EXTRACTING GOLD FROM GRAVITY CONCENTRATES USING GRINDING AND SIEVING

by

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Abstract

This thesis presents a novel, chemical-free method of extracting Yukon placer gold from various secondary and tertiary gravity middlings. This method exploits the differences in relative malleability rather than density. At a certain point in gravity gold processing, the middling material contains too many high density minerals to be amenable to further gold extraction through gravity separation. Yukon miners only use gravity extraction methods and low grade gold concentrates are stockpiled for later processing through tedious hand picking, which is rarely completed. An 8-inch diameter batch rod mill was field tested throughout the Yukon placer fields to demonstrate a chemical-free extraction alternative which exploits the resistance of gold to grinding and allows the separation of free gold particles from the finer grind products with sieving. Physical assays often indicated recoveries greater than 90% of the contained gold particles on the oversize portion of the screens, while losses reported to the undersize. The gold remaining in the fine and evenly classified loss material is now amenable to gravity processing. Furthermore, this paper reviews the importance of maximizing gold extraction in the modern Yukon placer environment, followed by reviews of gravity based upgrading equipment, the high malleability of gold, the justification for rod mill use over other comminution methods, and finally field testing methodology and preliminary results.

Preface

This dissertation is an original intellectual product of the author, G. Clarkson. Field and lab testing portions were performed with assistance from Randy Clarkson, P.Eng., of the Klondike Placer Miners Association and Dan Walsh, P.E., Emeritus Professor of Mineral Preparation Engineering at the University of Alaska Fairbanks.

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List of Symbols

N_c	Critical Speed
#	Mesh size
$	US dollars
D	Mill diameter
d	Grind medium diameter
G_r	Grade of feed concentrate
M_L	Mill volume in litres
r	reload time between grinds
t	Grind time in minutes
x	Charge size in kg

List of Abbreviations and Units

“	Inch
cm	Centimetre
CSF	Corey shape factor
g	Gram
hr	Hour
kg	Kilogram
L	Litre
m	Metre
m^3	Cubic metre
mg	Milligram
min	Minute
mm	Millimetre
oz t	Troy ounce
PSD	Particle size distribution
rpm	Revolutions per minute
S.G.	Specific Gravity
t	Metric tonne
V	Volt
wt%	Weight percent
µm	Micrometer

Glossary of terms

Crude ounces	Troy ounces of non-refined gold, occurring as electrum, produced in the placer fields
Doré bar	Semi-pure smelted bar of a gold-silver alloy, usually produced at the mine sight and shipped to refiners
Electrum	Alloy of gold and silver
Gravity recoverable gold	For this thesis, gold that is reasonably recovered by placer upgrading devices under ideal feed conditions
Long tom	Small, narrow sluice used in secondary ore concentration
Middlings	In placer, processed concentrate too high grade to discard but not high grade enough for sale
Placer deposit	Accumulation of valuable heavy minerals (like gold) through natural gravity sorting during sedimentary processes
Salting	Manually adding gold to a sample to guarantee a minimum grade
Sluice	A riffled chute through which a dilute mixture of water and ore flows which concentrates heavy minerals
Staking	Physical marking of a gold claim property with stakes

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Finally, the author would like to acknowledge the contributions of his graduate supervisor Dr. Michael Hitch for his edits, patience, and guidance.

Dedication

To my parents, without whose support this would not have been possible. And especially to my loving future wife Sheena, thank you for keeping me positive and sane.

Chapter 1 - Introduction

Exploiting the high density of gold to separate it from waste materials is the oldest mineral processing technique known. This basic premise is still widely used in gravity concentration circuits today. But gravity processing always results in high value middlings, especially in the Yukon placer gold fields. Along with a uniquely high density, gold is also the most malleable metal. Can this malleability be exploited to extract gold from otherwise inaccessible placer middlings through grinding?

Yukon placer gold mining has been ongoing continuously since the 1860s, with the industry weathering each economic downturn since that era. Today, that industry produces approximately 60,000 crude troy ounces of gold from the territory, with the majority of that gold being produced in the Klondike region near Dawson City (Van Loon & Bond, 2014). Gold mined from placer deposits most commonly occurs as electrum, and the term "crude ounces" refers to unrefined gold production in the form of gold grains, nuggets and doré bars. Although the use of mercury or cyanide extraction is popular in artisanal mines in the third world, modern Yukon placer miners extract gold from their concentrates using strictly gravity-based means (Hinton, Veiga, & Veiga, 2003; Van Loon & Bond, 2014). Placer gold is 18-times more dense than water (S.G. = 18), making it much heavier than the surrounding waste minerals it occurs with. Gravity based gold processing manipulates this density difference to separate the gold from the waste materials by first processing ore through a modern sluice box to produce a primary concentrate. This primary concentrate is then upgraded using smaller scale gravity-based clean-up equipment. Yukon miners typically do not sell concentrates until they are greater than 70 wt% gold, and as primary concentrates rarely exceed 1 wt%, the upgrading machines play an essential part in gold production. This emphasis on producing clean concentrates also means rejecting a portion of the gold recovered to middlings, material that is not high enough grade for sale but not low enough to discard. These middlings can be reprocessed using gravity-based separation equipment to a point, but the more this material is upgraded, the higher the proportion of bulkier, higher density waste materials. High density waste materials are an issue, for as the proportion of these in a concentrate increase, the relative difference in density between the waste and gold is reduced. Furthermore, the cubic to spherical blocky shape of these overly processed particles gives them a similar relative density to the typically flatter gold particles (Walsh & Kelly, 1993). This affects the recovery of processing equipment, and eventually these high density middlings cannot be further processed using gravity based methods. Since Yukon miners only use density separation, the material is stockpiled. This concentrate can be as high as 180,000 g/t, and in an attempt to recover some of this value it is occasionally processed via tedious manual separation. The Yukon mining season is short, typically less than 4 months, so invariably stockpiling of middlings outpaces the hand picking. A strict permitting regime, environmental concerns, lack of expertise, and fear of repercussions from regulators prevent miners from attempting upgrading using chemical means such as flotation, mercury, and cyanidation.

Accessing the gold from these middlings is more essential than ever. Total gold production in the Yukon has been in the decline since the mid 90's despite a rise in gold price from under $400/oz t to over $1,500 in recent years (Van Loon & Bond, 2014). Declining grades, increasing costs, and high fuel prices are some of the causes that have offset high gold prices, causing narrowing profit margins and less money available for ambitious exploration projects. This lack of exploration has also resulted in less annual staking activity, despite the higher prices (Van Loon & Bond, 2014). Given these stresses on the industry, it is crucial for miners to maximize the gold extraction from their concentrates and unlock the value seized up in non-responsive gravity middlings.

This thesis proposes a chemical-free, environmentally friendly alternative to automate the upgrading of high density, high value middlings quickly and effectively using grinding and sieving. High density is not the only inherently unique property to gold particles that can be manipulated for separation. It is also the most malleable metal on the periodic table (Grimwade, 1992). Through a unique combination of its low reactivity and efficient atomic crystal structure, gold can be reduced in thickness to less than the width of a wavelength of natural light (Nutting & Nuttall, 1977). This malleability has made it extraordinarily resistant in grinding mills, where gold can have recirculating loads of up to 6700% (Banisi, Laplante, & Marois, 1991, pg 72). High density wastes that occur in placer deposits are dominantly metallic sulphides, which are all brittle materials. Since gold accommodates stress through shape change while metallic sulphides fail and shatter, gold can be separated from waste through grinding and classification. Shattered waste material reports to the sieve undersize, while gold particles that have been rolled and flattened are retained on the screen. The concentrate produced from this method, referred to as grinding extraction, is greater than 80-90 wt% gold and ready for immediate sale or smelting without further processing.

The grinding apparatus for this purpose was chosen based on a number of qualifications. The grinding environment needed to efficiently shatter the brittle waste materials, minimize abrasion and size reduction of gold particles, and maximize the surface area of gold particles during grinding. Furthermore, the equipment needed to be inexpensive, simple, and accessible to the average miner. Review of the literature implied gold was most effectively flattened in a vibratory pulverizer, though the low capacity and expense of this device made widespread field application impractical. A rod mill became the grinder of choice. Rod mills preferentially reduce larger particles during grinding, minimizing abrasion of small, flat gold grains and maximizing size reduction of blocky waste minerals (Wills & Napier-Munn, 2006). They are also simple to build, maintain and operate for the average miner, making it accessible and easily adopted technology.

Three different testing phases were performed to explore the grinding for extraction concept: Pre-field lab testing, field tour, and post-field lab confirmation testing. The pre-field testing was performed with a 20x20 cm (8"x8") laboratory rod mill at the University of Alaska Fairbanks. During field and post-field tests, a lab sized 20 cm x 30 cm (8" x 12") internal diameter rod mill was constructed, loaded to approximately 40% of its volume with rods varying in diameter from 12 mm (1/2"), 18 mm (3/4"), and 25 mm (1"). The driving mechanism for this mill was a modified wheeled frame originally meant for a cement mixer. This allowed the heavy mill to be easily transported, loaded and discharged by a single person. During field testing, this device was toured around the Yukon placer gold fields to where on-site tests were performed with difficult gravity middlings pointed out by the property operator. Post field lab tests were done on identical bulk samples donated during the field tour to explore the recovery sensitivities of the grinding for extraction method.

