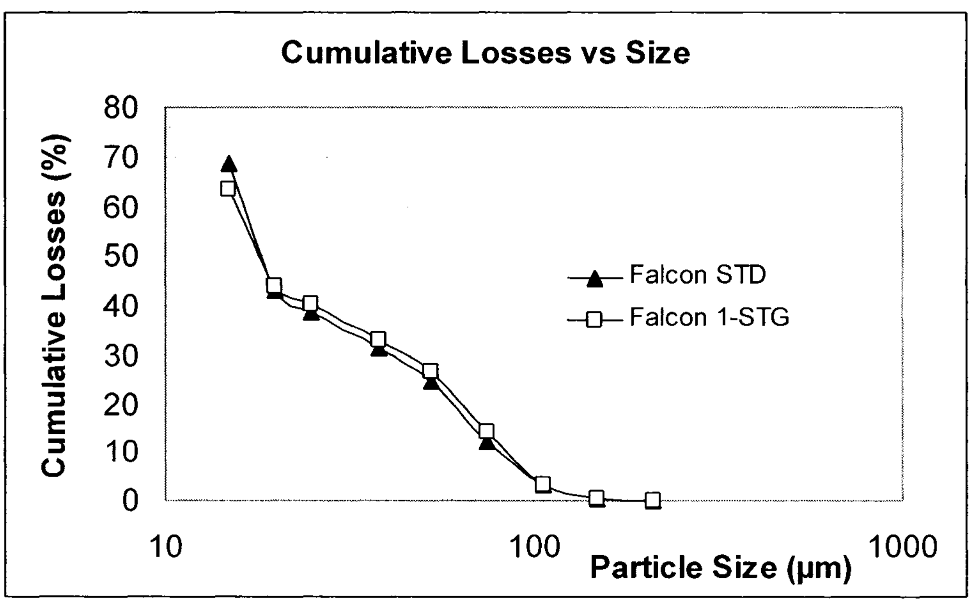


Figure 53: SA2 Falcon cumulative tailings distribution

5.14 Overall Results

An overall comparison of the simple GRG tests' results versus that of the standard is provided in Figure 54. For non-abrasive ores, the simplified GRG test results match those of the standard rather well (~10% relative error on average). However for abrasive ores the differences in reported GRG content can be significant (as high as ~32% relative in the case of the GH and AU samples), with loss of size distribution.

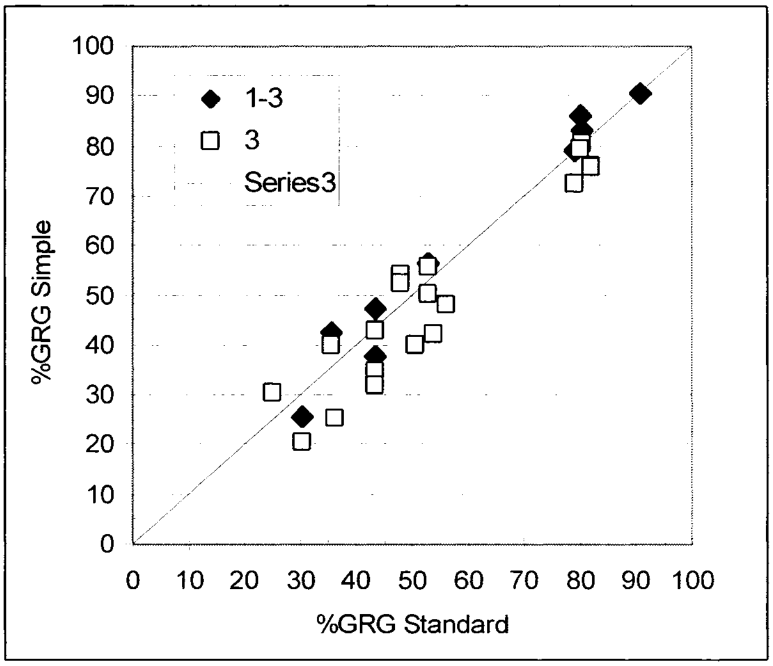


Figure 54: Simple GRG test results vs. Standard GRG results

Each of the ore samples processed varied in hardness and grind time to achieve the target product size (P_{80}) of 75 µm; it was originally assumed that samples ground longer would have higher GRG losses due to smearing and generation of fine non-GRG. However, as can be seen in Figure 55, many samples that were ground for as long as 180 minutes suffered lower relative GRG losses than other samples processed for approximately 120 minutes. No apparent relationship exists between the two; it is likely that abrasiveness is a better predictor of GRG loss in the 1-stage test than hardness. This hypothesis presents an interesting research avenue for further study.

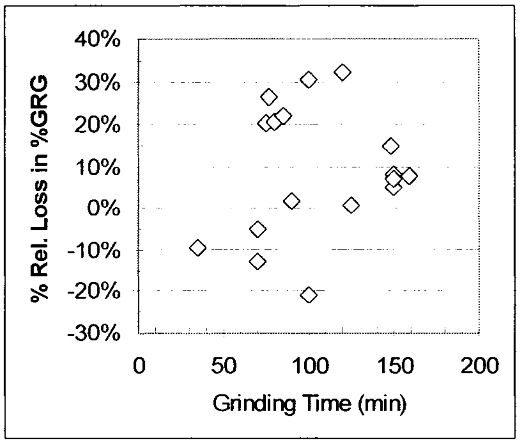


Figure 55: Effect of grinding time on GRG losses (positive implies loss of GRG)

The screen calibration test (section 2.5) was applied to a few samples (BC and SA2), but no significant changes occurred in the GRG distributions, only a minor smoothing out of the concentrate recovery fluctuations for individual size classes. Still the calibration process was a worthwhile effort to confirm that the screening procedure was not the cause of GRG distribution discrepancies between standard and simplified tests.

6 Conclusions

The sample database processed throughout the course of this research revealed several unanticipated results for the simplified GRG tests. These ranged from necessary modifications of the experimental procedure to finding limitations of the simple GRG test's capabilities. On a whole the 1-stage simplified GRG test has proven to be a viable option for those seeking a quick alternative to the standard GRG test; the 2-stage option proved to be superfluous since it did not provide any additional insight into the nature of the simplified test that was not already found with the 1-stage test.

6.1 General Conclusions

The Quebec Cu-Au series (QC1 & QC2) demonstrated the importance of proper ore sample procurement: though QC1 and QC2 were extracted from the same ore body, their simple test GRG contents (41% on average for QC1 versus 55% on average for QC2) proved that they were very different samples. As previously stated, drill-core rejects or long-term composite samples are the ideal feeds for the GRG test.

The Guyana series (GU1 & GU2) showed that variation in fluidization water flow-rate has little effect on the KC-MD3 performance so long as there is enough to maintain bed fluidization. The large difference in returned head grade for the 1-stage GU2 Quartzite sample (3.4 g/t versus the nominal 2.8 g/t for the other tests) reinforced the importance of proper sample mixing if the ore sample provided is to be split for multiple tests. These results lead to an extension of the riffling step in the lab procedure.

The Mexican sample demonstrated that massive sulphide ores, which by nature are not abrasive, do not seem to pose a problem for the 1-stage and 2-stage simplified GRG tests. The low sample mass processed and low feed rate allow the KC-MD3 to operate under near-ideal conditions.

The Ghanaian composite sample (GH) was the first indication that the 1-stage test encounters difficulty with abrasive and hard ore samples. The relative difference in returned GRG content for 1-stage test was ~33% lower than that of the standard; however the head feed grade and GRG curve shape were preserved. It was initially assumed that the abrasive nature of the ore caused smearing of GRG particles onto gangue during the grinding phase, leading to a decreased recovery with the KC-MD3.

The Western Australia copper-gold series (AU1 to AU3) provided several interesting insights: the first was that the 20-kg limitation of the simplified GRG test may be insufficient to obtain truly representative samples for low-grade stock material – i.e. 1 g/t or less – even when composited from drill core. The second insight came from the Falcon work, where it was found that the 1-stage and standard GRG tests generate similar marginal-GRG content, but that the 1-stage test possessed significantly more non-GRG in intermediate size classes (minus 106 μm to plus 38 μm), attributable to gold smearing.

The Quebec Gold series (QC3 to QC5) reinforced the generally observed fact that GRG content increases with increasing head grade, though the effect was only minor in this case despite the large differences in head grade observed (+16 g/t) between the standard and simplified test samples. If the simplified test is to be used as the sole measure of an ore body, it is important that the samples used be truly representative; these swings in grade can affect the GRG results generated and other samples may not return results as close as this series. Though two of the three samples in this series were extremely hard, they were not abrasive samples and the reported GRG contents and size distributions of the simplified tests were in accord with the standard; this lead to a refinement of the hypothesis proposed in the Ghanaian series: abrasiveness seems to be the more detrimental factor to simplified test performance rather than ore hardness. Other abrasive samples processed later further confirmed this theory. However, this work did not directly measure abrasiveness; rather, evidence of abrasiveness was obtained from the corporations from which samples were obtained.

The high-grade Saudi Arabian series (SA1 & SA2) shed further light onto the nature of the losses of GRG during the 1-stage test process. Since the grade was far higher than the Western Australia series (44 g/t versus 1 g/t on average), there were far more gold particles available for recovery with the Falcon SB40. This yielded a good data set from which to draw conclusions: approximately half of the gold was lost to smearing in the intermediate size classes, the other half of the gold losses were below 20 μm in the form of fine gold. The predominant smearing loss observed in the AU series was probably due to its extremely abrasive nature, but its low grade precluded the capability to discover the extent of the fine non-GRG component (i.e. marginal GRG).

The relative proportions of smearing losses and fines losses are dependent on the ore type; the SA2 sample had large fines losses, other samples have had very little.

Lastly the British Columbia (BC) sample's GRG results highlighted an important caveat of the simplified GRG test: susceptibility to the Nugget Effect. Ore samples possessing significantly coarse GRG flakes (above 425 μm) run the risk of omission in the small 20-kg sample size during the sub-sampling process, as was the case for this sample, for which only 2 large flakes were observed in the corresponding 66 kg sample used for the standard test. Circumstantial evidence strongly suggests that the 20-kg sample did not contain any such coarse gold particles. The absence of such large gold particles can cause significant drops in reported head feed grade and GRG content, and unfortunately there is no way to confirm the presence of such flakes in the 1-stage test feed due to the grinding phase prior to centrifuge processing.

The database processed thus far has demonstrated that for non-abrasive ores the two-stage test tends to slightly over-predict GRG content and the 1-stage test slightly under-predicts it. The positive bias of the 2-stage test is likely attributable to the lower feed mass used for two stages of recovery (20 kg for stage 1 and ~18kg for stage 3, versus the 60 kg for stage 1 and 24 kg for stage 3 in the standard test). Since the 1-stage test is far simpler to perform and returns similar results that are reproducible (as seen in SA2 and GU1), the 2-stage test has been deemed superfluous, hence its discontinued use approximately half-way through the test-work (unless the client(s) provided adequate mass for both simple tests).

Though the sample set of 18 ores processed during the course of this research has provided some interesting results, it has also raised many questions and there are several aspects that can and should be explored in further detail in future projects.

6.2 Recommendations and Future Work

While the Falcon tests performed on the AU and SA series have shed insight into the types of marginal GRG generated by the 1-stage tests, the picture is far from definitive and additional samples (preferably of high grade, >10 g/t) of varying abrasiveness and hardness should have their tailings processed with the Falcon SB40 to see if a more concrete relationship can be established between grade, abrasiveness, and grind time for the 1-stage test.

Since GRG recovery by flash flotation is often attempted prior to the flotation effort, a combination simple GRG / flash flotation test-work should be explored. The development of such a test could profoundly increase the usefulness of the simple GRG test.

The test database for this thesis uncovered some potential limitations for the 1-stage simple GRG test, but the list of caveats generated is far from complete and as such additional samples should be processed with both the standard and 1-stage test to continue to develop the ongoing database and draw further conclusions. There are an innumerable amount of ores being processed worldwide and the greater the variety of samples to which the 1-stage test is exposed, the greater the understanding of the test's potential and applicability will become.

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8 Appendix A: Calculation procedure

The GRG metallurgical balance is calculated from a combination of two items: the size-by-size masses recorded during the concentrate and tailings screenings, and the gold contents returned for each of these size-classes. All calculations are performed in a Microsoft Excel spreadsheet. This tutorial first illustrates how to calculate one stage's worth of GRG data, which is what is used for the one-stage simplified test. Next the overall GRG content worksheet for a 2-stage test is presented, and lastly the overall GRG content calculations for the standard GRG test will be presented.