The purpose of this research was to test the efficiency of manipulating the malleability, rather than density, of gold to solve the issue of high grade middlings being stockpiled and unprocessed by using a batch grinding mill. This method is meant to be chemical-free and environmentally friendly, requiring no further permitting and is accessible to the average Yukon miner. This would maximize the gold extraction of miners in a time where dwindling profit margins are threatening the Yukon placer industry.

Chapter 2 - Literature Review

Grinding extraction is focused on using comminution equipment and sieve classification to extract placer gold from difficult gravity middlings in placer mines. As such, the literature review focused on establishing knowledge in placer deposit formation, placer ore processing techniques, the need for maximizing gold extraction in the Klondike placer fields, gold malleability, and the reaction of gold particles to differing grinding environments. Firstly, understanding the behaviour and formation of economic placer deposits was necessary.

Placer deposits are economic deposits of "residual or detrital mineral grains in which a valuable mineral has been concentrated by a mechanical agent…usually running water, and the valuable mineral is usually denser than quartz" (Slingerland & Smith, 1986, pg 113). Gold is more than 15 times the density of quartz, and this difference in relative density causes natural concentration during fluid transport through gravity and hydrological sorting. It is a curiosity that more science is not dedicated to this type of deposit, as placers have boasted the highest gold grades in history (Slingerland & Smith, 1986). The famous Witwatersrand gold deposit is a preserved paleoplacer which has provided over half of all gold mined in the world from this single locality (Slingerland & Smith, 1986). The formation of one of these deposits is dependent on many interdependent factors, such as stream evolution, tectonic history, local geology, physiography, and climate.

Commercial gold deposits typically boast the highest grades within a few feet of bedrock (Tuck, 1968). This is reflective of the basic placer deposit formation mechanism, wherein immature dominantly downcutting streams erode gold and other minerals from the surrounding bedrock. The high density gold and minerals are concentrated into the bottom of the valley due to their difference in relative density compared to waste materials. As the stream is downcutting into bedrock, gold particles tend to be concentrated along the bedrock surface at the bottom of the channel, where particles greater than one milligram typically remain (Tuck, 1968). Consider a downcutting creek which represents 1,500 m of erosion. Placer gold resists transport to the point that it would move less than 3,000 to 4,500 m from the erosion point (Tuck, 1968, pg 192). Gold's particularly high density, as well as chemical stability, allows it to endure over the long erosion periods required for placer deposit accumulation and resist being broken down. During this time, gold particles are only smoothed, rounded, and further liberated from surrounding rock (Tuck, 1968). Less stable waste minerals are more susceptible to abrasion and comminution during creek transport than the gold particles (Slingerland & Smith, 1986).

Characteristically, gold concentrates in a narrow streak where stream downcutting ceased. Enrichment typically occurs where stream flow is diverted or slowed, encouraging dense material to settle out. Settings such as bar heads within meandering flow or bar margins along stable banks are common accumulation concentration areas (Figure 2.1). Bars tend to decrease in particle size from head to tail, and the coarse rough bar head is more ideal for dense mineral entrainment. The bar head is also continuously eroded and becomes enriched as dense materials are resistant to transport (Slingerland & Smith, 1986). Enrichment occurs on bar margins at stable banks due to the diversion of the main flow rolling light minerals up to the bar, while heavy minerals resist this motion and drop out along the bar margin. Despite these typical scenarios, deciphering deposition history is not so simple. Deposition is complicated by multiple temporary base levels, flooding, scouring, variations along the stream profile in aggradation and degradation, and glacial events.

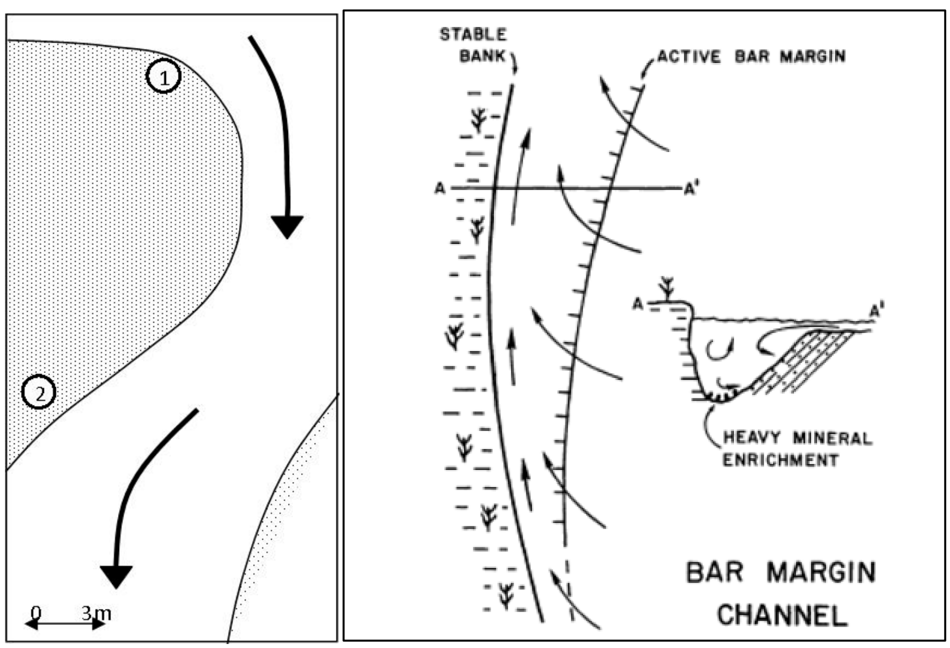


Figure 2.1: (Left) Bar head enrichment. (1): Highest concentration of heavy minerals. (2) Lowest concentration (modified from Slingerland, 1984, pg 147). (Right) Bar margin enrichment along stable bank (from Slingerland & Smith, 1986, pg 141).

At the grain scale, heavy mineral enrichment occurs through a combination of density and hydrological factors. It is interesting to note that much of these natural concentration conditions are being imitated by modern gravity concentration equipment, discussed later. It is intuitive to expect that settling velocity, or suspension sorting, is the controlling factor. This is the differential speed at which minerals sink in a fluid based on density. However, this is the least important mechanism. A large grain is often hydraulically equivalent to a smaller more dense grain, and as dense mineral crystal sizes typically occur in smaller size fractions than lighter ones, settling velocity alone is not enough to create commercial placer deposits (Slingerland & Smith, 1986). It is actually a combination of settling velocity and three other main grain scale sorting processes: Entrainment, shear and transport sorting. Entrainment sorting is separation based on grain size, density and shape based by differential "pick up" off of a bed (Slingerland, 1984). Lighter, commonly larger grains protrude further into the moving stream waters, making them more susceptible to being removed from the stream bed than dense minerals. Shear sorting separates grains into different horizons due to dispersive pressures arising from grain collisions in turbulent flow. In a moving granular layer subject to fluid shear, the dispersive pressure acts in a vertical direction away from the bed, which forces the largest and densest grains to the surface of the flowing grain layer (Hughes, Keene, & Joseph, 2000). Shear sorting is one of the mechanisms modern jigs employ for concentration. Transport sorting, the most important mechanism, separates light and heavy minerals based on relative grain transport velocities, dependent on grain size, density, shape, flow velocity and bed roughness (Slingerland & Smith, 1986). Dense grains with high entrainment potential travel the least distance compared to light ones, encouraging heavy mineral concentration.

Commercial placer deposits only accumulate under ideal conditions. A bedrock source of gold, downcutting creeks, and the right combination of hydrologic and geologic conditions to allow the enrichment and preservation of gold-bearing pay gravels must be present. These conditions are present in the Yukon gold fields to a degree that has sustained an independent gold mining industry for more than 100 years. However, dwindling profit margins have made maximizing the potential of gold bearing gravels more essential than ever.

There are eleven defined Yukon placer mining areas (Figure 2.2). The majority of gold production (i.e. currently and historically) has been in areas 1 to 3; near the famous Dawson City area that hosted the gold rush of the 1890s. Gold mining has been ongoing here continuously since that time. The placer gold industry has been able to weather economic downturns like no other Yukon mining enterprises. Unlike hard rock projects, the majority of placer operations are independently owned and operated (Van Loon & Bond, 2014). During economic stress, miners do not have the choice to abandon their projects until the markets are more favourable, and must either produce or perish. This has made the placer industry an important economic mainstay of the Yukon since their discovery. During economic upswings, hard rock exploration has dedicated significant time to exploring these gold fields in an attempt to discover the elusive in-situ gold deposit that sourced the creeks, but so far have proven unsuccessful. It is possible that the original deposit has been fully eroded over geologic time, such as at the gold fields of Nome, Alaska (Tuck, 1968). Despite their economic resistance, gold production and claim staking trends in recent years have indicated that is more crucial than ever to maximize gold extraction from claims in times of dwindling profit margins.