8.1 Notations used to simplify calculations:

F – Total feed mass (g)
C – Total concentrate mass (g)
T – Total tailings mass (g)
T_* – Total tailings mass of screening sub-sample (g) [actually a constant = 600 g]
F_i – Feed mass of size class i (where i denotes the lower boundary of the size class in μm)
C_i – Concentrate mass of size class i (g)
T_i – Tailings mass of size class i (g)
T_{i*} - Tailings mass of sub-sample's size class i (g)

R_c – Total concentrate GRG recovery (%)
R_t – Total tailings GRG recovery (%)
R_{ci} – Concentrate recovery of size class i (%)
R_{ti} – Tailings recovery of size class i (%)
D_i – Distribution of gold in feed size class i (%)
Y – Yield of concentrate (%)

c_i – Concentrate grade of size class i (g/t)
c – Overall concentrate grade (g/t)
t_i – Tailings grade of size class i (g/t)
t – Overall tailings grade (g/t)
f_i – Feed grade of size class i (g/t)
f – Overall feed grade (g/t)

U_c – Total units of Au in concentrate (g^2/t)
U_t – Total units of Au in tailings (g^2/t)
U_f – Total units of Au in feed (g^2/t)
U_{ci} – Units of Au in concentrate size class i (g^2/t)
U_{ti} – Units of Au in tailings size class i (g^2/t)
U_{fi} – Units of Au in feed size class i (g^2/t)

8.2 Examples of GRG tables

Feed = 19.99 kg

QC5, Stage 1, 6.010 L/min, 3.0 psi (3.4 psi), 25:46 min, 713.3 g/min

(File Simple QC5.XLS) %-75µm: **21.46** %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)
600	42.70	21.97	4.4	189	5.44	2731	13.80	1.20	3277	94.56	2774	13.88	1.25	3466	9.71
425	57.83	29.76	6.2	356	6.42	3575	18.06	1.45	5183	93.58	3633	18.17	1.52	5539	15.52
300	40.31	20.74	10.3	417	8.91	2666	13.47	1.60	4265	91.09	2706	13.54	1.73	4682	13.12
212	22.95	11.81	21.3	489	14.12	2285	11.54	1.30	2970	85.88	2308	11.54	1.50	3459	9.69
150	12.28	6.32	91.8	1127	37.11	1528	7.72	1.25	1910	62.89	1540	7.70	1.97	3036	8.51
106	7.27	3.74	195.2	1419	48.79	1354	6.84	1.10	1489	51.21	1361	6.81	2.14	2908	8.15
75	4.72	2.43	370.2	1747	58.57	1373	6.94	0.90	1236	41.43	1378	6.89	2.17	2983	8.36
53	3.34	1.72	490.1	1637	51.79	1792	9.05	0.85	1523	48.21	1796	8.98	1.76	3160	8.86
37	1.82	0.94	676.4	1231	62.57	921	4.65	0.80	737	37.43	922	4.61	2.13	1968	5.51
25	0.67	0.34	1078.7	723	48.50	595	3.00	1.29	767	51.50	595	2.98	2.50	1490	4.18
15	0.44	0.23	1841.7	810	27.11	977	4.93	2.23	2178	72.89	977	4.89	3.06	2989	8.38
Total	194.33	100.00	52.2	10144	28.43	19796	100.00	1.29	25536	71.57	19990	100.00	1.78	35681	100.00

0.97 % yield 19796 Verification

Figure 56: GRG data for one stage

QC5 2-Stage, Overall Results (Stage 1 to Stage 3)

(File Simple QC5.XLS)

Size (µm)	First Stage: 100% -850 µm Stage Recov.	Dist'n	Rec. g/t	Third Stage: 88.32 % -75 µm Stage Recov.	Dist'n	Rec. g/t	Losses g/t	Total Recov. g/t	Total Recov. %	Cumul. Recov. %
600	5.44	9.71	0.009	0.00	0.00	0.000	0.00	0.009	0.53	0.53
425	6.42	15.52	0.018	0.00	0.00	0.000	0.00	0.018	1.00	1.53
300	8.91	13.12	0.021	0.00	0.00	0.000	0.00	0.021	1.17	2.70
212	14.12	9.69	0.024	99.64	0.76	0.010	0.00	0.034	1.91	4.61
150	37.11	8.51	0.056	98.04	0.56	0.007	0.00	0.063	3.56	8.16
106	48.79	8.15	0.071	94.68	4.83	0.058	0.00	0.129	7.25	15.42
75	58.57	8.36	0.087	92.78	11.24	0.133	0.01	0.220	12.37	27.78
53	51.79	8.86	0.082	87.19	16.31	0.181	0.03	0.263	14.77	42.55
37	62.57	5.51	0.062	88.50	12.55	0.142	0.02	0.203	11.40	53.95
25	48.50	4.18	0.036	83.85	9.36	0.100	0.02	0.136	7.64	61.59
15	27.11	8.38	0.041	48.45	44.39	0.274	0.29	0.315	17.66	79.25
Total	28.5	100.0	0.507	71.0	100.0	0.906	0.37	1.413	79.25	
O/A	28.5			50.8						
Yield	0.00972			0.00491						
Grade	1.78 g/t			1.29 g/t						
Calc.:	1.78 g/t									

Figure 57: Overall GRG data for a two-stage test

1-stage: Calculations

Given variables (i.e. measured at the beginning of the test)
F (g)

Known variables (after the test is complete)
All C_i's (g)
All T_{i*}'s (g) for the sub-sample
All c_i's (mg Au)
All t_i's (g/t)

Step 1) Calculate all mass-related data (Figure 58)

Concentrate

Size (μm)	Sample Wt (g)	Weight (%)
+600	42.70	21.97
+425	57.83	29.76
+300	40.31	20.74
+212	22.95	11.81
+150	12.28	6.32
+106	7.27	3.74
+75	4.72	2.43
+53	3.34	1.72
+37	1.82	0.94
+25	0.67	0.34
-25	0.44	0.23
Total	194.33	100.00

Tails

Size (μm)	Sample Wt (g)	% Wt
+600	82.56	13.80
+425	108.06	18.06
+300	80.58	13.47
+212	69.07	11.54
+150	46.18	7.72
+106	40.93	6.84
+75	41.51	6.94
+53	54.18	9.05
+37	27.83	4.65
+25	17.98	3.00
-25	2.37	0.40
Wet U/S	27.16	4.54
Total	598.41	100.00

Figure 58: Screening Data

First C is calculated using:
$C\ (g) = \Sigma C_i\ (g)$ for all C_i's screened

Next T is calculated using:
$T\ (g) = F - C$

Then yield is obtained via:
$Y\ (\%) = [\ C / F\ ] * 100\%$

Concentrate's % weight for each size can be generated via:
$\%WtC_i = [\ C_i / C\ ] * 100\%$

Similarly the tailings' % weight:
$\%WtT_i = [\ T_{i*} / T_*\ ] * 100\%$

Then full tailings weights:
T_i (g) = [T * %WtT_i] / 100%

Now the individual feed weights:
F_i (g) = $C_i + T_i$

Feed % weights:
%WtF_i = [F_i / F] * 100%

Lastly the individual F_i's:
F_i (g) = $C_i + T_i$

Step 2) Calculate all assay-related data

Now that all mass-related data has been calculated, the assay data can be processed. No calculations are necessary for the tailings assays since they are already returned from the lab in g/t. However the concentrate assays are all returned in terms of mg Au and sample weights (g) per size class.

Concentrate
VRAC 113Q Stage 1

Size (µm)	Sample	Sample Wt (g)	Au (mg)	Grade (g/t)
+600	JC447	22.00	0.098	4.42
		20.55	0.090	
+425	JC448	19.00	0.141	6.15
		19.00	0.108	
		19.71	0.106	
+300	JC449	20.00	0.216	10.35
		20.21	0.200	
+212	JC450	11.00	0.221	21.29
		11.83	0.265	
+150	JC451	12.15	1.115	91.77
+106	JC452	7.27	1.419	195.19
+75	JC453	4.70	1.740	370.21
+53	JC454	3.33	1.632	490.09
+37	JC455	1.78	1.204	676.40
+25	JC456	0.61	0.658	1078.69
-25	JC457	0.36	0.663	1841.67
Total		193.50		

Tails
VRAC 113Q Stage 1

Size (µm)	Sample	Au (g/t|)	Sample	Au (g/t|)
+600	JC480	1.20		
+425	JC481	1.45		
+300	JC482	1.60		
+212	JC483	1.30		
+150	JC484	1.25		
+106	JC485	1.10		
+75	JC486	0.90		
+53	JC487	0.85		
+37	JC488	0.80		
+25	JC489	1.29		
-25	JC490	2.14	JC490 dup.	2.32

Figure 59: Assay Data

Concentrate assay:
c_i (g/t) = [mgAu_i (mg) / sampleWt_i (g)] * 10^3 (g/t)/(mg/g)

Now the units of Au per stream size class can be calculated via:
U_{ci} (g^2/t) = $C_i * c_i$ and
U_{ti} (g^2/t) = $T_i * t_i$

Then the total units of Au per stream are obtained:
$U_c = \Sigma U_{ci}$
$U_t = \Sigma U_{ti}$
$U_f = U_c + U_t$

And size-by-size feed units Au:
$U_{fi} = U_{ci} + U_{ti}$

Then size-by-size feed grades can be back-calculated via:
f_i (g/t) = [U_{fi} / F_i]

Step 3) Calculate GRG recovery and distribution data

Recoveries are the last to be calculated now that all mass and assay data for each size class is known. The most important data returned from the GRG test is the gold grade and GRG content. Feed grade was already calculated at the end of step 2 above; recovery calculations are illustrated below.

For size-by-size concentrate and tailings recoveries:
R_{ci} (%) = [U_{ci} / U_{fi}] * 100%
R_{ti} (%) = [U_{ti} / U_{fi}] * 100%

Total stream recovery:
R_c (%) = [U_c / U_f] * 100%
R_t (%) = [U_t / U_f] * 100%

Feed distribution represents the % of the total gold present in that particular size class:
D_i (%) = [U_{fi} / U_f] * 100%

Concentrate total recovery per size class is calculated via:
$TotR_{ci}$ (%) = [$R_{ci} * D_i$] / 100%

Cumulative concentrate recovery represents "total GRG recovery coarser than":
$CumR_{ci}$ (%) = Σ $TotR_{ci}$ for i = 1 to j,
where 1 represents the coarsest size class calculated and j represents the size class of interest.

Note: j must be less than or equal to the coarsest size class in order for the calculation to work, e.g. if the coarsest size class is 600 μm, there is no such thing as a cumulative recovery coarser than 850 μm since there is no data for this size class; however cumulative recovery coarser than 300 μm would be the summation of the total recoveries for the 300-425, 425-600, and 600+ μm size classes.

8.3 2-stage overall calculations

The overall recovery sheet is somewhat different than its 1-stage predecessor. It is used to assess the contributions of each stage's recovery to the overall GRG test recovery. The first two columns of each stage is a copy of the individual stage recovery data (R_{ci} and D_i), followed by individual recoveries in g/t. Since stage 3 assays are taken as the best assessment of gold losses, the stage 3 calculations differ slightly from those of the previous two stages. The calculations for stage 1 will be presented first, followed by those for stage 3 and how they tie together to provide the overall GRG data.

Step 1) Stage 1 data

Stage 1 size-by-size recoveries in g/t:
$R_{1i}\ (g/t) = [\ R_1c_i\ (\%) * D_{1i}\ (\%)\ ] * f_{1i}\ (g/t) / 10^4\ (\%^2)$
Please note that the subscript 1 was inserted to indicate which stage's values are to be used.

Then the total stage 1 recovery in g/t:
$TotR_1\ (g/t) = \Sigma\ R_{1i}$

Step 2) Stage 3 data

Stage 3 size-by-size recoveries in g/t:
$R_{3i}\ (g/t) = [\ R_3c_i\ (\%) * D_{3i}\ (\%) * \{\ 100\% - Y_1\ (\%)\ \}\ ] / 10^6\ (\%)$
Note that the additional term $(1 - Y)$ is used to correct the total mass of stage 3 since the mass of concentrate removed in stage 1, although small, is not negligible.

Then total stage 3 recovery in g/t:
$TotR_3\ (g/t) = \Sigma\ R_{3i}$

Next the size-by-size losses in g/t:
$L_{3i}\ (g/t) = [\ R_{3i}\ (g/t) * \{\ 100\% - R_{3ci}\ (\%)\}\ ] / R_{3ci}\ (\%)$

The total losses in g/t:
$TotL_3\ (g/t) = \Sigma\ L_{3i}$

Step 3) Calculate the overall feed grade, total recovery, and overall recoveries per stage

Overall feed grade (g/t):
$f\ (g/t) = TotR_1 + TotR_3 + TotL_3$

Stage 1 Overall & total recovery is calculated via:
$OAR_1\ (\%) = [\ TotR_1 / f\ ] * 100\%$
$TotR_1\ (\%) = [\ TotR_1\ (g/t) / f\ (g/t)\ ] * 100\%$

Stage 3 Overall & total recovery:
OAR_3 (%) = [$TotR_3$ / f] * 100%
$TotR_3$ (%) = 100% * $TotR_3$ (g/t) / [f (g/t) – $TotR_1$ (g/t)]
The divisor represents the total amount of gold that was available for recovery in stage 3 after stage 1.

Size-by-size total recovery (g/t):
$TotR_{ci}$ (g/t) = $TotR_{1ci}$ + $TotR_3c_i$
$TotR_{ci}$ (%) = [$TotR_{ci}$ (g/t) / f (g/t)] * 100%

Cumulative recovery is calculated from the size-by-size total recoveries the same as before.

8.4 *Plotting GRG content:*

The main use for the cumulative recovery data is the generation of a GRG curve such as the one depicted below.