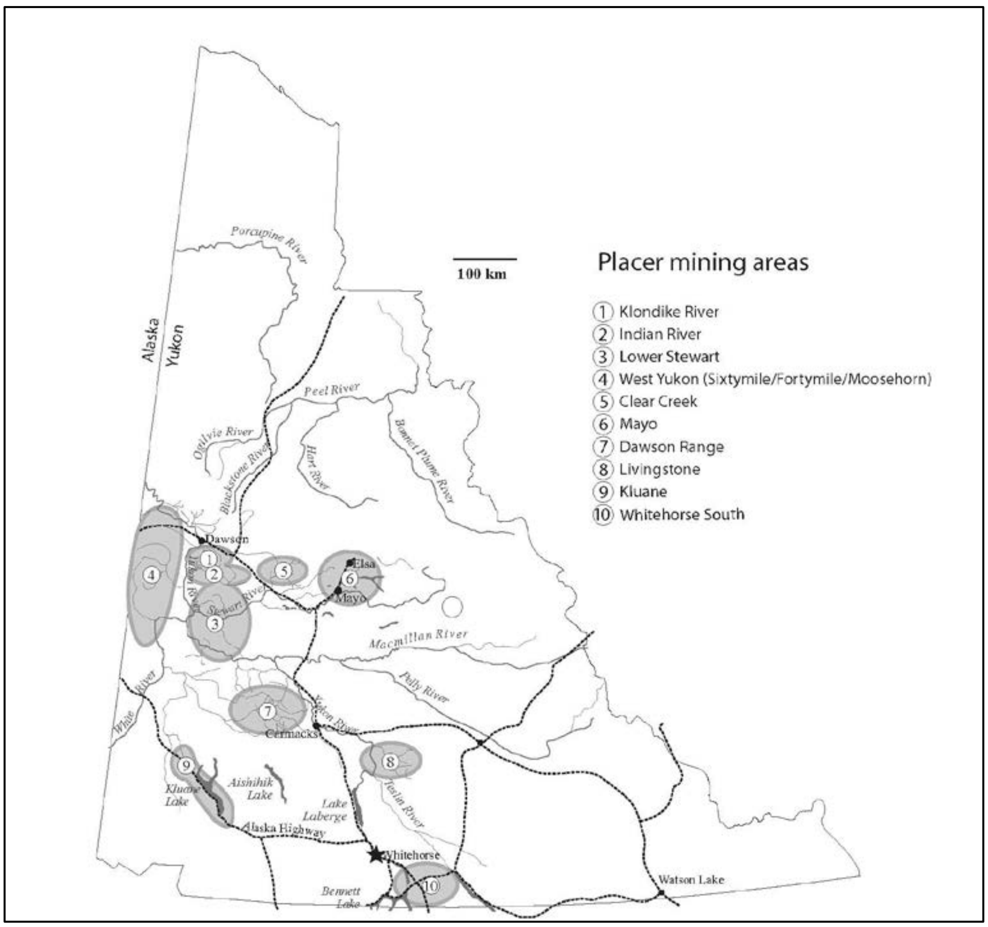


Figure 2.2: Yukon Territory and the main placer producing areas (from Bond, 2012, pg 77).

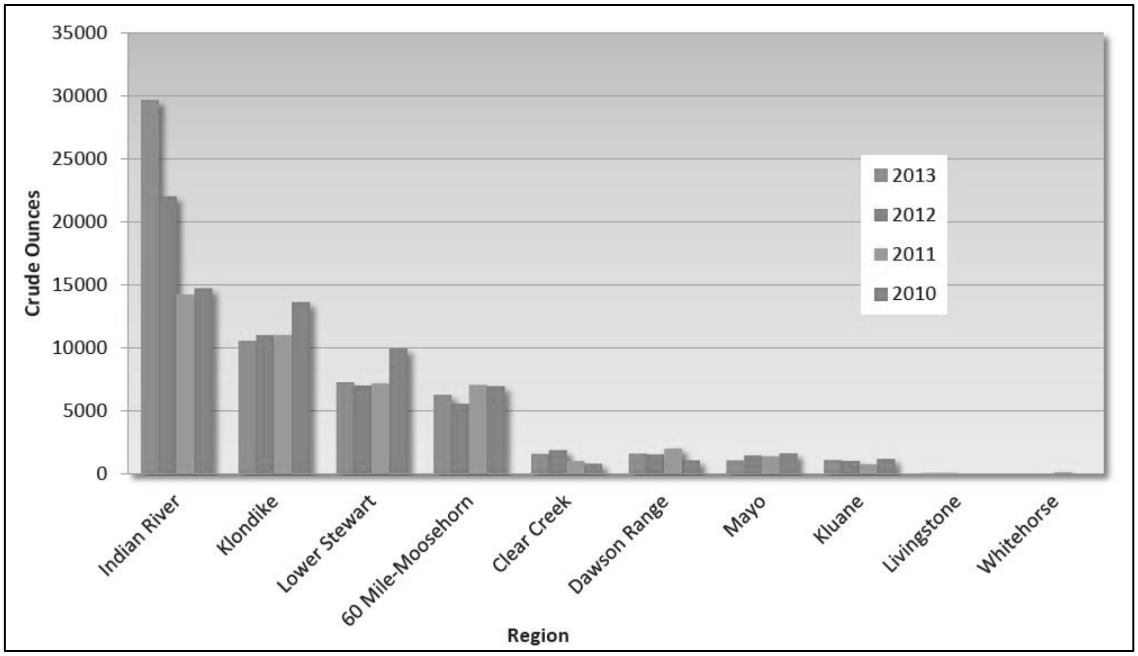


Figure 2.3: Yukon gold production from the major producing regions from 2010 to 2013 (from Van Loon & Bond, 2014, pg 6).

Despite higher gold prices, overall gold production has been on the decline from recent peaks in the mid-1990s (Figure 2.4) (Van Loon & Bond, 2014). Similarly, there has been no matching increase in placer ground staked as a result of increases in gold price, with less annual staking activity now than in 1996 when gold was just $386/oz t (Van Loon & Bond, 2014). This is indicative of declining gold grades and increasing costs related to access to deposits and high fuel prices (Van Loon & Bond, 2014). Narrowing profit margins have also effected placer exploration, leaving less money available for ambitious projects (Van Loon & Bond, 2014). Miners are forced to be conservative and typically explore near existing operations where costs are low and the discovery potential is very high. There is little attention paid to more expensive exploration ventures such as frontier valleys with no prior history. These areas have a lesser chance of significant discoveries, and many independent miners are hesitant to be the first to take the risk. Maximizing efficiency and recovery is more important than it ever has been in the Yukon placer industry that only uses gravity processing methods (Van Loon & Bond, 2014). Although gold recoveries start to drop as the concentrates become progressively enriched in high density waste materials, under ideal conditions the varied gold recovery equipment is remarkably efficient.

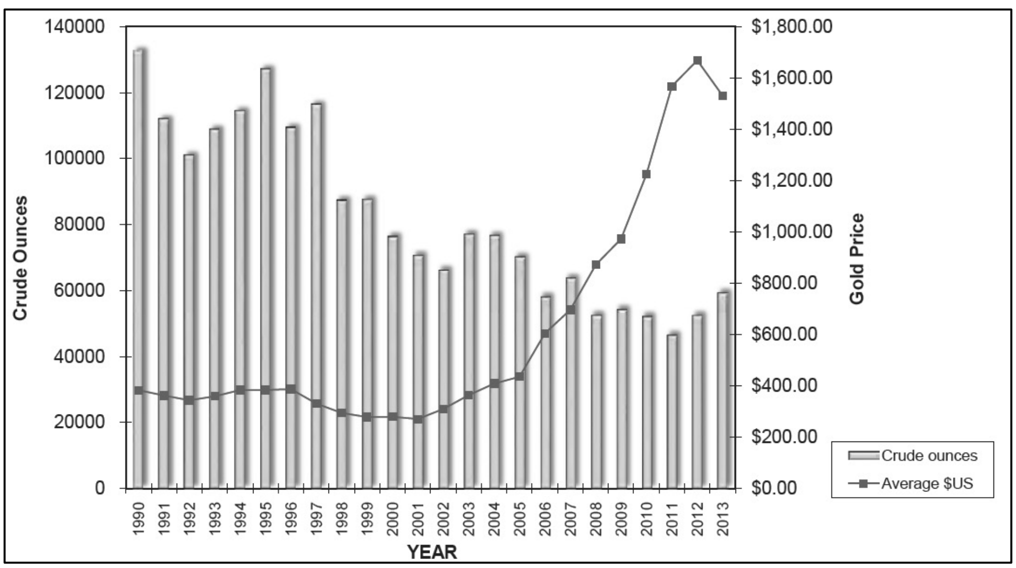


Figure 2.4: Annual Yukon crude ounce production compared to average annual gold price (from Van Loon & Bond, 2014, pg 5).