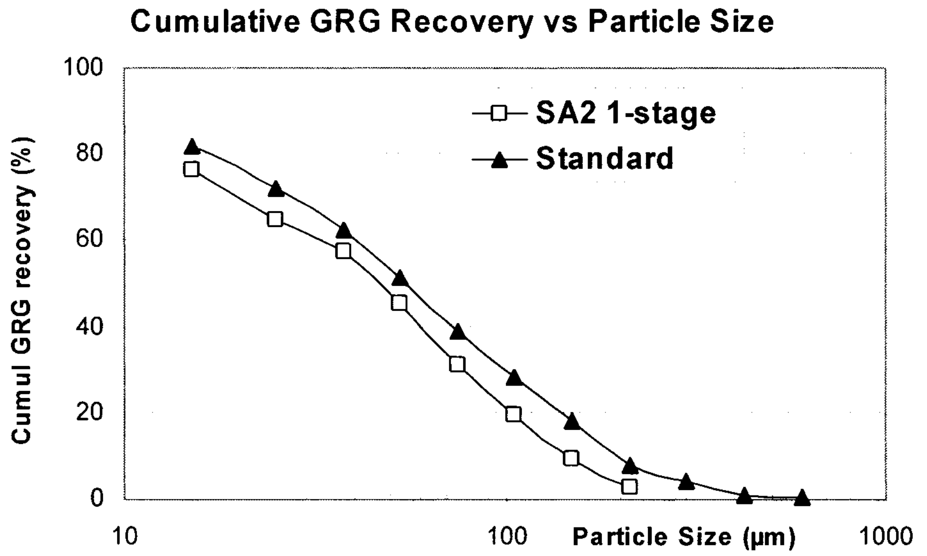

Figure 60: Cumulative GRG curve

The GRG plot allows the reader to develop an intuitive understanding of the GRG's size distribution. The slope of each segment describes the amount of GRG present in the respective size class: the flatter the segment, the less GRG in that size class; the more inclined, the more GRG. The intersection between the curve and the x-intercept gives the cumulative % GRG coarser than size x (µm); the cumulative recovery by the finest size class represents the sample's total GRG content.

9 Appendix B: Experimental Data

NV, Stage 1, 7.1 L/min, 4.0 psi (4.6), 53 min, 1020 g/min CORRECTED

File = Simple NV.xls

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov (%)	Cumul Recov (%)
850	0.00	0.00	0.0	0	0.00	0	0.00	0.00	0	0.00	0	0.00	0.00	0	0.00	0.00	0.00
600	22.08	25.10	12480	275548	98.78	6782	12.60	0.50	3391	1.22	6804	12.62	41.00	278939	44.46	43.92	43.92
425	17.54	19.94	4532	79499.4	88.29	8114	15.07	1.30	10548	11.71	8132	15.08	11.07	90048	14.35	12.67	56.59
300	16.10	18.30	2661	42848.1	88.26	6335	11.77	0.90	5702	11.74	6351	11.78	7.64	48550	7.74	6.83	63.42
212	10.80	12.28	1563	16877.2	66.26	5544	10.30	1.55	8594	33.74	5555	10.30	4.59	25471	4.06	2.69	66.11
150	9.50	10.80	1519	14427.5	80.03	4000	7.43	0.90	3600	19.97	4009	7.43	4.50	18027	2.87	2.30	68.41
106	5.47	6.22	3461	18930.2	81.22	3243	6.02	1.35	4378	18.78	3248	6.02	7.18	23308	3.71	3.02	71.42
75	3.46	3.93	7963	27550.9	84.25	2575	4.78	2.00	5150	15.75	2578	4.78	12.68	32701	5.21	4.39	75.81
53	1.55	1.76	10063	15598.2	61.17	3001	5.57	3.30	9903	38.83	3002	5.57	8.49	25501	4.06	2.49	78.30
37	0.86	0.98	12598	10833.9	63.20	1589	2.95	3.97	6307	36.80	1590	2.95	10.78	17141	2.73	1.73	80.03
25	0.381	0.43	16578	6322.99	40.34	1719	3.19	5.44	9350	59.66	1719	3.19	9.12	15673	2.50	1.01	81.03
20	0.074	0.08	24903	1837.86	20.89	989	1.84	7.04	6962	79.11	989	1.83	8.90	8800	1.40	0.29	81.33
15	0.159	0.18	12457	1978.19	4.57	9952	18.48	4.15	41301	95.43	9952	18.45	4.35	43279	6.90	0.32	81.64
Total	87.97	100.00	5823	512253	81.64	53842	100.00	2.14	115185	18.36	53930	100.00	11.63	627437	100.00	81.64	

Table 22: NV Stage 1 mass balance

NV, Stage 3, 5.0 L/min, 2.9 psi (3.7), 51:40 min, 300 g/min

File = Simple NV.xls %-75µm: 87.86 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov (%)	Cumul Recov (%)
106	9.41	9.71	346.9	3264	96.44	317	1.77	0.38	120	3.56	326	1.81	10.37	3385	8.92	8.60	8.60
75	36.86	38.04	55.4	2043	69.16	1823	10.18	0.50	911	30.84	1859	10.33	1.59	2955	7.79	5.39	13.99
53	24.68	25.47	84.9	2095	56.79	3188	17.81	0.50	1594	43.21	3212	17.85	1.15	3689	9.72	5.52	19.51
37	14.04	14.49	190.7	2677	71.42	1429	7.98	0.75	1071	28.58	1443	8.01	2.60	3749	9.88	7.06	26.57
25	7.60	7.84	429.0	3260	66.96	1462	8.17	1.10	1608	33.04	1470	8.17	3.31	4868	12.83	8.59	35.16
15	4.30	4.44	1052.3	4525	23.45	9685	54.10	1.53	14770	76.55	9689	53.83	1.99	19295	50.86	11.93	47.09
Total	96.89	100.00	184.4	17865	47.09	17903	100.00	1.12	20075	52.91	18000	100.00	2.11	37940	100.00	47.09	

Table 23: NV Stage 3 mass balance (1-Stage)

QC1.xls, Stage 1, 6.9 L/min, 3.9 psi (4.1), 35:33 min, 609.6 g/min

File = Simple QC1.xls %-75µm: 35.39 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)
850	0.00	0.00	0.0	0	0.00	0	0	0	0	0.00	0	0.00	0.00	0	0.00
600	17.90	22.63	1.6	28	3.00	3002	13.51	0.30	901	97.00	3020	13.54	0.31	928	7.74
425	16.40	20.73	2.1	35	2.59	3282	14.77	0.40	1313	97.41	3299	14.79	0.41	1348	11.23
300	14.30	18.08	39.0	558	37.43	2334	10.50	0.40	933	62.57	2348	10.53	0.64	1492	12.43
212	8.50	10.75	36.6	311	23.53	1837	8.27	0.55	1010	76.47	1846	8.28	0.72	1321	11.01
150	7.20	9.10	36.8	265	29.84	1247	5.61	0.50	624	70.16	1254	5.62	0.71	889	7.41
106	4.50	5.69	33.0	149	22.54	1277	5.75	0.40	511	77.46	1282	5.75	0.51	659	5.49
75	3.60	4.55	64.8	233	40.78	1355	6.10	0.25	339	59.22	1359	6.09	0.42	572	4.77
53	2.70	3.41	59.9	162	29.09	1972	8.87	0.20	394	70.91	1974	8.85	0.28	556	4.63
37	1.50	1.90	31.9	48	9.06	1919	8.64	0.25	480	90.94	1921	8.61	0.27	528	4.40
25	1.90	2.40	82.5	157	10.68	1456	6.55	0.90	1311	89.32	1458	6.54	1.01	1467	12.23
15	0.60	0.76	137.7	83	3.69	2539	11.43	0.85	2158	96.31	2540	11.39	0.88	2241	18.67
Total	79.10	100.00	25.6	2028	16.90	22221	100.00	0.45	9974	83.10	22300	100.00	0.54	12002	100.00

Table 24: QC1 Stage 1 mass balance (2-Stage)

QC1.xls, Stage 3, 3.37 L/min, 1.8 psi (2.6), 52:02 min, 356.4 g/min

File = Simple QC1.xls %-75µm: 80.28 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)
150	7.98	9.91	23.2	185	55.06	256	1.48	0.59	151	44.94	264	1.52	1.27	336	4.50
106	21.12	26.22	11.9	251	37.92	1175	6.77	0.35	411	62.08	1196	6.86	0.55	662	8.87
75	21.22	26.34	17.7	376	32.44	1958	11.28	0.40	783	67.56	1979	11.35	0.59	1159	15.52
53	12.82	15.92	23.1	153	14.66	2959	17.04	0.30	888	85.34	2972	17.04	0.35	1040	13.92
37	9.18	11.40	33.7	310	32.71	3185	18.35	0.20	637	67.29	3195	18.32	0.30	947	12.67
25	5.70	7.08	59.5	339	39.68	2580	14.86	0.20	516	60.32	2586	14.83	0.33	855	11.45
15	2.53	3.14	250.6	634	25.67	5246	30.22	0.35	1836	74.33	5249	30.09	0.47	2470	33.07
Total	80.55	100.00	27.9	2248	30.09	17359	100.00	0.30	5222	69.91	17440	100.00	0.43	7471	100.00

Table 25: QC1 Stage 3 mass balance (2-Stage)

QC1, Stage 3 Direct, 3.944 L/min, 2.0 psi (2.7), 56:12 min, 336.8 g/min

File = Simple QC1.xls %-75μm: 77.99 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul. Recov. (%)
150	0	0.00	0.0	0	0.00	0	0.00	0.00	0	0.00	0	0.00	0.00	0	0.00	0.00	0.00
106	24.70	33.74	38.9	961	64.14	1791	8.44	0.30	537	35.86	1815	8.53	0.83	1498	15.16	9.72	9.72
75	19.90	27.19	25.4	505	33.61	2849	13.44	0.35	997	66.39	2869	13.48	0.52	1502	15.20	5.11	14.83
53	11.90	16.26	37.9	463	28.30	3909	18.44	0.30	1173	71.70	3921	18.43	0.42	1636	16.55	4.69	19.52
37	4.80	6.56	18.8	90	13.96	3706	17.48	0.15	556	86.04	3711	17.44	0.17	646	6.54	0.91	20.43
25	9.40	12.84	104.1	978	69.65	2842	13.40	0.15	426	30.35	2851	13.40	0.49	1404	14.21	9.90	30.33
15	2.50	3.42	386.4	966	30.23	6109	28.81	0.37	2230	69.77	6111	28.72	0.52	3196	32.34	9.77	40.10
Total	73.20	100.00	54.1	3963	40.10	21205	100.00	0.28	5919	59.90	21278	100.00	0.46	9882	100.00		

Table 26: QC1 Stage 3 mass balance (1-Stage)

GU1 A, Stage 3 Direct, 2.901 L/min, 1.4 psi (1.8), 64:24 min, 186.8 g/min

File = GU1.xls %-75μm: 90.44 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul. Recov. (%)
150	29.04	43.30	58.7	1704	94.22	348	1.78	0.30	104	5.78	377	1.92	4.79	1808	10.23	9.64	9.64
106	4.43	6.61	232.7	1031	91.16	294	1.50	0.34	100	8.84	299	1.52	3.79	1131	6.40	5.83	15.47
75	7.94	11.84	156.1	1240	63.36	1195	6.10	0.60	717	36.64	1203	6.12	1.63	1957	11.07	7.01	22.48
53	8.92	13.30	123.7	2076	77.28	3052	15.57	0.20	610	22.72	3061	15.57	0.88	2686	15.19	11.74	34.22
37	7.36	10.97	128.1	943	74.01	2208	11.27	0.15	331	25.99	2215	11.27	0.58	1274	7.21	5.33	39.55
25	4.69	6.99	161.5	757	70.44	3179	16.22	0.10	318	29.56	3183	16.19	0.34	1075	6.08	4.28	43.84
20	2.04	3.04	243.4	497	78.06	930	4.75	0.15	140	21.94	932	4.74	0.68	636	3.60	2.81	46.65
15	2.65	3.95	388.9	1031	14.49	8389	42.81	0.73	6082	85.51	8391	42.68	0.85	7112	40.23	5.83	52.48
Total	67.07	100.00	138.3	9278	52.48	19595	100.00	0.43	8402	47.52	19662	100.00	0.90	17680	100.00		

Table 27: GU1A Stage 3 mass balance (1-Stage)

GU1 B, Stage 3 Direct, 5.028 L/min, 2.6 psi (2.8), 90:00 min, 211.8 g/min

File = GU1.xls %-75µm: 90.68 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul. Recov. (%)
150																0.00	0.00
106	12.90	29.09	292.9	3778	96.70	415	2.15	0.31	129	3.30	428	2.22	9.12	3907	21.19	20.49	20.49
75	12.71	28.66	77.5	985	70.71	1361	7.06	0.30	408	29.29	1373	7.11	1.01	1393	7.56	5.34	25.84
53	7.48	16.87	135.3	2191	77.39	2560	13.28	0.25	640	22.61	2567	13.28	1.10	2831	15.35	11.88	37.72
37	5.21	11.75	173.9	906	75.48	1962	10.18	0.15	294	24.52	1967	10.18	0.61	1200	6.51	4.92	42.63
25	3.01	6.79	242.6	730	69.69	3176	16.47	0.10	318	30.31	3179	16.45	0.33	1048	5.68	3.96	46.59
20	1.38	3.12	351.6	486	82.15	1056	5.48	0.10	106	17.85	1057	5.47	0.56	592	3.21	2.64	49.23
15	1.65	3.72	547.1	902	12.08	8751	45.39	0.75	6563	87.92	8753	45.29	0.85	7465	40.49	4.89	54.12
Total	44.34	100.00	225.0	9978	54.12	19281	100.00	0.44	8458	45.88	19325	100.00	0.95	18436	100.00		