Mineral processing by gravity concentration is simply manipulating the relative difference in density of ore and waste for ore extraction, typically by separating high density minerals within a fluid medium. Due to its simplicity, gravity processing is one of the oldest forms of mining, with simplistic jigs and sluices in use since antiquity (Burt, 1987). As gold containing placers have already been enriched by naturally occurring gravity based processes, placer miners rarely find the need to process their ores using anything other than density-based separation equipment. When used properly, ore processors such as sluice boxes, tables, wheels and jigs can recover greater than 90% of the contained gold.

In Yukon placer mining the ore is initially concentrated with a riffle lined sluice box. At its most basic, this is a rectangular flume through which a dilute slurry of water and pay material flows. Assorted steel grating, also known as riffles, line these flumes on top of dense, tangled matting. These riffles create areas of turbulence, or vortexes, which encourage the settling of heavy minerals into the coarse matting, where it is retained until collected for further processing. These devices are the oldest known mechanical processing units in human history. While the basic premise is the same, modern units are capable of 40 to 250 m^3/hr, concentration ratios of 50,000:1 and recoveries exceeding 95% down to 180 µm (Clarkson, 1996, pg 59; Hamilton, 1988, pg 3). The simplicity and reliability of sluice boxes have maintained their use for thousands of years, and in their modern capacity are the most popular primary concentrators in the Yukon placer industry (Clarkson, 1996; Clarkson, 1998; Shiman, 2005; Van Loon & Bond, 2014). High recoveries are obtained when the installed riffles are performing effectively, which are designed to imitate natural conditions of concentration.

The basic purpose of sluice riffles is to retard high density gold that has sank to the bottom of the process slurry and force it into the matting. The riffles form vortexes of flow about the horizontal axis that classify this gold from the sand and force heavy particles to remain in the mats (Burt, 1984). In the Klondike gold fields, the most commonly used riffles can be separated into two categories: Angle iron and expanded metal (Figure 2.5). Both of these riffles create cascading vortexes that drive high density particles into the matting beneath (Figure 2.6). Spacing, slurry speed and slurry density are all critical factors in gold recoveries of these riffles, for if the vortex is too strong it could scour out the matting below, and if too weak will not retain gold particles (Clarkson, 1996). Both sets of riffles are often used in tandem as expanded metal riffles are more appropriate for smaller gold particles (< 1mm) and angle iron for larger ones (> 1mm) (Clarkson, 1994, pg 31).

Despite high concentration ratios, the primary concentrate is not of sufficiently high grade for a sellable product and invariably requires further upgrading. To emphasize this with an example using rough averages, imagine a plant capacity of 100 m^3/hr processing satisfactory average grades of 0.5 g/m^3 gold (Van Loon & Bond, 2014, pg 6) running for 48 hours before primary concentrate collection at a reduction ratio of 1:50,000 (Clarkson, 1996, pg 59) and a material density of 3 g/cm^3. For simplicity recovery is 100%. This would produce 288 kg of primary concentrate containing 2,400 g of gold, or a grade of 0.83%.

Figure 2.5: Sluice run with expanded metal (upstream) and angle iron (bottom left) riffles, separated by flat slick plates (photo credit Gavin Clarkson 2014).

During interviews with miners at testing sites, they indicated they preferred selling very clean concentrates (typically >70 wt% gold) to avoid high melt losses and the resulting uncertainties in smelted gold grades and payments. Although local smelters are capable of processing concentrate as low as 50 wt% gold, higher grades are preferred to avoid any controversy with clients. In order to achieve this concentration, the primary ore must be further upgraded. This is done in the gold room using a combination of water and gravity based separation equipment. The practice of using flotation or cyanide for concentrate processing is non-existent and even the simple technique of mercury extraction has not seen prominent use in the Klondike for many years (Osler, 1983).

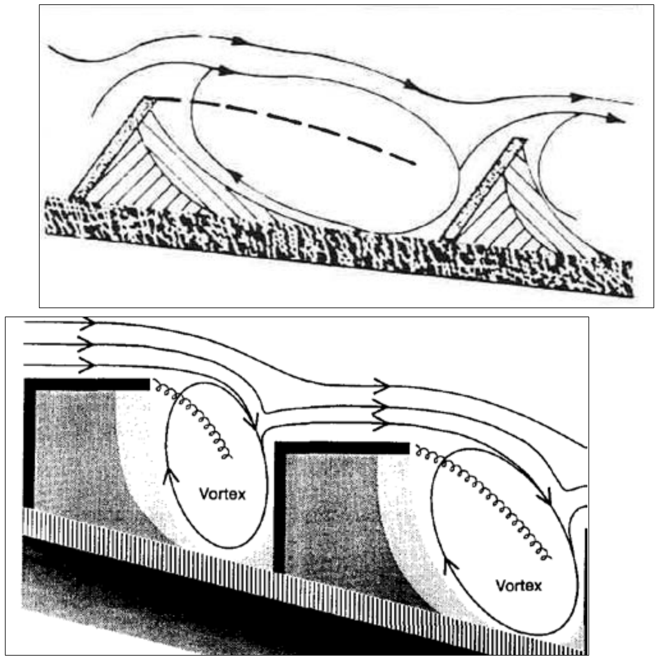


Figure 2.6 (Above) Cross section of expanded metal riffles (from Clarkson, 1994, pg 30). (Below) Cross section of angle iron riffles (from Kelly, Subasinghe, & Walsh, 1995, pg 1406).

Mercury, cyanide or other chemical extraction techniques are not used in the Yukon gold fields due to environmental concerns, lack of expertise, and a strict permitting regime. The most recent official survey of mercury use in the Yukon and northern British Columbia placer mines found of 260 mines surveyed only 60 had mercury in their possession, of which only 36 planned on using it for gold recovery (Osler, 1983). More than 30 years ago, “the art of mercury for amalgamating appears to be dying [in the Yukon]” (Osler, 1983, pg 82). Reasons cited for the dismissal of a once common gold upgrading technique included fear of toxicity, environmental contamination, increased cost in mercury acquirement and the increased recovery capability of new sluices and other gravity concentration equipment (Osler, 1983). Mercury extraction of gold in Yukon placer is certainly a thing of the past, though it is important to note it is still abundant in small scale gold extraction in the developing world (Hinton et al., 2003). Some small scale gold miners in less developed countries have also begun to apply cyanidation and flotation gold processing, the capital and operating costs subsidized by larger mining companies (Hinton et al., 2003). This extraction is more applicable for developing countries where artisanal miners often mine gold in-situ from weathered gold-hosting veins (Hinton et al., 2003). Placer deposits, like those in the Yukon, are typically coarse, well liberated, and naturally gravity concentrated and recoverable, making more expensive chemical extraction unnecessary (Mitchell, Evans, & Styles, 1997). Because of this, upgrading is strictly limited to further gravity concentration. The most commonly used upgrading devices are jigs, concentrating tables, spirals and long narrow sluices also known as "long toms". Each of these devices has their own specific application and limitations (Table 2.1).

Table 2.1: Applications of common gold upgrading machinery in the Yukon goldfields (compiled from Laplante, Putz, Huang, & Vincent, 1994; Mitchell et al., 1997; Silva, 1986; Wills & Napier-Munn, 2006).

	Size Range	Recovery	Main Application	Gold Type
Shaker Table	3mm to 15µm	>90%	Cleaning	Flat, fine
Gold Wheel	3mm to 75µm	>95%	Cleaning	Coarse, round
Jig	25mm to 150µm	>95%	Cleaning/Upgrading	Nuggets to fines
Long tom	>1mm to 35 µm	>95%	Upgrading/cleaning	Nuggets to fines

A common first upgrading step is to run the primary sluice concentrate over a smaller recovery sluice, typically narrow (approximately 30 cm wide), ranging from 3 to 6 m long and referred to as a long tom (Figure 2.7). There is little published literature focusing on this common device, though concentrates reviewed informally by the British Geological Survey found high recoveries (>90%) down to 35 µm gold grain sizes (Clarkson, 2015). Furthermore, testing at the UAF ran long tom tailings over a shaker table to discover losses not exceeding 2% (Clarkson, 2015). These devices are commonly used as the first upgrading step, and the first 30 cm or so of the inclined sluice often has gold clean enough for direct sale (Figure 2.8).

Figure 2.7: Long tom operation in a Klondike gold room (photo credit Gavin Clarkson 2014).

Figure 2.8: Top section of a long tom with clean gold packing the riffles (photo credit Gavin Clarkson 2014).