Table 28: GU1B Stage 3 mass balance (1-Stage)

GU2 A, Stage 1, 3.98 L/min, 2.1 psi (2.5), 23:44 min, 806.6 g/min

File = GU2.xls %-75µm: 34.87 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)
850	0.00	0.00	0.0	0	0.00	0	0	0	0	0.00	0	0.00	0.00	0	0.00
600	14.09	18.77	79.0	1114	42.58	2311	11.44	0.65	1502	57.42	2325	11.47	1.13	2615	4.35
425	13.56	18.06	175.3	2378	40.39	2600	12.87	1.35	3510	59.61	2613	12.89	2.25	5887	9.79
300	11.37	15.14	345.6	3930	48.36	1907	9.44	2.20	4196	51.64	1919	9.46	4.24	8126	13.51
212	9.26	12.33	339.9	3148	62.29	1906	9.44	1.00	1906	37.71	1915	9.45	2.64	5053	8.40
150	7.98	10.63	311.4	2485	63.41	1594	7.89	0.90	1434	36.59	1602	7.90	2.45	3920	6.52
106	6.78	9.03	460.7	3123	71.94	1433	7.10	0.85	1218	28.06	1440	7.10	3.02	4341	7.22
75	4.84	6.45	552.7	2675	79.43	1385	6.86	0.50	693	20.57	1390	6.86	2.42	3368	5.60
53	3.72	4.95	864.9	3217	80.35	1967	9.74	0.40	787	19.65	1971	9.72	2.03	4004	6.66
37	2.18	2.90	1491.4	3251	91.49	1008	4.99	0.30	302	8.51	1010	4.98	3.52	3554	5.91
25	0.80	1.07	4242.9	3394	86.55	1055	5.22	0.50	527	13.45	1056	5.21	3.72	3922	6.52
15	0.50	0.67	14075.6	7038	45.78	3030	15.01	2.75	8334	54.22	3031	14.95	5.07	15372	25.55
Total	75.08	100.00	476.2	35753	59.43	20196	100.00	1.21	24409	40.57	20271	100.00	2.97	60162	100.00

Table 29: GU2A Stage 1 mass balance (2-Stage)

GU2 A, Stage 3, 2.472 L/min, 1.2 psi (1.6), 96:00 min, 245.0 g/min

File = GU2.xls %-75µm: 80.26 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)
150	0	0.00		0	0.00	0	0.00	0.00	0	0.00	0	0.00	0.00	0	0.00
106	32.24	35.73	50.5	1627	90.03	1201	6.62	0.15	180	9.97	1234	6.77	1.46	1807	9.59
75	19.88	22.03	66.4	1320	78.95	2346	12.93	0.15	352	21.05	2365	12.97	0.71	1671	8.87
53	17.66	19.57	79.6	891	56.02	4663	25.70	0.15	699	43.98	4680	25.67	0.34	1590	8.44
37	11.21	12.42	138.0	1547	81.63	2321	12.79	0.15	348	18.37	2333	12.79	0.81	1896	10.06
25	5.20	5.76	248.2	1291	77.83	2451	13.51	0.15	368	22.17	2456	13.47	0.68	1658	8.80
15	4.04	4.48	901.8	3643	35.62	5163	28.45	1.28	6583	64.38	5167	28.34	1.98	10226	54.25
Total	90.23	100.00	114.4	10318	54.74	18145	100.00	0.47	8530	45.26	18235	100.00	1.03	18848	100.00

Table 30: GU2A Stage 3 mass balance (2-Stage)

GU2 B, Stage 3 Direct, 2.720 L/min, 1.3 psi (1.7), 107:05 min, 241.7 g/min

File = GU2.xls %-75µm: 81.71 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul. Recov. (%)
150	0	0.00	0.0	0	0.00	0	0.00	0.00	0	0.00	0	0.00	0.00	0	0.00	0.00	0.00
106	14.02	15.00	717.5	10060	98.46	631	3.36	0.25	158	1.54	645	3.42	15.84	10217	15.84	15.60	15.60
75	32.64	34.92	127.0	4146	88.21	2772	14.77	0.20	554	11.79	2805	14.87	1.68	4700	7.29	6.43	22.02
53	23.34	24.97	197.5	16747	94.59	4787	25.50	0.20	957	5.41	4810	25.50	3.68	17704	27.45	25.96	47.98
37	12.43	13.30	405.2	5036	94.95	1785	9.51	0.15	268	5.05	1797	9.53	2.95	5304	8.22	7.81	55.79
25	6.98	7.47	802.9	5604	88.90	2798	14.91	0.25	700	11.10	2805	14.87	2.25	6304	9.77	8.69	64.48
15	4.07	4.35	2551.0	10383	51.21	5995	31.94	1.65	9892	48.79	5999	31.81	3.38	20275	31.43	16.10	80.58
Total	93.48	100.00	556.0	51975	80.58	18769	100.00	0.67	12529	19.42	18862	100.00	3.42	64504	100.00		

Table 31: GU2B Stage 3 mass balance (1-Stage)

ME, Stage 1, 4.956 L/min, 2.4 psi (3.6 psi), 29 min, 761.6 g/min

File = Simple ME.xls %-75µm: **29.42** %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	36.97	20.19	20.1	744	10.51	2162	10.60	2.93	6334	89.49	2199	10.69	3.22	7078	10.94	1.15	1.15
425	40.22	21.97	8.5	343	2.82	2215	10.86	5.33	11807	97.18	2255	10.96	5.39	12149	18.78	0.53	1.68
300	28.75	15.70	9.0	259	3.93	1828	8.97	3.47	6344	96.07	1857	9.03	3.56	6603	10.21	0.40	2.08
212	21.29	11.63	50.1	1067	17.58	2083	10.22	2.40	5000	82.42	2105	10.23	2.88	6066	9.38	1.65	3.73
150	16.65	9.09	79.0	1316	15.09	2017	9.89	3.67	7403	84.91	2034	9.89	4.29	8718	13.47	2.03	5.76
106	12.78	6.98	111.9	1429	23.30	2073	10.17	2.27	4707	76.70	2086	10.14	2.94	6136	9.48	2.21	7.97
75	9.65	5.27	118.8	1146	27.51	1975	9.68	1.53	3021	72.49	1984	9.65	2.10	4168	6.44	1.77	9.74
53	7.42	4.05	136.9	1016	21.45	2227	10.92	1.67	3719	78.55	2234	10.86	2.12	4735	7.32	1.57	11.31
37	4.84	2.64	126.1	610	28.45	1003	4.92	1.53	1535	71.55	1008	4.90	2.13	2145	3.32	0.94	12.26
25	2.51	1.37	96.3	242	13.90	832	4.08	1.80	1498	86.10	834	4.06	2.08	1739	2.69	0.37	12.63
15	2.01	1.10	49.7	100	1.94	1974	9.68	2.57	5063	98.06	1976	9.60	2.61	5163	7.98	0.15	12.78
Total	183.09	100.00	45.2	8272	12.78	20390	100.00	2.77	56430	87.22	20573	100.00	3.14	64702	100.00	12.78	

Table 32: ME Stage 1 mass balance (2-Stage)

ME, Stage 3, 4.610 L/min, 2.4 psi (3.8 psi), 57 min, 316.1 g/min

File = Simple ME.xls **%-75µm:** 70.84 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	0.18	0.12	5.56	1	29.71	1	0.01	1.96	2	70.29	1	0.01	2.43	3	0.01	0.00	0.00
425	3.62	2.39	36.46	132	94.10	4	0.02	1.96	8	5.90	8	0.04	17.88	140	0.30	0.28	0.28
300	18.48	12.20	20.81	385	76.21	61	0.34	1.96	120	23.79	80	0.44	6.33	505	1.08	0.82	1.11
212	26.74	17.65	39.92	1067	65.14	291	1.61	1.96	571	34.86	318	1.74	5.15	1639	3.51	2.29	3.39
150	13.86	9.15	149.04	2066	71.55	401	2.21	2.05	821	28.45	415	2.27	6.96	2887	6.18	4.42	7.82
106	12.65	8.35	264.23	3343	64.31	1262	6.97	1.47	1855	35.69	1274	6.98	4.08	5197	11.13	7.16	14.98
75	19.63	12.96	182.62	3585	44.37	3211	17.73	1.40	4495	55.63	3231	17.69	2.50	8080	17.30	7.68	22.65
53	22.39	14.78	151.54	3393	41.44	4243	23.42	1.13	4794	58.56	4265	23.35	1.92	8187	17.53	7.27	29.92
37	16.65	10.99	129.96	2164	42.42	1920	10.60	1.53	2937	57.58	1937	10.60	2.63	5101	10.92	4.63	34.55
25	8.93	5.89	122.83	1097	28.43	1596	8.81	1.73	2762	71.57	1605	8.79	2.40	3859	8.26	2.35	36.90
15	8.37	5.52	101.81	852	7.68	5124	28.28	2.00	10247	92.32	5132	28.10	2.16	11099	23.77	1.82	38.72
Total	151.50	100.00	119.37	18084	38.72	18115	100.00	1.58	28615	61.28	18266	100.00	2.56	46699	100.00	38.72	

Table 33: ME Stage 3 mass balance (2-Stage)

ME, Stage 3 Direct, 4.288 L/min, 2.3 psi (3.6 psi), 66 min, 321.3 g/min

File = Simple ME.xls %-75µm: 80.04 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	6.43	3.94	16.1	103	70.97	12	0.06	3.40	42	29.03	19	0.10	7.72	146	0.26	0.18	0.18
425	16.37	10.04	13.1	214	45.18	77	0.39	3.40	260	54.82	93	0.47	5.11	475	0.83	0.38	0.56
300	20.60	12.63	24.2	498	56.47	113	0.58	3.40	384	43.53	133	0.68	6.61	881	1.55	0.87	1.43
212	17.39	10.66	59.9	1041	65.24	163	0.84	3.40	555	34.76	181	0.92	8.84	1596	2.80	1.83	3.26
150	10.76	6.60	209.2	2251	75.39	216	1.11	3.40	735	24.61	227	1.16	13.16	2986	5.24	3.95	7.20
106	9.67	5.93	506.4	4897	78.58	741	3.81	1.80	1335	21.42	751	3.83	8.30	6232	10.93	8.59	15.79
75	16.54	10.14	309.7	5122	63.09	2497	12.83	1.20	2997	36.91	2514	12.81	3.23	8119	14.23	8.98	24.77
53	23.89	14.65	200.4	4787	41.59	4574	23.50	1.47	6724	58.41	4598	23.42	2.50	11511	20.18	8.39	33.16
37	19.43	11.91	153.2	2977	49.70	2152	11.06	1.40	3013	50.30	2172	11.06	2.76	5990	10.50	5.22	38.38
25	11.66	7.15	127.7	1489	34.72	1905	9.79	1.47	2800	65.28	1917	9.76	2.24	4289	7.52	2.61	40.99
15	10.38	6.36	99.0	1028	6.94	7015	36.04	1.97	13784	93.06	7025	35.79	2.11	14812	25.97	1.80	42.79
Total	163.12	100.00	149.6	24408	42.79	19466	100.00	1.68	32629	57.21	19629	100.00	2.91	57037	100.00	42.79	

Table 34: ME Stage 3 mass balance (1-Stage)

GH.xls Stage 1, 5.490 L/min, 3.0 psi (3.4 psi), 24:40 min, 859.3 g/min

File = Simple GH.xls %-75μm: 19.88 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	69.44	34.07	4.1	284	2.84	3527	15.59	2.75	9698	97.16	3596	15.75	2.78	9982	16.09	0.46	0.46
425	60.12	29.50	6.8	408	4.11	4048	17.89	2.35	9512	95.89	4108	17.99	2.41	9920	15.99	0.66	1.11
300	31.89	15.65	15.6	499	7.19	2993	13.23	2.15	6435	92.81	3025	13.25	2.29	6934	11.18	0.80	1.92
212	17.72	8.69	34.1	605	9.21	2773	12.26	2.15	5962	90.79	2791	12.22	2.35	6567	10.59	0.98	2.89
150	10.10	4.96	35.1	355	7.09	1980	8.75	2.35	4652	92.91	1990	8.72	2.52	5007	8.07	0.57	3.47
106	6.12	3.00	34.4	211	4.66	1541	6.81	2.80	4314	95.34	1547	6.77	2.93	4524	7.29	0.34	3.81
75	3.55	1.74	65.5	233	6.42	1232	5.45	2.75	3389	93.58	1236	5.41	2.93	3622	5.84	0.38	4.18
53	2.38	1.17	103.8	247	6.57	1232	5.45	2.85	3512	93.43	1235	5.41	3.04	3759	6.06	0.40	4.58
37	1.24	0.61	297.5	369	17.48	603	2.66	2.89	1742	82.52	604	2.65	3.49	2111	3.40	0.59	5.17
25	0.55	0.27	589.1	324	17.60	468	2.07	3.24	1517	82.40	469	2.05	3.93	1841	2.97	0.52	5.70
15	0.72	0.35	1083.1	780	10.05	2230	9.85	3.13	6979	89.95	2230	9.77	3.48	7758	12.51	1.26	6.95
Total	203.83	100.00	21.2	4313	6.95	22626	100.00	2.55	57713	93.05	22830	100.00	2.72	62026	100.00	6.95	

Table 35: GH Stage 1 mass balance (2-Stage)

GH.xls Stage 3, 4.986 L/min, 2.4 psi (3.2 psi), 50 min, 362.4 g/min

File = Simple GH.xls .xls) %-75μm: 70.92 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.26	0.31	13.04	3	36.53	5	0.03	1.13	6	63.47	5	0.03	1.70	9	0.02	0.01	0.01
150	2.22	2.62	41.82	93	37.12	139	0.71	1.13	157	62.88	141	0.72	1.77	250	0.49	0.18	0.19
106	15.93	18.78	64.17	1022	33.60	1683	8.64	1.20	2020	66.40	1699	8.69	1.79	3042	5.97	2.00	2.19
75	29.98	35.35	76.86	2304	27.41	3814	19.58	1.60	6102	72.59	3844	19.65	2.19	8407	16.48	4.52	6.71
53	21.26	25.07	77.96	1657	16.98	4051	20.79	2.00	8101	83.02	4072	20.81	2.40	9758	19.13	3.25	9.96
37	9.24	10.89	111.43	1030	19.40	1901	9.76	2.25	4277	80.60	1910	9.76	2.78	5306	10.40	2.02	11.98
25	3.30	3.89	273.91	904	20.08	1468	7.54	2.45	3597	79.92	1472	7.52	3.06	4501	8.83	1.77	13.75
15	2.62	3.09	1280.48	3355	17.01	6420	32.95	2.55	16371	82.99	6422	32.82	3.07	19726	38.68	6.58	20.33
Total	84.81	100.00	122.26	10368	20.33	19481	100.00	2.09	40631	79.67	19566	100.00	2.61	51000	100.00	20.33	

Table 36: GH Stage 3 mass balance (2-Stage)

GH.xls **Stage 3 Direct, 5.118 L/min, 2.4 psi (3.6 psi), 53 min, 302.3 g/min**

File = Simple GH.xls %-75μm: 70.96 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
300	0.26	0.31	4.3	1	10.16	7	0.04	1.43	10	89.84	7	0.04	1.53	11	0.02	0.00	0.00
212	0.15	0.18	100.0	15	54.78	9	0.04	1.43	12	45.22	9	0.04	3.11	27	0.05	0.03	0.03
150	2.13	2.53	85.4	182	39.48	195	0.98	1.43	279	60.52	197	0.99	2.34	461	0.81	0.32	0.35
106	14.39	17.11	80.0	1151	34.00	1656	8.31	1.35	2235	66.00	1670	8.35	2.03	3386	5.92	2.01	2.36
75	27.90	33.18	81.3	2269	24.96	3898	19.57	1.75	6821	75.04	3926	19.62	2.32	9090	15.88	3.96	6.32
53	22.59	26.86	88.4	1996	16.83	4587	23.03	2.15	9863	83.17	4610	23.05	2.57	11859	20.72	3.49	9.81
37	10.38	12.34	130.8	1358	21.30	2048	10.28	2.45	5017	78.70	2058	10.29	3.10	6375	11.14	2.37	12.18
25	3.64	4.33	308.9	1124	21.68	1504	7.55	2.70	4061	78.32	1508	7.54	3.44	5186	9.06	1.96	14.15
15	2.65	3.15	1393.8	3694	17.72	6016	30.20	2.85	17146	82.28	6019	30.09	3.46	20840	36.41	6.45	20.60
Total	84.09	100.00	140.2	11790	20.60	19920	100.00	2.28	45445	79.40	20004	100.00	2.86	57236	100.00	20.60	

Table 37: GH Stage 3 mass balance (1-Stage)

GH.xls Composite, Overall Results

Size (µm)	First Stage: 100% -850 µm Stage Recov.	Dist'n	Rec. g/t	Second Stage: 54.1% -75 µm Stage Recov.	Dist'n	Rec. g/t	Third Stage: 82.3% -75 µm Stage Recov.	Dist'n	Rec. g/t	Losses g/t	Total Recov. g/t	Total Recov. %	Cumul Recov. %	
850														
600	2.42	11.80	0.008								0.008	0.3	0.29	600
425	2.09	16.76	0.010	0.00	0.00	0.000					0.010	0.4	0.68	425
300	4.46	11.44	0.015	23.20	2.80	0.018					0.033	1.2	1.91	300
212	3.81	10.29	0.011	22.07	7.50	0.044	0.00	0.00	0.00	0.00	0.056	2.1	3.97	212
150	3.27	8.33	0.007	26.62	12.45	0.093	2.18	0.10	0.00	0.00	0.100	3.7	7.67	150
106	2.67	7.39	0.005	20.82	12.55	0.070	25.06	3.42	0.02	0.05	0.096	3.6	11.23	106
75	3.58	5.26	0.005	13.39	10.16	0.036	19.51	10.20	0.05	0.17	0.089	3.3	14.54	75
53	7.45	5.07	0.010	10.45	9.80	0.028	13.86	14.10	0.05	0.26	0.085	3.1	17.67	53
37	10.45	3.67	0.010	13.15	6.97	0.024	11.39	10.33	0.03	0.21	0.061	2.3	19.94	37
25	26.11	1.92	0.012	20.28	5.54	0.028	17.14	8.77	0.03	0.16	0.073	2.7	22.63	25
20	22.50	1.29	0.007	25.46	2.93	0.017	26.91	4.01	0.02	0.06	0.048	1.8	30.42	15
15	2.45	16.79	0.011	6.17	29.31	0.047	10.10	49.07	0.10	0.97	0.162	6.0		
Total	4.2	100.0	0.113	15.6	100.0	0.40	13.9	100.0	0.3	1.9	0.82	30.4		
O/A	4.2			15.0			11.3							
Yield	0.00181			0.00368			0.00285							
Grade	2.78 g/t			2.60 g/t			2.20 g/t							
Calc.:	2.70 g/t													

Table 38: GH standard composite balance

AU1.xls **Stage 3 Direct, 4.900 L/min, 2.4 psi (2.8 psi), 64:40 min, 338.2 g/min**
File: Simple AU1.xls **%-75μm:** 78.37 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.28	0.49	2462.1	689	99.25	7	0.03	0.80	5	0.75	7	0.03	101.94	695	2.71	2.69	2.69
150	2.50	4.35	207.2	518	83.43	128	0.62	0.80	103	16.57	131	0.63	4.75	621	2.43	2.02	4.72
106	10.02	17.44	70.6	707	41.59	1241	6.03	0.80	993	58.41	1251	6.06	1.36	1700	6.64	2.76	7.48
75	14.49	25.22	67.6	980	29.92	3060	14.87	0.75	2295	70.08	3075	14.90	1.07	3275	12.79	3.83	11.31
53	12.23	21.29	96.6	1181	29.38	4056	19.71	0.70	2839	70.62	4068	19.71	0.99	4020	15.70	4.61	15.92
37	8.38	14.59	161.9	1357	49.46	2520	12.24	0.55	1386	50.54	2528	12.25	1.08	2743	10.71	5.30	21.22
25	4.55	7.92	257.6	1172	50.82	2268	11.02	0.50	1134	49.18	2273	11.01	1.01	2306	9.01	4.58	25.80
15	5.00	8.70	734.0	3670	35.84	7301	35.47	0.90	6571	64.16	7306	35.40	1.40	10241	40.00	14.34	40.13
Total	57.45	100.00	178.8	10274	40.13	20582	100.00	0.74	15326	59.87	20639	100.00	1.24	25601	100.00	40.13	

Table 39: AU1 Stage 3 mass balance (1-Stage)

AU2.xls **Stage 3 Direct, 4.842 L/min, 2.4 psi (2.8 psi), 56:00 min, 336.3 g/min**
File = Simple AU2.xls **%-75μm:** 75.60 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.33	0.56	6.3	2	23.64	16	0.08	0.42	7	76.36	16	0.08	0.54	9	0.06	0.01	0.01
150	4.26	7.27	37.1	158	62.45	225	1.12	0.42	95	37.55	230	1.14	1.10	253	1.60	1.00	1.01
106	12.54	21.41	18.2	228	23.60	1643	8.16	0.45	740	76.40	1656	8.20	0.58	968	6.13	1.45	2.46
75	14.05	23.99	30.1	423	21.93	3010	14.95	0.50	1505	78.07	3024	14.98	0.64	1928	12.20	2.68	5.13
53	10.73	18.32	47.4	509	16.96	3835	19.05	0.65	2493	83.04	3846	19.05	0.78	3002	19.00	3.22	8.36
37	7.64	13.05	75.4	576	37.63	2387	11.86	0.40	955	62.37	2395	11.86	0.64	1531	9.69	3.65	12.00
25	4.50	7.68	116.8	526	37.38	2201	10.93	0.40	880	62.62	2205	10.92	0.64	1406	8.90	3.33	15.33
15	4.51	7.70	353.0	1592	23.75	6815	33.85	0.75	5111	76.25	6819	33.77	0.98	6703	42.43	10.08	25.41
Total	58.56	100.00	68.5	4014	25.41	20132	100.00	0.59	11785	74.59	20191	100.00	0.78	15799	100.00	25.41	

Table 40: AU2 Stage 3 mass balance (1-Stage)

AU3.xls Stage 3 Direct, 4.636 L/min, 2.4 psi (2.9 psi), 60:11 min, 348.3 g/min

File = Simple AU3.xls %-75μm: 76.57 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	1.04	1.59	491.4	511	97.79	28	0.13	0.42	12	2.21	29	0.14	18.23	523	4.59	4.49	4.49
150	7.59	11.61	16.4	124	43.38	388	1.88	0.42	162	56.62	396	1.91	0.72	286	2.51	1.09	5.58
106	11.67	17.86	18.5	216	24.67	1466	7.10	0.45	660	75.33	1477	7.14	0.59	876	7.68	1.90	7.47
75	12.79	19.57	17.9	229	18.21	2936	14.23	0.35	1028	81.79	2949	14.25	0.43	1256	11.03	2.01	9.48
53	12.42	19.01	20.1	250	13.68	4510	21.86	0.35	1579	86.32	4523	21.85	0.40	1829	16.05	2.20	11.68
37	9.10	13.93	40.7	371	36.69	2560	12.41	0.25	640	63.31	2569	12.41	0.39	1011	8.87	3.25	14.93
25	5.06	7.74	78.0	395	46.85	2238	10.85	0.20	448	53.15	2243	10.84	0.38	842	7.39	3.46	18.39
15	5.68	8.69	238.5	1355	28.39	6507	31.54	0.53	3416	71.61	6513	31.47	0.73	4771	41.88	11.89	30.28
Total	65.35	100.00	52.8	3450	30.28	20632	100.00	0.38	7943	69.72	20697	100.00	0.55	11393	100.00	30.28	

Table 41: AU3 Stage 3 mass balance (1-Stage)

Simple AU Composite Stage 3 Tails Falcon - 5.634 L/min, 59:00 min, 431.4 g/min

File = Simple AU Falcon.xls %-75μm: 72.16 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	1.52	2.31	4.5	7	45.56	18	0.06	0.45	8	54.44	20	0.07	0.76	15	0.08	0.04	0.04
150	3.26	4.95	0.9	3	1.47	451	1.51	0.45	203	98.53	454	1.52	0.45	206	1.13	0.02	0.05
106	9.25	14.06	1.2	11	0.73	2489	8.34	0.60	1494	99.27	2499	8.36	0.60	1505	8.28	0.06	0.11
75	14.33	21.78	0.2	3	0.09	5339	17.89	0.65	3470	99.91	5353	17.90	0.65	3473	19.11	0.02	0.13
53	13.60	20.67	1.9	26	0.72	7156	23.98	0.50	3578	99.28	7170	23.98	0.50	3604	19.84	0.14	0.27
37	9.48	14.41	4.2	40	1.59	4527	15.17	0.55	2490	98.41	4536	15.17	0.56	2530	13.92	0.22	0.50
25	6.01	9.13	16.5	99	5.64	4151	13.91	0.40	1660	94.36	4157	13.90	0.42	1760	9.68	0.55	1.04
20	2.99	4.54	47.0	141	24.36	1248	4.18	0.35	437	75.64	1251	4.18	0.46	577	3.18	0.77	1.82
15	5.36	8.15	398.5	2136	47.46	4461	14.95	0.53	2364	52.54	4466	14.93	1.01	4500	24.77	11.75	13.57
Total	65.80	100.00	37.5	2466	13.57	29839	100.00	0.53	15704	86.43	29905	100.00	0.61	18170	100.00	13.57	

Table 42: AU Simple Falcon mass balance

Standard AU Composite Stage 3 Tails **Falcon - 5.834 L/min, 73:38 min, 426.4 g/min**