One of the most common gravity upgrading devices in the Yukon is the concentrating table (Laplante et al., 1994). These are effective in particle size ranges from 3 mm to 15 µm, with recoveries greater than 90% at 40 µm (Mitchell et al., 1997, pg 9). An inclined riffled deck is driven by asymmetrical acceleration along its long access while water flows over the short axis, driving material across the deck and sorting by density. The shaking action encourages size and density stratification behind the longitudinal riffles, and heavier particles trickle to the bottom of the material beds gathering behind the riffles. Lower density and coarser particles are washed to the lower side of the table and tailings. Good classification and even feed is essential to the performance of tables (Silva, 1986; Wills & Napier-Munn, 2006).

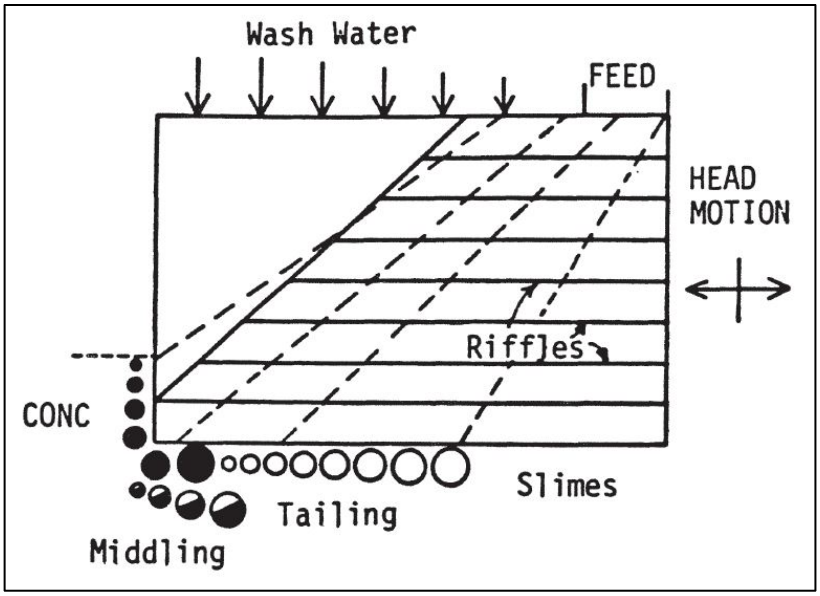


Figure 2.9: Particle paths of high density (dark) and low density (light) particles on a concentration table (from Burt, 1987, pg 125).

Figure 2.9 illustrates the travel paths of particles across a table surface. The particles that become the concentrate are typically of a high enough grade for smelting or sale. A higher grade concentrate requires rejecting more material to create a lower yield resulting in lower recoveries, and a significant portion of gold reports to the middlings (Figure 2.10) (Laplante & Gray, 2005). During processing on the concentrating table, all lower density materials are washed away to tailings, so middling density is significantly higher than the original feed. Tables are more suited to finer, flatter gold particles as coarse or round gold tends to roll into the tailings.

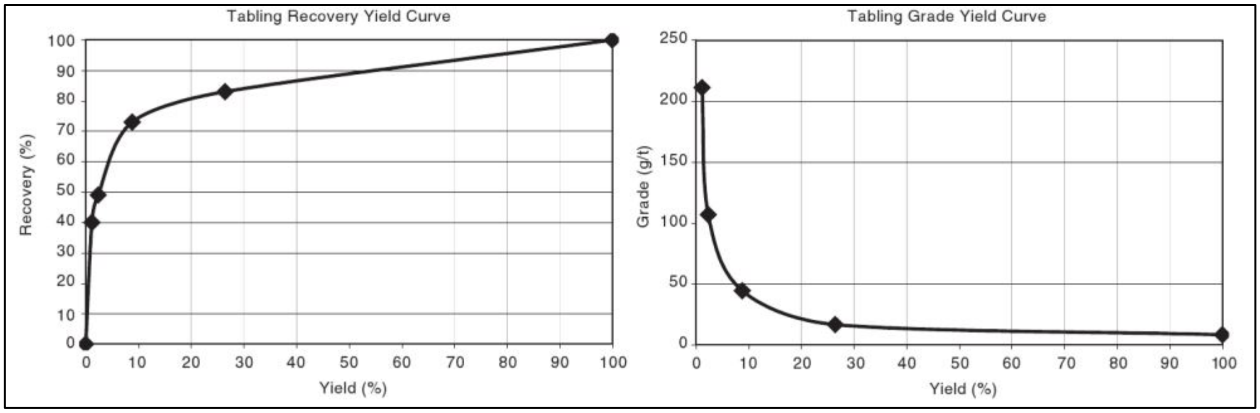


Figure 2.10: (Left) Tabling recovery increases as concentrate yield increases. (Right) Tabling concentrate grade decreases as yield increases (from Laplante & Gray, 2005, pg 286).

Gold wheels are especially common in placer projects and are often used in series with concentrating tables. They are also known as rotating spirals or rotary tables (Silva, 1986). Wheels consist of a shallow inclined rotating spiral with a film of water running down from a spray bar placed along the wheel's radius. The rotating spirals collect higher density particles on the riffles while lighter (i.e. lower specific gravity) material is washed below to tailings. High density, rounder particles are retained and collected at the centre of the wheel in response to the rotating motion. These units are low capacity and require constant attention, so their use is mostly limited to placer operation. Limited testing has indicated recoveries exceeding 95%, and they produce a very clean gold concentrate (Silva, 1986, pg 15). Rounder particles respond well to a concentrating wheel, and as such many gold rooms will use a gold wheel in conjunction with a concentrating table. Like most processors, they perform best under certain conditions, and a high proportion of flattened gold or high density gangues will result in significant portions lost to tailings (Table 2.1).

Figure 2.11: Gold wheel or rotary table concentrator. Concentrate is collected through the hole in the centre; tailings wash out over the bottom lip (photo credit Gavin Clarkson 2014).

Small water activated jigs are also fairly common in Yukon gold rooms. These devices are effective from 25 mm to 150 µm, with recovery reported to fall below 50% in sizes below 100 µm (Mitchell et al., 1997, pg 9). Jigs operate on the basis of ore particles being suspended in a layer on a perforated plate or screen, and alternating rising and falling fluid flows repeatedly dilate and close the bed. This allows high density materials to travel through the particle bed to the bottom and through the perforated plate into the concentrate collector, or hutch. Often within the ore layer is a layer of “ragging”, or heavy particles at the bottom of the bed just larger than the screen perforations and of a density between that of the waste minerals and gold (Miller, Demull, & Matorey, 1985). This encourages separation of waste from ore. A stream of water running over the concentrating layer washes away lower density particles that migrate to the top of the concentration bed. In Yukon placer operations, there are often two jigs in series with the tailings of the first feeding the second and a long tom recovering the -150 µm fines. Fine gold is concentrated in the hutches, and coarser nuggets are concentrated over the perforated screen at the bottom of the ragging. Clean up jigs generally require consistent attention, and separating the nuggets from the jig ragging is time consuming. They have a higher capacity but create lower grade concentrates when compared to wheels or tables and are often used in series with these other technologies. Material from a second hutch in the series is especially high density and low grade, with the easiest recoverable gold particles having been retained in the primary hutch.

Each of these devices is well suited to upgrading appropriate concentrates, and is often used in series to maximize recovery of different gold particles. However, to create a high grade final concentrate requires sacrificing a significant portion to middlings. These middlings are saved but are often too unclassified and too dense to separate through further gravity processing. As a result they are stockpiled and saved for hand picking and other manual extraction techniques which are very time consuming. But high density is not the only unique property of gold. Relative differences in malleability, instead of density, could be exploited for extraction.

Gold possesses a high degree of malleability and chemical stability (Grimwade, 1992). Historically, its color, high reflectivity, and occurrence in the Earth's crust as a native metal made it one of the very first mined commodities and laid the framework for currency and metallurgy (Grimwade, 1992). In modern use, gold's nobility and conductivity have made it essential in electrical applications where long term reliability and lack of tarnishing are required. Gold's malleability is the greatest of all of the metals, with the ability to be reduced in thickness far beyond other native elements like copper or silver. The metals high malleability is partially related to its unique low reactivity, as well as the efficiency of its cubic crystal structure (Nutting & Nuttall, 1977)

The crystal lattice of gold is face centred cubic, the most efficiently packed cubic unit cell. In plastic deformation, crystal lattice defects allow the movement of lattice dislocations throughout the solid structure to accommodate shape change (Petch, 1954). This slip occurs along preferred crystal planes and preferred crystal directions, emphasizing planes and directions within the lattice with the closest packing of atoms. Since only the hexagonal lattice matches the face centred cubic lattice in closeness of packed atoms, this structure is most amenable to slip deformation, with twelve possible slip systems that can operate during plastic deformation (Grimwade, 1992, pg 372). Hexagonal close packed lattices and body centred cubic systems can also accommodate slip in this way, but the planes are not so closely packed and therefore malleable slippage is more difficult. The face centred cubic metals generally have good malleability and ductility and include silver, copper and lead (Grimwade, 1992). Despite occurring with the same efficient cubic crystal structure, these metals fail under strain long before gold does. This unique property of gold is related to its low reactivity.