File = Standard AU Falcon.xls **%-75μm:** 77.43 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	2.72	3.47	8.0	22	47.58	45	0.15	0.54	24	52.42	47	0.16	0.97	46	0.28	0.13	0.13
150	5.91	7.54	0.5	3	1.28	430	1.45	0.54	232	98.72	435	1.46	0.54	235	1.41	0.02	0.15
106	9.94	12.68	2.0	20	1.82	1959	6.60	0.55	1077	98.18	1969	6.62	0.56	1097	6.59	0.12	0.27
75	13.37	17.06	2.0	27	1.26	4248	14.32	0.50	2124	98.74	4261	14.33	0.50	2151	12.91	0.16	0.43
53	13.16	16.79	2.1	27	0.94	5704	19.23	0.50	2852	99.06	5717	19.22	0.50	2879	17.29	0.16	0.59
37	11.10	14.16	3.1	34	2.13	3920	13.21	0.40	1568	97.87	3931	13.21	0.41	1602	9.62	0.21	0.80
25	8.75	11.16	9.0	79	4.33	4350	14.66	0.40	1740	95.67	4359	14.65	0.42	1819	10.92	0.47	1.27
20	4.50	5.74	20.9	94	17.12	1302	4.39	0.35	456	82.88	1306	4.39	0.42	550	3.30	0.56	1.84
15	8.92	11.38	293.1	2615	41.65	7711	25.99	0.48	3663	58.35	7720	25.95	0.81	6278	37.69	15.70	17.53
Total	78.37	100.00	37.3	2921	17.53	29668	100.00	0.46	13736	82.47	29746	100.00	0.56	16656	100.00	17.53	

Table 43: AU Standard Falcon mass balance

QC3, Stage 1, 5.964 L/min, 3.0 psi (3.3 psi), 28:25 min, 724.3 g/min

File = Simple QC3.xls %-75μm: **21.95** %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	53.78	30.70	1.7	91	4.75	3050	15.20	0.60	1830	95.25	3104	15.33	0.62	1921	11.31	0.54	0.54
425	52.65	30.05	2.4	124	4.55	3721	18.54	0.70	2605	95.45	3774	18.64	0.72	2729	16.06	0.73	1.27
300	29.32	16.74	9.5	279	8.80	2753	13.72	1.05	2890	91.20	2782	13.74	1.14	3169	18.66	1.64	2.91
212	16.23	9.26	5.2	85	4.21	2404	11.98	0.80	1923	95.79	2420	11.96	0.83	2008	11.82	0.50	3.41
150	9.97	5.69	24.8	247	17.05	1605	8.00	0.75	1204	82.95	1615	7.98	0.90	1451	8.54	1.46	4.86
106	5.99	3.42	43.6	261	16.55	1196	5.96	1.10	1315	83.45	1202	5.94	1.31	1576	9.28	1.54	6.40
75	3.01	1.72	51.7	156	18.71	901	4.49	0.75	676	81.29	904	4.46	0.92	831	4.89	0.92	7.31
53	1.85	1.06	122.0	226	23.32	928	4.62	0.80	742	76.68	929	4.59	1.04	968	5.70	1.33	8.64
37	1.01	0.58	208.2	210	40.88	491	2.44	0.62	304	59.12	492	2.43	1.05	515	3.03	1.24	9.88
25	0.50	0.29	386.7	193	38.99	458	2.28	0.66	303	61.01	459	2.27	1.08	496	2.92	1.14	11.02
15	0.89	0.51	308.5	275	20.72	2562	12.77	0.41	1051	79.28	2563	12.66	0.52	1325	7.80	1.62	12.64
Total	175.20	100.00	12.3	2147	12.64	20069	100.00	0.74	14843	87.36	20244	100.00	0.84	16989	100.00	12.64	

Table 44: QC3 Stage 1 mass balance (2-Stage)

QC3, Stage 3, 4.836 L/min, 2.5 psi (3.6 psi), 49:00 min, 342.0 g/min

File = Simple QC3.xls %-75μm: 72.86 %

	CONCENTRATE					TAILS					FEED						
Size (μm)	Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.28	0.23	407.41	114	92.16	19	0.11	0.52	10	7.84	19	0.11	6.53	124	0.79	0.72	0.72
150	4.81	3.97	60.09	289	64.13	311	1.84	0.52	162	35.87	316	1.85	1.43	451	2.86	1.84	2.56
106	20.85	17.19	24.47	510	30.15	1576	9.33	0.75	1182	69.85	1597	9.38	1.06	1692	10.76	3.24	5.80
75	33.68	27.77	21.08	710	25.05	2655	15.71	0.80	2124	74.95	2689	15.80	1.05	2834	18.01	4.51	10.32
53	31.04	25.60	25.71	798	23.95	3168	18.75	0.80	2535	76.05	3199	18.79	1.04	3333	21.18	5.07	15.39
37	17.37	14.32	36.45	633	35.19	1666	9.86	0.70	1166	64.81	1683	9.89	1.07	1799	11.43	4.02	19.41
25	7.61	6.28	69.04	525	33.76	1472	8.71	0.70	1031	66.24	1480	8.69	1.05	1556	9.89	3.34	22.75
15	5.63	4.64	218.67	1231	31.20	6034	35.70	0.45	2715	68.80	6040	35.48	0.65	3946	25.08	7.82	30.57
Total	121.27	100.00	39.67	4811	30.57	16902	100.00	0.65	10925	69.43	17023	100.00	0.92	15735	100.00	30.57	

Table 45: QC3 Stage 3 mass balance (2-Stage)

QC3 Stage 3 Direct, 4.868 L/min, 2.6 psi (3.6 psi), 65:00 min, 306.9 g/min

File = Simple QC3.xls %-75μm: 76.50 %

	CONCENTRATE					TAILS					FEED						
Size (μm)	Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.30	0.25	600.0	180	96.66	18	0.09	0.34	6	3.34	19	0.09	10.03	186	0.98	0.95	0.95
150	4.31	3.54	119.7	516	84.11	287	1.44	0.34	97	15.89	291	1.46	2.11	613	3.23	2.71	3.66
106	21.43	17.59	31.7	679	45.13	1500	7.56	0.55	825	54.87	1522	7.62	0.99	1504	7.91	3.57	7.23
75	35.95	29.51	28.0	1008	30.84	2825	14.24	0.80	2260	69.16	2861	14.33	1.14	3268	17.19	5.30	12.53
53	30.46	25.01	32.0	975	24.50	3537	17.82	0.85	3006	75.50	3567	17.87	1.12	3982	20.94	5.13	17.66
37	16.51	13.55	52.9	873	40.08	1865	9.40	0.70	1306	59.92	1882	9.42	1.16	2179	11.46	4.59	22.25
25	7.07	5.80	97.4	689	35.24	1688	8.50	0.75	1266	64.76	1695	8.49	1.15	1955	10.28	3.62	25.87
15	5.78	4.75	289.3	1672	31.38	8124	40.94	0.45	3656	68.62	8130	40.72	0.66	5328	28.02	8.79	34.67
Total	121.81	100.00	54.1	6592	34.67	19844	100.00	0.63	12423	65.33	19966	100.00	0.95	19014	100.00	34.67	

Table 46: QC3 Stage 3 mass balance (1-Stage)

QC4, Stage 1, 6.002 L/min, 3.0 psi (3.5 psi), 27:00 min, 780.9 g/min

File = Simple QC4.xls %-75μm: **21.06** %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	42.37	25.09	29.8	1264	1.67	3240	16.18	23.00	74514	98.33	3282	16.25	23.09	75778	14.64	0.24	0.24
425	48.51	28.73	74.8	3629	4.90	3703	18.49	19.00	70356	95.10	3751	18.58	19.72	73985	14.29	0.70	0.95
300	31.65	18.74	63.0	1993	3.77	2584	12.90	19.70	50900	96.23	2615	12.95	20.22	52893	10.22	0.39	1.33
212	18.23	10.80	485.2	8845	17.84	2184	10.91	18.65	40741	82.16	2203	10.91	22.51	49586	9.58	1.71	3.04
150	10.33	6.12	1601.1	16539	39.04	1476	7.37	17.50	25829	60.96	1486	7.36	28.51	42368	8.18	3.19	6.23
106	6.77	4.01	3751.7	25399	62.47	1282	6.40	11.90	15258	37.53	1289	6.38	31.54	40657	7.85	4.91	11.14
75	4.52	2.68	6859.6	31006	74.06	1308	6.53	8.30	10859	25.94	1313	6.50	31.89	41864	8.09	5.99	17.13
53	3.44	2.04	10702.6	36817	76.83	1835	9.16	6.05	11102	23.17	1839	9.11	26.06	47919	9.26	7.11	24.24
37	1.90	1.13	14743.3	28012	84.17	924	4.61	5.70	5267	15.83	926	4.59	35.94	33279	6.43	5.41	29.65
25	0.72	0.43	22636.8	16298	74.44	629	3.14	8.90	5597	25.56	630	3.12	34.78	21895	4.23	3.15	32.80
15	0.42	0.25	40851.4	17158	45.78	858	4.28	23.69	20317	54.22	858	4.25	43.67	37474	7.24	3.31	36.11
Total	168.86	100.00	1107.2	186960	36.11	20023	100.00	16.52	330740	63.89	20192	100.00	25.64	517700	100.00	36.11	

Table 47: QC4 Stage 1 mass balance (2-Stage)

QC4 Stage 3, 4.718 L/min, 2.5 psi (3.4 psi), 48:50 min, 375.2 g/min

File = Simple QC4.xls **%-75μm:** 81.34 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.25	0.28	9756.0	2439	98.92	6	0.03	4.69	27	1.08	6	0.03	414.28	2466	0.87	0.87	0.87
150	1.8	2.02	3554.1	6397	95.91	58	0.32	4.69	273	4.09	60	0.33	111.14	6671	2.37	2.27	3.14
106	11.19	12.55	1567.5	17541	90.38	718	4.01	2.60	1868	9.62	730	4.05	26.60	19409	6.89	6.22	9.36
75	24.11	27.05	1239.7	29890	84.53	2543	14.18	2.15	5468	15.47	2567	14.24	13.77	35358	12.54	10.60	19.96
53	24.08	27.02	1841.0	44331	88.87	4439	24.75	1.25	5549	11.13	4463	24.76	11.18	49880	17.70	15.73	35.69
37	15.69	17.60	2566.4	40266	91.70	3036	16.93	1.20	3643	8.30	3052	16.93	14.39	43909	15.58	14.29	49.98
25	7.29	8.18	3934.5	28683	87.13	2230	12.44	1.90	4238	12.87	2238	12.41	14.71	32920	11.68	10.18	60.15
15	4.72	5.30	10398.5	49081	53.79	4904	27.34	8.60	42171	46.21	4908	27.23	18.59	91252	32.37	17.41	77.56
Total	89.13	100.00	2452.9	218627	77.56	17935	100.00	3.53	63237	22.44	18024	100.00	15.64	281864	100.00	77.56	

Table 48: QC4 Stage 3 mass balance (2-Stage)

QC4 Stage 3 Direct, 4.888 L/min, 2.6 psi (3.0 psi), 58:05 min, 349.1 g/min

File = Simple QC4.xls %-75μm: **80.26** %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	28.12	16.16	39.9	1122	32.26	207	1.07	11.39	2357	67.74	235	1.21	14.80	3479	0.71	0.23	0.23
425	44.85	25.77	38.6	1731	19.04	345	1.79	21.34	7363	80.96	390	2.00	23.33	9095	1.87	0.36	0.59
300	26.81	15.40	101.1	2710	39.96	296	1.54	13.74	4071	60.04	323	1.66	20.99	6781	1.39	0.56	1.14
212	11.77	6.76	617.2	7265	63.89	263	1.36	15.63	4107	36.11	275	1.41	41.42	11372	2.33	1.49	2.63
150	4.95	2.84	3508.9	17369	87.05	201	1.04	12.87	2584	12.95	206	1.06	96.99	19953	4.09	3.56	6.20
106	5.39	3.10	7904.4	42605	95.44	443	2.29	4.60	2036	4.56	448	2.30	99.66	44641	9.16	8.74	14.94
75	12.57	7.22	4566.7	57404	91.15	1955	10.13	2.85	5571	8.85	1967	10.10	32.01	62975	12.92	11.78	26.72
53	15.26	8.77	4954.1	75600	89.53	4420	22.91	2.00	8839	10.47	4435	22.78	19.04	84439	17.33	15.51	42.23
37	12.34	7.09	5762.2	71106	93.62	2938	15.23	1.65	4848	6.38	2951	15.16	25.74	75954	15.58	14.59	56.82
25	6.70	3.85	6850.0	45895	89.55	2231	11.56	2.40	5355	10.45	2238	11.49	22.90	51250	10.52	9.42	66.23
15	5.30	3.04	12313.7	65262	55.57	5997	31.08	8.70	52174	44.43	6002	30.83	19.57	117436	24.10	13.39	79.62
Total	174.06	100.00	2229.5	388069	79.62	19295	100.00	5.15	99304	20.38	19469	100.00	25.03	487374	100.00	79.62	

Table 49: QC4 Stage 3 mass balance (1-Stage)

QC5, Stage 1, 6.010 L/min, 3.0 psi (3.4 psi), 25:46 min, 713.3 g/min

File = Simple QC5.xls %-75μm: **21.46** %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	42.70	21.97	4.4	189	5.44	2731	13.80	1.20	3277	94.56	2774	13.88	1.25	3466	9.71	0.53	0.53
425	57.83	29.76	6.2	356	6.42	3575	18.06	1.45	5183	93.58	3633	18.17	1.52	5539	15.52	1.00	1.53
300	40.31	20.74	10.3	417	8.91	2666	13.47	1.60	4265	91.09	2706	13.54	1.73	4682	13.12	1.17	2.69
212	22.95	11.81	21.3	489	14.12	2285	11.54	1.30	2970	85.88	2308	11.54	1.50	3459	9.69	1.37	4.06
150	12.28	6.32	91.8	1127	37.11	1528	7.72	1.25	1910	62.89	1540	7.70	1.97	3036	8.51	3.16	7.22
106	7.27	3.74	195.2	1419	48.79	1354	6.84	1.10	1489	51.21	1361	6.81	2.14	2908	8.15	3.98	11.20
75	4.72	2.43	370.2	1747	58.57	1373	6.94	0.90	1236	41.43	1378	6.89	2.17	2983	8.36	4.90	16.10
53	3.34	1.72	490.1	1637	51.79	1792	9.05	0.85	1523	48.21	1796	8.98	1.76	3160	8.86	4.59	20.68
37	1.82	0.94	676.4	1231	62.57	921	4.65	0.80	737	37.43	922	4.61	2.13	1968	5.51	3.45	24.13
25	0.67	0.34	1078.7	723	48.50	595	3.00	1.29	767	51.50	595	2.98	2.50	1490	4.18	2.03	26.16
15	0.44	0.23	1841.7	810	27.11	977	4.93	2.23	2178	72.89	977	4.89	3.06	2989	8.38	2.27	28.43
Total	194.33	100.00	52.2	10144	28.43	19796	100.00	1.29	25536	71.57	19990	100.00	1.78	35681	100.00	28.43	

Table 50: QC5 Stage 1 mass balance (2-Stage)

QC5 Stage 3, 4.646 L/min, 2.4 psi (3.2 psi), 46:00 min, 367.6 g/min

File = Simple QC5.xls %-75μm: 88.32 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.47	0.55	361.70	170	99.64	6	0.04	0.10	1	0.36	7	0.04	25.89	171	0.76	0.75	0.75
150	0.23	0.27	539.13	124	98.04	25	0.14	0.10	2	1.96	25	0.14	5.06	126	0.56	0.55	1.30
106	2.65	3.08	389.06	1031	94.68	161	0.92	0.36	58	5.32	164	0.93	6.66	1089	4.83	4.57	5.88
75	19.44	22.63	120.95	2351	92.78	1830	10.51	0.10	183	7.22	1849	10.57	1.37	2534	11.24	10.43	16.31
53	30.29	35.26	105.81	3205	87.19	4707	27.04	0.10	471	12.81	4738	27.08	0.78	3676	16.31	14.22	30.53
37	18.38	21.39	136.22	2504	88.50	3254	18.69	0.10	325	11.50	3273	18.71	0.86	2829	12.55	11.11	41.64
25	8.45	9.84	209.24	1768	83.85	2270	13.04	0.15	341	16.15	2279	13.02	0.93	2109	9.36	7.85	49.49
15	6	6.98	807.74	4846	48.45	5157	29.62	1.00	5157	51.55	5163	29.51	1.94	10003	44.39	21.50	70.99
Total	85.91	100.00	186.23	15999	70.99	17410	100.00	0.38	6537	29.01	17496	100.00	1.29	22537	100.00	70.99	

Table 51: QC5 Stage 3 mass balance (2-Stage)

QC5 Stage 3 Direct, 5.080 L/min, 2.5 psi (3.6 psi), 58:15 min, 336.8 g/min

File = Simple QC5.xls %-75μm: 84.18 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	1.03	1.02	239.2	246	93.17	11	0.05	1.70	18	6.83	12	0.06	22.70	264	0.61	0.57	0.57
150	3.15	3.13	300.7	947	85.66	93	0.47	1.70	159	14.34	96	0.48	11.47	1106	2.55	2.18	2.75
106	9.20	9.14	273.9	2520	96.16	559	2.82	0.18	101	3.84	569	2.85	4.61	2620	6.04	5.81	8.56
75	24.51	24.34	170.3	4175	91.89	2458	12.37	0.15	369	8.11	2482	12.43	1.83	4543	10.47	9.62	18.18
53	29.79	29.58	170.6	5083	44.42	5299	26.68	1.20	6359	55.58	5329	26.69	2.15	11441	26.37	11.71	29.89
37	18.95	18.82	212.2	4022	50.99	3362	16.93	1.15	3866	49.01	3381	16.94	2.33	7888	18.18	9.27	39.16
25	8.46	8.40	306.6	2594	67.81	2462	12.40	0.50	1231	32.19	2471	12.38	1.55	3825	8.81	5.98	45.14
15	5.62	5.58	707.8	3978	33.99	5617	28.28	1.38	7724	66.01	5623	28.17	2.08	11702	26.97	9.17	54.31
Total	100.71	100.00	234.0	23564	54.31	19861	100.00	1.00	19826	45.69	19962	100.00	2.17	43390	100.00	54.31	

Table 52: QC5 Stage 3 mass balance (1-Stage)

QC5 Rescreen Stage 3 Direct, 5.080 L/min, 2.5 psi (3.6 psi), 58:15 min, 336.8 g/min

File = Simple QC5.xls %-75μm: 84.18 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	1.03	1.02	239.2	246	99.57	11	0.05	0.10	1	0.43	12	0.06	21.24	247	0.76	0.76	0.76
150	3.15	3.13	300.7	947	99.03	93	0.47	0.10	9	0.97	96	0.48	9.92	956	2.94	2.92	3.67
106	9.20	9.14	273.9	2520	96.16	559	2.82	0.18	101	3.84	569	2.85	4.61	2620	8.07	7.76	11.43
75	24.51	24.34	170.3	4175	94.44	2458	12.37	0.10	246	5.56	2482	12.43	1.78	4420	13.61	12.85	24.28
53	29.79	29.58	170.6	5083	90.56	5299	26.68	0.10	530	9.44	5329	26.69	1.05	5613	17.28	15.65	39.93
37	18.95	18.82	212.2	4022	88.86	3362	16.93	0.15	504	11.14	3381	16.94	1.34	4526	13.93	12.38	52.31
25	8.46	8.40	306.6	2594	87.53	2462	12.40	0.15	369	12.47	2471	12.38	1.20	2963	9.12	7.98	60.29
15	5.62	5.58	707.8	3978	35.71	5617	28.28	1.28	7162	64.29	5623	28.17	1.98	11140	34.29	12.24	72.53
Total	100.71	100.00	234.0	23564	72.53	19861	100.00	0.45	8923	27.47	19962	100.00	1.63	32487	100.00	72.53	

Table 53: QC5 Stage 3 Re-screen mass balance (1-Stage)

PE.xls **Stage 3 Direct, 5.066 L/min, 2.6 psi (3.6 psi), 51:20 min, 370.0 g/min**
File = Simple PE.xls **%-75μm:** 70.77 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.23	0.20	2283.4	525	99.17	9	0.05	0.48	4	0.83	9	0.05	56.10	530	1.27	1.26	1.26
150	2.92	2.60	166.3	486	89.37	120	0.66	0.48	58	10.63	123	0.67	4.41	543	1.30	1.16	2.42
106	20.30	18.05	45.6	925	51.35	1752	9.55	0.50	876	48.65	1773	9.60	1.02	1801	4.32	2.22	4.64
75	34.04	30.27	36.6	1245	35.65	3457	18.84	0.65	2247	64.35	3491	18.91	1.00	3492	8.37	2.99	7.63
53	27.93	24.84	69.0	1928	35.91	4302	23.44	0.80	3441	64.09	4330	23.45	1.24	5370	12.88	4.62	12.25
37	16.12	14.33	152.6	2460	48.78	2583	14.07	1.00	2583	51.22	2599	14.08	1.94	5043	12.09	5.90	18.15
25	6.63	5.90	439.4	2914	61.23	1677	9.14	1.10	1845	38.77	1684	9.12	2.83	4758	11.41	6.99	25.14
15	4.29	3.81	2209.6	9479	47.01	4452	24.26	2.40	10684	52.99	4456	24.13	4.53	20163	48.35	22.73	47.87
Total	112.46	100.00	177.5	19961	47.87	18353	100.00	1.18	21739	52.13	18465	100.00	2.26	41700	100.00	47.87	

Table 54: PE Stage 3 mass balance (1-Stage)

BC.xls **Stage 3 Direct, 5.162 L/min, 2.6 psi (3.8 psi), 46:00 min, 349.1 g/min**
File = Simple BC.xls **%-75μm:** 81.66 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.08	0.07	29075.0	2326	99.69	4	0.02	1.65	7	0.31	4	0.02	527.66	2333	6.58	6.56	6.56
150	1.04	0.96	1940.2	2018	97.19	35	0.18	1.65	58	2.81	36	0.18	56.97	2076	5.86	5.69	12.25
106	12.01	11.11	81.9	984	61.62	681	3.41	0.90	613	38.38	693	3.45	2.30	1597	4.50	2.77	15.03
75	31.63	29.26	33.2	1050	27.51	2913	14.60	0.95	2767	72.49	2944	14.68	1.30	3817	10.77	2.96	17.99
53	28.81	26.65	41.4	1192	19.54	4269	21.40	1.15	4909	80.46	4297	21.42	1.42	6101	17.21	3.36	21.35
37	16.59	15.35	75.0	1244	33.76	2219	11.12	1.10	2441	66.24	2236	11.14	1.65	3685	10.39	3.51	24.86
25	9.00	8.33	120.0	1080	33.73	1697	8.51	1.25	2121	66.27	1706	8.50	1.88	3201	9.03	3.04	27.90
15	8.93	8.26	163.9	1463	11.57	8133	40.77	1.38	11183	88.43	8142	40.59	1.55	12646	35.67	4.13	32.03
Total	108.09	100.00	105.1	11357	32.03	19951	100.00	1.21	24099	67.97	20059	100.00	1.77	35456	100.00	32.03	

Table 55: BC Stage 3 mass balance (1-Stage)

BC Sample, Standard Overall Results

Size (µm)	First Stage: 100% -850 µm Stage Recov.	Dist'n	Rec. g/t	Second Stage: 52.8% -75 µr Stage Recov.	Dist'n	Rec. g/t	Third Stage: 81.3% -75 µm Stage Recov.	Dist'n	Rec. g/t	Losses g/t	Total Recov. g/t	Total Recov. %	Cumul Recov. %	
850														
600	75.27	21.13	0.338								0.338	15.9	15.89	600
425	13.65	8.83	0.026								0.026	1.2	17.09	425
300	21.96	9.26	0.043	60.24	3.03	0.025					0.069	3.2	20.32	300
212	18.42	8.14	0.032	35.91	4.23	0.021					0.053	2.5	22.82	212
150	22.45	7.44	0.035	14.79	9.80	0.020	0.00	0.00	0.000	0.00	0.056	2.6	25.44	150
106	24.00	7.34	0.037	6.35	12.68	0.011	10.76	2.70	0.004	0.03	0.052	2.5	27.90	106
75	24.64	5.91	0.031	9.26	10.40	0.013	4.37	10.21	0.006	0.12	0.050	2.4	30.25	75
53	28.48	6.33	0.038	9.42	11.69	0.015	4.01	13.63	0.007	0.17	0.061	2.8	33.10	53
37	33.98	5.50	0.040	12.59	9.03	0.016	5.78	11.15	0.008	0.13	0.064	3.0	36.10	37
25	49.63	4.37	0.046	23.99	6.43	0.022	8.58	10.80	0.012	0.12	0.079	3.7	39.83	25
20	14.52	4.33	0.013	13.37	5.69	0.011	13.06	4.40	0.007	0.05	0.031	1.5	43.53	15
15	3.56	11.42	0.009	4.41	27.04	0.017	3.71	47.11	0.022	0.57	0.047	2.2		
Total	32.4	100.0	0.69	11.9	100.0	0.172	5.1	100.0	0.066	1.20	0.92	43.5		
O/A	32.4			8.1			3.1							
Yield	0.00161			0.00415			0.00419							
Grade	2.12 g/t			1.40 g/t			1.27 g/t							
Calc.:	2.12 g/t													