Gold is the most malleable of its metallic face centred counter parts. This ability is related to gold as a noble element, being the least reactive of all elemental metals. It can be rolled and beaten to widths less than the wavelength of visible light, up to a 99.9996 % reduction of thickness (Nutting & Nuttall, 1977, pg 2). It has been suggested that the lack of an oxide coating due to gold's nobility is what allows this to occur. This film can hold lattice dislocations within the metal, causing build up and failure. Gold lacks an oxide film that allows lattice dislocations to escape to the surface, without preferential deformation along subgrain boundaries leading to failure (Nutting & Nuttall, 1977). Other metals tarnish and oxidize readily, creating a microscopic surface boundary which seizes lattice dislocations. During deformation, instead of the stress being released to the outside of the metal, these dislocations accumulate until no further shape change can occur, at which point failure occurs. This high malleability, along with characteristic density, make gold significantly resistant to being broken down and reduced in size. Not only does this resistance allow the accumulation of placer gold deposits in highly active creek beds, it also makes gold particles resistant to comminution in modern mill circuits.

High density and malleability makes gold very difficult to grind compared to other minerals (Figure 2.12). The unique response of gold to mill grinding circuits is a well-documented phenomenon (Banisi et al., 1991; Noaparast & Laplante, 2004; Ofori-Sarpong & Amankwah, 2011). Its unique attributes lead to high recirculating loads in conventional grinding circuits, affecting its breakage, classification and liberation. Coarse gold has been found to grind 6 times slower than ore and up to 20 times slower in certain size classes, with finer gold reaching high survival rates in grinding circuits (Banisi et al., 1991, pg 78). Modeling gold's behaviour in grinding circuits is difficult, as not only does it require longer grinding than conventional ore, grains can be flattened, cold welded into coarser size classes or be smeared onto other minerals or mill linings (Banisi et al., 1991; Noaparast & Laplante, 2002; Noaparast & Laplante, 2004; Ofori-Sarpong & Amankwah, 2011). Issues arising from gold recirculating in circuits can include recovery losses, poor distinction of head grades or gold inventory, and security risks especially when coarse gold is involved (Banisi et al., 1991). Plastic material consumes energy via changing shape instead of fracturing, a behaviour which ignores comminution theory equations and can make mill modeling unpredictable (Wills & Napier-Munn, 2006).

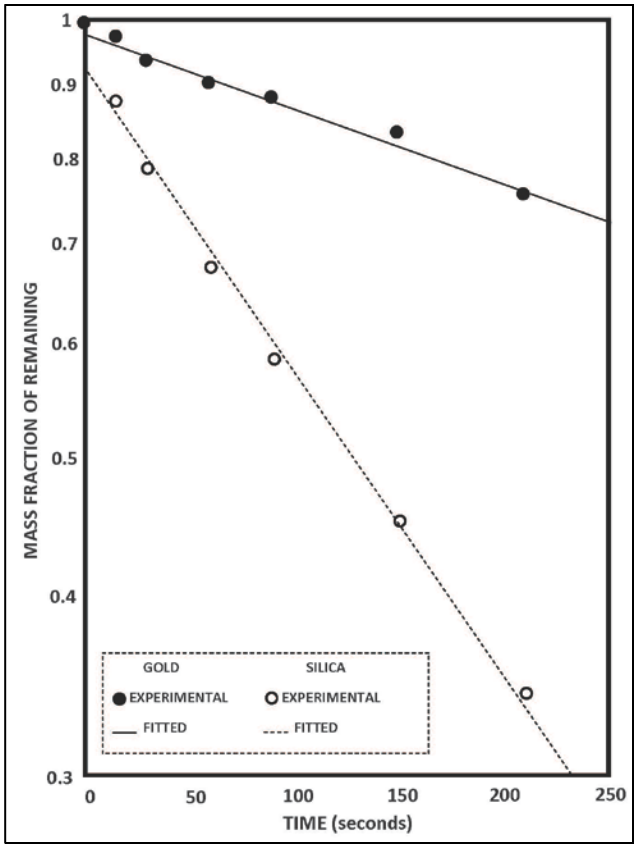


Figure 2.12: Mass fraction remaining in 840-1200 µm size range as a function of time for 5 g of gold and 50 g of silica sand (from Banisi et al., 1991, pg 73).

Noaparast & Laplante, 2004, conducted a test on the breakage functions of gold particles in the Hemlo grinding circuit. They pointed out that although gold malleability can wreak havoc in terms of grind predictability, recirculating gold grains have a greater chance of being recovered during many passes through processing equipment. This study started with gravity concentrate recovered by a mill circuit Knelson concentrator. These concentrates are ground sequentially, maintaining initial sample weight with silica, and specific size classes chosen and removed. These separate size classes are processed again within the Knelson to determine the effect of grinding on the contained gold particles. Material coarser and finer than the initial grind size was removed every 30 seconds, and this process repeated 10 times, meaning no gold was exposed to grinding longer than 5 minutes total. It was observed that unlike brittle minerals and ores, free gold particles had the ability to flatten without losing any weight and report to coarser size classes. Other ores always report to finer size classes after grinding. Furthermore, gold particles that did report to the undersize were often folded rather than failing. This means further grinding of folded gold grains has the potential to flatten it, again increasing its classification size. Figure 2.13 illustrates gold grain reactions to grinding, with some particles flattening and increasing in size class with no change in weight, as well as reporting to a finer size class with no change in weight through folding.

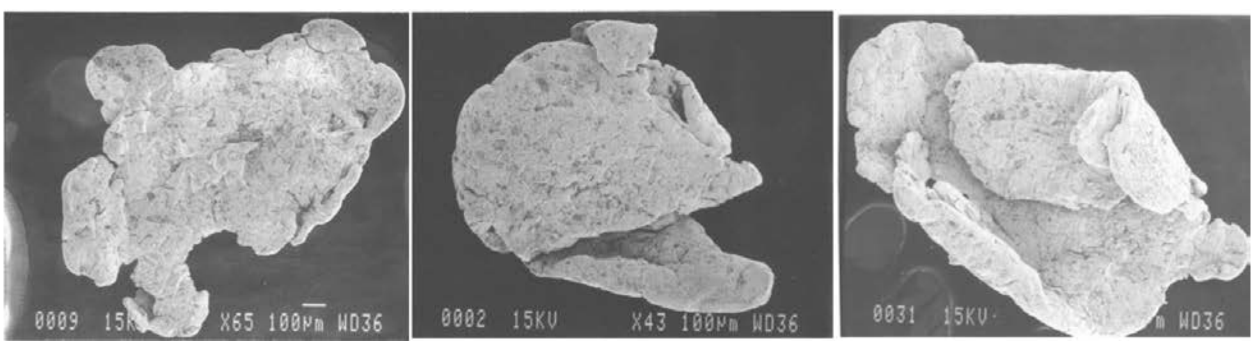
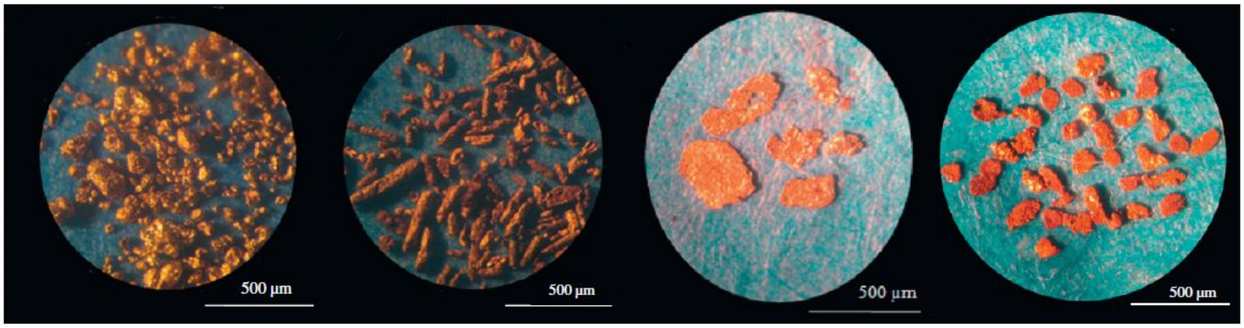


Figure 2.13: SEM of gold grain reactions when subjected to grinding. (Left to right) Increase in size class, remained in original size class, and decrease in size class (from Noaparast & Laplante, 2004, pg 676).