Table 56: BC Standard overall balance

QC6.xls Stage 1, 6.586 L/min, 3.2 psi (3.8 psi), 22:36 min, 729.3 g/min
(File = Simple QC6.xls %-75µm: 33.13 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
600	59.40	31.42	88.7	5268	16.06	2960	14.27	9.30	27524	83.94	3019	14.43	10.86	32791	16.38	2.63	2.63
425	51.20	27.09	60.7	3107	12.16	2842	13.71	7.90	22452	87.84	2893	13.83	8.83	25559	12.77	1.55	4.18
300	29.54	15.63	140.8	4160	19.21	2046	9.87	8.55	17492	80.79	2075	9.92	10.43	21652	10.82	2.08	6.26
212	17.56	9.29	307.2	5394	21.42	1885	9.09	10.50	19792	78.58	1903	9.09	13.24	25186	12.58	2.69	8.96
150	11.02	5.83	586.0	6458	38.41	1438	6.94	7.20	10354	61.59	1449	6.93	11.60	16812	8.40	3.23	12.18
106	7.00	3.70	867.0	6069	40.07	1325	6.39	6.85	9077	59.93	1332	6.37	11.37	15146	7.57	3.03	15.21
75	4.69	2.48	1228.0	5759	46.72	1314	6.34	5.00	6568	53.28	1318	6.30	9.35	12327	6.16	2.88	18.09
53	3.61	1.91	1605.1	5794	46.15	1756	8.47	3.85	6762	53.85	1760	8.41	7.13	12556	6.27	2.89	20.98
37	2.42	1.28	2197.4	5318	59.57	1128	5.44	3.20	3610	40.43	1130	5.40	7.90	8928	4.46	2.66	23.64
25	1.20	0.63	2907.2	3489	49.05	1006	4.85	3.60	3623	50.95	1008	4.82	7.06	7112	3.55	1.74	25.38
15	1.39	0.74	3194.7	4441	20.06	3033	14.63	5.84	17697	79.94	3034	14.50	7.30	22137	11.06	2.22	27.60
Total	189.03	100.00	292.3	55256	27.60	20733	100.00	6.99	144951	72.40	20922	100.00	9.57	200207	100.00	27.60	

Table 57: QC6 Stage 1 mass balance (2-Stage)

QC6.xls Stage 3, 4.948 L/min, 2.4 psi (3.5 psi), 53:15 min, 339.3 g/min
(File = Simple QC6.xls %-75µm: 78.05 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.22	0.22	5589.47	1230	93.16	16	0.09	5.51	90	6.84	17	0.09	79.53	1320	0.95	0.89	0.89
150	5.82	5.85	546.30	3179	74.05	202	1.08	5.51	1114	25.95	208	1.10	20.64	4294	3.09	2.29	3.18
106	23.8	23.92	193.27	4600	33.80	1345	7.15	6.70	9010	66.20	1369	7.24	9.94	13610	9.80	3.31	6.49
75	27.46	27.60	194.90	5352	26.08	2528	13.45	6.00	15170	73.92	2556	13.52	8.03	20522	14.78	3.86	10.35
53	19.92	20.02	342.91	6831	32.55	3453	18.36	4.10	14156	67.45	3473	18.37	6.04	20987	15.12	4.92	15.27
37	11.98	12.04	601.01	7200	50.27	2226	11.84	3.20	7122	49.73	2238	11.84	6.40	14322	10.32	5.19	20.45
25	5.71	5.74	1074.91	6138	50.86	2081	11.07	2.85	5929	49.14	2086	11.04	5.78	12067	8.69	4.42	24.88
15	4.59	4.61	3196.21	14671	28.38	6952	36.97	5.33	37018	71.62	6956	36.80	7.43	51689	37.24	10.57	35.44
Total	99.50	100.00	494.47	49200	35.44	18803	100.00	4.77	89611	64.56	18902	100.00	7.34	138811	100.00	35.44	

Table 58: QC6 Stage 3 mass balance (2-Stage)

QC6.xls **Stage 3 Direct, 5.080 L/min, 2.5 psi (3.6 psi), 58:15 min, 336.8 g/min**
(File = Simple QC6.xls %-75μm: 81.21 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.16	0.15	60553.3	9689	99.00	16	0.08	6.22	98	1.00	16	0.08	616.62	9786	5.25	5.20	5.20
150	4.68	4.41	1950.7	9130	91.74	132	0.68	6.22	822	8.26	137	0.70	72.69	9952	5.34	4.90	10.10
106	23.74	22.36	400.2	9501	58.29	1038	5.32	6.55	6798	41.71	1062	5.41	15.35	16299	8.75	5.10	15.19
75	30.50	28.73	354.3	10806	43.28	2441	12.51	5.80	14160	56.72	2472	12.60	10.10	24967	13.40	5.80	20.99
53	21.97	20.70	557.4	12245	43.88	3519	18.04	4.45	15661	56.12	3541	18.05	7.88	27906	14.97	6.57	27.56
37	13.11	12.35	923.7	12109	61.54	2259	11.58	3.35	7569	38.46	2273	11.58	8.66	19678	10.56	6.50	34.06
25	6.57	6.19	1474.2	9685	58.01	2225	11.40	3.15	7010	41.99	2232	11.38	7.48	16695	8.96	5.20	39.26
15	5.43	5.11	3341.5	18145	29.70	7882	40.39	5.45	42955	70.30	7887	40.20	7.75	61099	32.78	9.74	48.99
Total	106.16	100.00	860.1	91310	48.99	19513	100.00	4.87	95073	51.01	19619	100.00	9.50	186383	100.00	48.99	

Table 59: QC6 Stage 3 mass balance (1-Stage)

SA1 **Stage 3 Direct, 4.950 L/min, 2.6 psi (3.2 psi), 65:00 min, 323.4 g/min**
File = Simple SA1-2.xls %-75μm: 82.99 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.40	0.45	7893	3157	99.67	8	0.04	1.28	11	0.33	9	0.04	366.74	3168	4.88	4.86	4.86
150	3.98	4.49	722	2873	94.64	127	0.62	1.28	163	5.36	131	0.64	23.17	3035	4.67	4.42	9.29
106	11.52	12.99	267	3079	86.62	731	3.58	0.65	475	13.38	743	3.62	4.78	3554	5.47	4.74	14.03
75	29.24	32.97	165	4821	71.38	2577	12.62	0.75	1933	28.62	2607	12.71	2.59	6754	10.40	7.42	21.45
53	25.08	28.28	206	5159	50.35	3768	18.45	1.35	5087	49.65	3793	18.49	2.70	10246	15.78	7.95	29.40
38	11.24	12.67	365	4102	55.06	1969	9.64	1.70	3347	44.94	1980	9.65	3.76	7449	11.47	6.32	35.71
25	4.03	4.54	738	2974	48.22	1774	8.69	1.80	3193	51.78	1778	8.67	3.47	6167	9.50	4.58	40.29
15	3.20	3.61	1979	6332	25.78	9468	46.36	1.93	18226	74.22	9471	46.17	2.59	24557	37.82	9.75	50.05
Total	88.69	100.00	366	32495	50.05	20423	100.00	1.59	32435	49.95	20512	100.00	3.17	64931	100.00	50.05	

Table 60: SA1 Stage 3 mass balance (1-Stage)

SA2 **Stage 3 Direct, 5.002 L/min, 2.6 psi (3.4 psi), 59:17 min, 381.2 g/min**
File = Simple SA1-2.xls %-75µm: 83.13 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.17	0.19	156507	26606	99.62	10	0.05	9.69	101	0.38	11	0.05	2520.2	26707	2.94	2.93	2.93
150	0.70	0.78	79805	55863	98.88	65	0.33	9.69	632	1.12	66	0.33	856.6	56496	6.23	6.16	9.09
106	8.88	9.87	10471	92981	95.68	594	2.96	7.07	4202	4.32	603	2.99	161.1	97183	10.71	10.25	19.34
75	30.19	33.57	3557	107375	85.63	2689	13.41	6.70	18016	14.37	2719	13.50	46.1	125391	13.82	11.84	31.18
53	28.38	31.56	4572	129746	81.86	3940	19.65	7.30	28761	18.14	3968	19.70	39.9	158507	17.48	14.30	45.49
38	13.04	14.50	8325	108563	87.93	2069	10.32	7.20	14899	12.07	2082	10.34	59.3	123462	13.61	11.97	57.46
25	4.65	5.17	14525	67543	79.22	1885	9.40	9.40	17719	20.78	1890	9.38	45.1	85261	9.40	7.45	64.90
15	3.92	4.36	26474	103777	44.35	8799	43.88	14.80	130225	55.65	8803	43.70	26.6	234002	25.80	11.44	76.34
Total	89.93	100.00	7700	692454	76.34	20052	100.00	10.70	214555	23.66	20142	100.00	45.03	907009	100.00	76.34	

Table 61: SA2 Stage 3 mass balance (1-Stage)

SA2 Repe: **Stage 3 Direct, 4.90 L/min, 2.6 psi (3.3 psi), 64:00 min, 369.9 g/min**
File = SA1-2.xls %-75µm: 79.60 %

Size (µm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.12	0.13	237975	28557	99.39	18	0.09	9.94	174	0.61	18	0.09	1629.6	28731	3.10	3.08	3.08
150	1.61	1.71	36932	59461	98.27	105	0.52	9.94	1048	1.73	107	0.52	565.4	60508	6.53	6.42	9.50
106	13.34	14.15	6075	81046	91.33	987	4.83	7.80	7697	8.67	1000	4.87	88.7	88743	9.58	8.75	18.25
75	31.83	33.76	3625	115369	82.11	3029	14.83	8.30	25141	17.89	3061	14.92	45.9	140510	15.17	12.45	30.71
53	25.25	26.78	5189	131027	80.60	3893	19.06	8.10	31533	19.40	3918	19.09	41.5	162560	17.55	14.14	44.85
38	12.75	13.52	8535	108825	87.38	1976	9.68	7.95	15712	12.62	1989	9.69	62.6	124537	13.44	11.75	56.60
25	5.23	5.55	13596	71109	79.21	1894	9.27	9.85	18659	20.79	1900	9.26	47.3	89767	9.69	7.68	64.27
15	4.16	4.41	25454	105891	45.85	8523	41.73	14.68	125081	54.15	8528	41.56	27.1	230971	24.93	11.43	75.71
Total	94.29	100.00	7438	701285	75.71	20426	100.00	11.02	225044	24.29	20520	100.00	45.14	926329	100.00	75.71	

Table 62: SA2 Repeat Stage 3 mass balance (1-Stage)

SA2 1-Stg Falcon, 5.330 L/min, 1 psi, 38:30 min, 498.7 g/min

File = SA1-2.xls %-75μm: 79.39 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.61	0.66	39	24	25.05	5	0.03	13.56	71	74.95	6	0.03	16.2	95	0.05	0.01	0.01
150	1.04	1.12	27	29	3.77	54	0.27	13.56	729	96.23	55	0.28	13.8	758	0.37	0.01	0.03
106	7.39	7.97	56	416	6.55	855	4.37	6.95	5940	93.45	862	4.38	7.4	6356	3.08	0.20	0.23
75	21.87	23.57	76	1672	6.95	3108	15.88	7.20	22379	93.05	3130	15.92	7.7	24051	11.67	0.81	1.04
53	23.01	24.80	133	3060	10.45	4095	20.93	6.40	26210	89.55	4118	20.95	7.1	29270	14.20	1.48	2.52
38	15.04	16.21	274	4114	24.08	2162	11.05	6.00	12971	75.92	2177	11.07	7.8	17085	8.29	2.00	4.52
25	8.84	9.53	657	5808	27.52	2301	11.76	6.65	15298	72.48	2309	11.75	9.1	21106	10.24	2.82	7.34
20	4.69	5.05	1350	6329	47.55	970	4.96	7.20	6982	52.45	974	4.96	13.7	13311	6.46	3.07	10.41
15	10.29	11.09	5257	54095	57.48	6018	30.76	6.65	40020	42.52	6028	30.66	15.6	94115	45.65	26.24	36.65
Total	92.78	100.00	814	75547	36.65	19567	100.00	6.67	130600	63.35	19660	100.00	10.49	206147	100.00	36.65	

Table 63: SA2 Simple Falcon mass balance

SA2 Std Falcon, 5.478 L/min, 4 psi, 36:00 min, 525.3 g/min

File = SA1-2.xls %-75μm: 80.56 %

Size (μm)	CONCENTRATE Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	TAILS Weight (g)	% Weight	Grade (g/t)	Units	Rec. (%)	FEED Weight (g)	% Weight	Grade (g/t)	Units	Dist'n (%)	Total Recov. (%)	Cumul Recov (%)
212	0.47	0.33	22	10	39.78	3	0.01	5.82	15	60.22	3	0.02	8.2	26	0.01	0.01	0.01
150	1.81	1.27	15	26	6.85	61	0.31	5.82	358	93.15	63	0.32	6.1	384	0.22	0.02	0.02
106	11.43	8.02	19	213	4.14	836	4.22	5.90	4932	95.86	847	4.25	6.1	5145	2.98	0.12	0.14
75	29.31	20.58	27	784	4.71	2934	14.82	5.40	15845	95.29	2963	14.86	5.6	16628	9.62	0.45	0.60
53	34.75	24.40	45	1579	6.78	4344	21.94	5.00	21718	93.22	4378	21.96	5.3	23297	13.48	0.91	1.51
38	25.21	17.70	79	2003	14.95	2257	11.40	5.05	11396	85.05	2282	11.44	5.9	13399	7.75	1.16	2.67
25	14.45	10.15	189	2737	17.98	2230	11.26	5.60	12487	82.02	2244	11.25	6.8	15224	8.81	1.58	4.25
20	7.79	5.47	560	4366	34.53	1183	5.97	7.00	8280	65.47	1191	5.97	10.6	12646	7.31	2.53	6.78
15	17.21	12.08	2472	42547	49.40	5950	30.06	7.33	43587	50.60	5968	29.93	14.4	86134	49.82	24.61	31.39
Total	142.43	100.00	381	54265	31.39	19798	100.00	5.99	118618	68.61	19940	100.00	8.67	172883	100.00	31.39	

Table 64: SA2 Standard Falcon mass balance