Since the malleability of gold makes it resistant to grinding, the particles are preferentially preserved compared to other common brittle gangue minerals including galena, sphalerite, pyrite or hematite. These relative differences can be exploited by submitting a high value concentrate to grinding and separating the pulverised gangue from the preserved gold particles through sieving. The finest size that can be amenable to this extraction is limited by the nature of brittle and ductile materials. Ductile behaviour of a material is independent of the particle size, as the plastic deformation is being accommodated within the particle volume by molecular scale dislocations (Austin & Trass, 1997). However, brittle materials become more resistant to grinding as larger particle flaws are progressively removed during particle reduction. With less large scale imperfections, the brittle material's strength becomes more dependent on the stronger molecular bonds and is progressively more difficult to grind (Austin & Trass, 1997). Conversely, the rate of stress application is more important for ductile than brittle materials, as rapid application of stress can cause failure of an otherwise ductile material, whereas slow application would allow time for ductile strain. For the purpose of grind extraction, the minimal size range recoverable is potentially limited by the increasing resistance of brittle particles.

The comminution device appropriate for extracting gold via grinding needed to be both effective and simple enough for widespread application. Field tests were designed to recover coarse gold on a +50# screen, so the grinding environment must be capable of effectively reducing the gangue materials to at least -50# size while avoiding the fragmenting of the original gold particles. Flattening of gold is most desirable, as this increases the surface area for recovery via sieves. Producing coarse flat gold is generally avoided as it reduces the effective specific gravity for gravity recovery, but this material already does not respond to further gravity upgrading methods. As such, reducing the gravity recoverability of the gold particles is of no consequence, as recovery is now based on relative differences in malleability, not specific gravity. Previous work has explored the varied particle shapes of gold when exposed to different grinding environments, the results of which can be used to choose the most appropriate grinding apparatus for grind extraction of gold.

Ofori-Sarpong & Amankwah, 2011, conducted a study on the response of gold grains to different comminution environments at a lab scale. The authors intended to better understand the resulting grain shapes of gold after varied mill grinding environments in order to better select the downstream processes based on the resulting grain shape. The purpose of this study was to better understand the response of gold to grinding apparatus, but stopped short of manipulating this malleability for recovery. After gold-containing ore was crushed to -25 mm in a jaw crusher, the material was then dried and treated separately in a lab sized disc mill, ball mill, vibratory pulveriser and hammer mill. The resulting shapes can be seen in Figure 2.14.



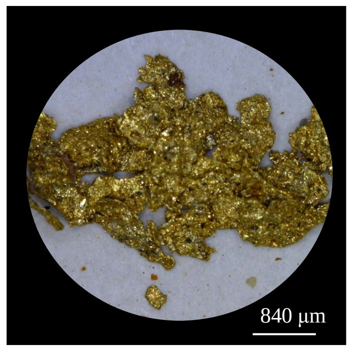


Figure 2.14: (Above) Gold particle shape after grinding in (left to right): hammer mill, disc mill, vibratory pulveriser, ball mill (from Ofori-Sarpong & Amankwah, 2011, pg 591). (Below) Gold particles ground in rod mill from this thesis (photo credit Gavin Clarkson 2014).

Each piece of equipment used employs differing forces on the materials. The hammer mill employs mainly impact forces, disc mill shear forces, and vibratory pulveriser compressive forces. The ball mill employs a chaotic combination of forces, including impact, compression, chipping and abrasion (Ofori-Sarpong & Amankwah, 2011). Their study revealed that for ore gangue, the finest grinds were produced by the vibratory pulveriser, followed by the ball mill, disc mill and coarsest grinds produced by the hammer mill. The gold particle size distribution was significantly different, with the vibratory pulveriser producing the coarsest grains that had been flattened by the dominantly compressive forces employed.

The hammer mill gold grains were globular and preserved their original nugget shapes, likely having had little contact with the hammers in the tested size range. The disc mill produced cigar-shaped grains that had been rolled in the shear forces of the discs, and the ball mill produced irregular, flat shapes. Figure 2.15 compares the sphericity of the gold particles produced by each grinder using the corey shape factor (CSF), a measure of aspect ratio dividing the largest particle diameter by the square of the product of the intermediate and minimum diameters (Ofori-Sarpong & Amankwah, 2011, pg 591). A CSF of 1 is a perfect sphere, while lesser numbers represent flatter particles.

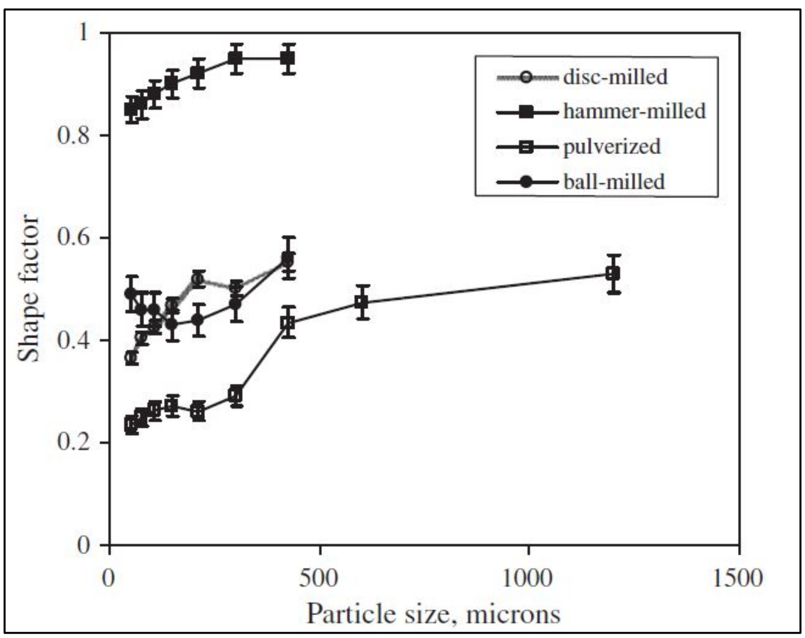


Figure 2.15: CorShape factor vs. particle size for gold grains in each grinding environment. Pulverizer produces the flattest grains across all sizes (from Ofori-Sarpong & Amankwah, 2011, pg 592).

This previous research has emphasized the resistance of gold particles to varying types of comminution. It has further defined their response to different grinding environments in terms of the shapes and coarseness produced. It has provided a framework for the relative difference in malleability of gold particles and surrounding gangue materials, but none have seen fit to exploit this difference to extract gold from grind products. Research up to this point has focused on defining gold grain shape to better select downstream gravity or chemical extraction. For placer properties, when gravity extraction fails gold particles are lost, as gravity concentration is the only concentration method used.

From the above results, the vibratory pulveriser would be the most effective at flattening and preserving gold grains for sieve capture while pulverizing the undesirable gangue. However, these grinders are bulky, expensive and low capacity. A simpler and almost as effective device would be a ball or rod mill, as it would be easy to construct and maintain while still producing flattened gold grains (Figure 2.14). Rod and ball mills can be operated wet, meaning expensive and time consuming drying of the mill feed can be avoided. Though not included in the study of gold flake reactions to grinding by Ofori-Sarpong & Amankwah, 2011, rod mills are known to preferentially reduce larger particles with a minimum of fines when compared to ball mills, and as such give a product with a relatively narrow particle size distribution (Wills & Napier-Munn, 2006). Larger particles bear the brunt of the compression between rods, sheltering fine particles from direct abrasion (Figure 2.16). This is also beneficial for preserving flat gold particles which would only be directly impacted once the larger gangue minerals are reduced to less than the flake thickness. Ball mills are better suited to fine crushing due to their greater surface area per unit weight than rods, and grind through random impacts, chipping and abrasion with no preference to particle size, meaning greater chances of gold particles being reduced. For the purpose of grind extraction, rod milling would be the most effective at preserving and flattening gold grains when compared to ball mills.

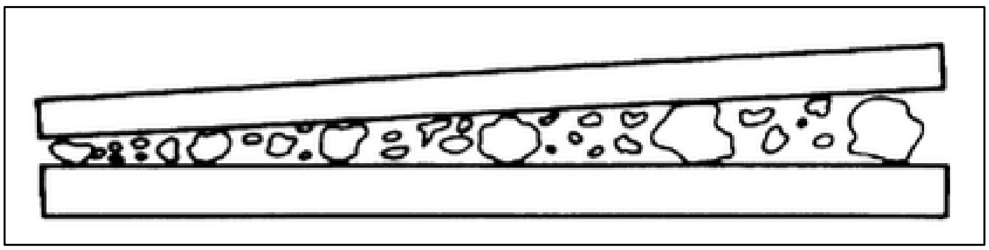


Figure 2.16: Rod mill grinding mechanism. Larger particles are preferentially reduced (from Wills & Napier-Munn, 2006, pg 158).

Reviewing the literature has established a sufficient knowledge of placer deposit formation, as well as the current need to maximize gold extraction and broaden the profit margins of modern Yukon placer mines. The extensive malleability of gold as an element is well established, along with the resistance of gold to comminution especially when compared to other brittle waste materials. Furthermore, previous studies on the resulting shape of gold particles when submitted to varying grinding apparatus have shown that gold particles are best rolled out in vibratory pulverisers. However, rod mills are far simpler and cheaper to build and maintain, with the added benefit of preferentially grinding larger blockier particles over small flat gold grains. Based on this review, the rod mill is the best candidate for field application of the grinding for gold recovery concept.

Chapter 3 - Methodology

There were three main phases of testing the grinding for extraction method: Pre-field lab testing to prove the concept of extracting gold based on malleability, field testing where a grinding apparatus was toured through the Klondike to determine practicality, and post-field lab testing to establish the main recovery sensitivities of this method. Pre-field tests used a 20x20 cm laboratory rod mill, while the field and post field lab tests were done with the same portable rod mill constructed for the site visits. Outlined below are descriptions of the equipment used in each of the test settings, followed by the testing procedures used during each phase. The pre-field and post-field lab settings used similar techniques, though post-field tests used the experience of the field tour to create more reproducible results. Testing results are presented in the following chapter.

First, pre-field lab testing was performed in the fall of 2013 at the University of Alaska, Fairbanks, to establish the efficacy of grinding high density unresponsive concentrates to improve gold recovery. Pre-field tests were performed on concentrates donated from various Yukon placer operators, using a 20x20 cm (8x8") interior rod mill rotated on variable speed pins. This was loaded to approximately 40% by volume with varying diameter steel rods (Figure 3.2).

Figure 3.1: 20x20 cm (8x8”) rod mill used in the UAF pre-field tests.

For field tests, a rod mill of sufficient size to do effective batch samples at multiple mine settings was required. This meant the machine needed to be portable, mobile and easily powered. The rod mill was constructed at a local welding shop in Whitehorse, built to internal dimensions of 20x30cm with an internal calculated volume of approximately 10 L. Grinding media were cold rolled carbon steel bars with diameters of 12 mm, 18 mm and 25 mm, cut 12 mm shorter than the inside of the mill to avoid tangling and seizing. The grinding medium charge was selected to occupy approximately 40% of the internal capacity of the mill (Figure 3.2). Varied sizes were used to allow interlocking of the rods and create a larger effective grinding area while reducing the number of rods needed for between grind clean-up.

Laboratory test mills are commonly laid on a set of counter-rotating rollers or a stationary rotating engine. Given that this device was to be toured to multiple locations, mobility was important. The most convenient option was repurposing the mount for a cement mixer with a centre drive axle, rather than the more common perimeter driven mixers. This also allowed it to be operated with a simple 120 V plugin, common at generator-powered placer mines.

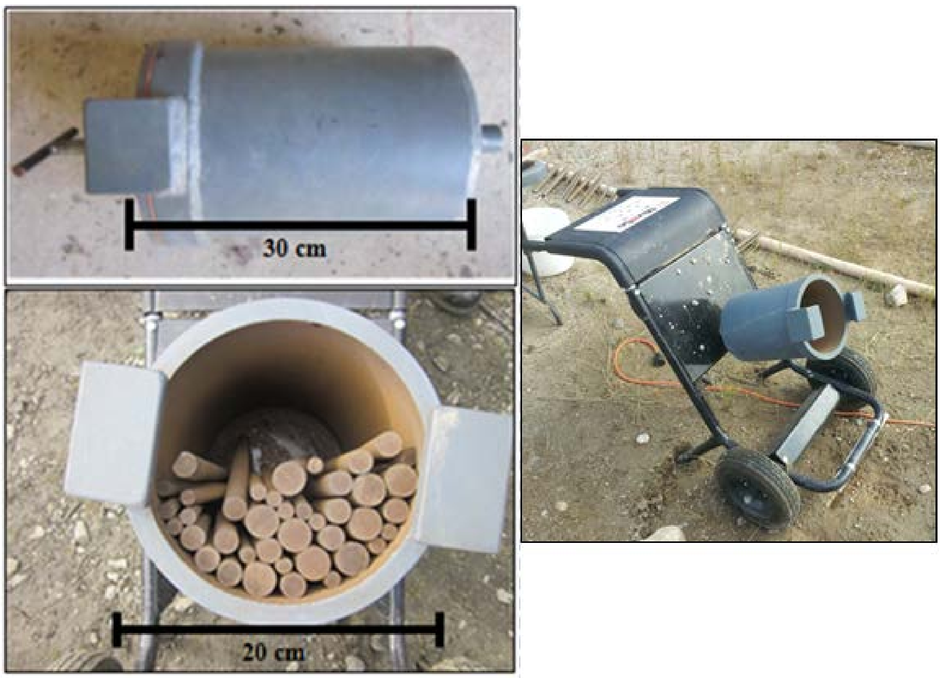


Figure 3.2: The field test rod mill. (Above left) Side view. (Below left) Charged with grinding media. (Right) With cement mixer mount (photo credit Gavin Clarkson 2014).

The mixer provided a convenient way to transport the heavy mill around a test site, as well allowing for a single person to load, manoeuvre and tip out the mill with little assistance.

Cement mixers are typically sold with a single speed setting designed for a cement bowl with a rotational speed far below the requirement for effective rod mill operation. This required switching out the drive pulleys to allow for adjusting the rotational speed. For the purpose of gold grinding for recovery, the mill speed was set for a dominantly cascading motion of the rods allowing abrasive comminution and a finer grind, with limited cataracting grind medium (Figure 3.3). As such, rotational speed was set for approximately 75% of critical.

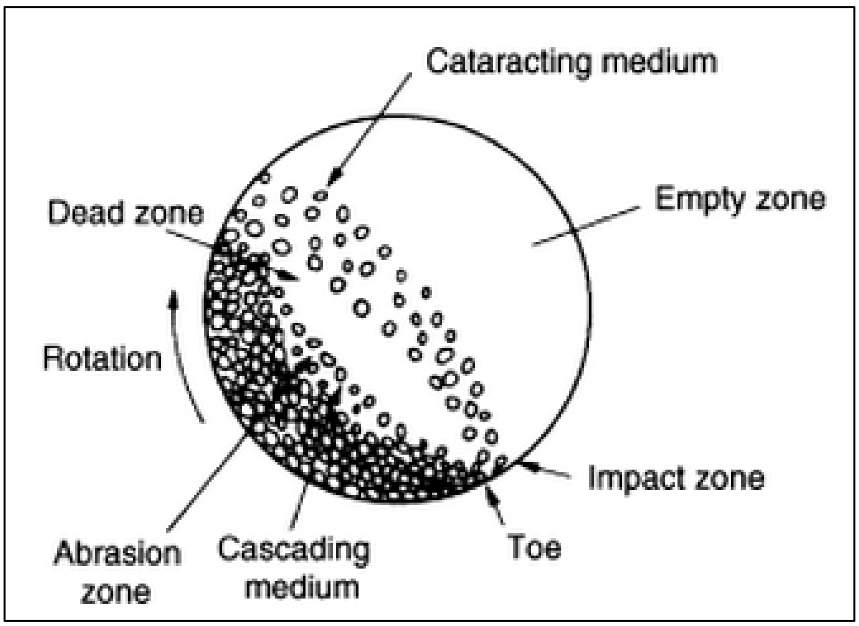


Figure 3.3: Behaviour of grinding medium in a rotating mill (from Wills & Napier-Munn, 2006, pg 147).

The critical speed of the laboratory mill was determined using Equation (3.1), where N_c = critical speed, D = mill diameter and d = grind medium diameter (Wills & Napier-Munn, 2006, pg 148). Since 3 different diameters of grinding medium are used, the average critical speed using all three diameters was determined. As illustrated in Table 3.1, this is calculated as 72 rpm. This is considerably faster than the default 25 rpm setting of the cement mixer used.

$$N_c = \frac{42.3}{\sqrt{D-d}} \text{ rev min}^{-1} \tag{3.1}$$

Table 3.1: Critical speed lab mill calculations.

Mill diameter (m)	0.2	0.2	0.2
Medium diameter (m)	0.012	0.0018	0.0025
Critical Speed (rpm)	98	95	95
Average Critical Speed	96	75% Crit Speed	72

To set the proper operational speed of the mill mount, the drive pulleys were switched from a 15 cm (6”) to a 11.4 cm (4.5”) fixed pulley on the gearbox, and from a 4.4 cm (1.75”) to a 10.2 cm (4”) adjustable pulley on the motor (Figure 3.4). With some fine tuning of the adjustable pulley, these changes allowed the default 25 rpm to be increased to the required 72 rpm for effective grinding. Furthermore, the adjustable pulley allowed for any future rotational speed adjustments, should the need arise.

During field testing, placer operators would indicate which materials in their gold room had been abandoned due to difficulty in processing, but stockpiled due to perceived high contained values. Upgrading was attempted using a small portable shaker table, increasing the effective throughput of the grinding mill (Figure 3.5).

Figure 3.4: Replacement pulleys for the cement mixer mount and engine (Photo: Gavin Clarkson 2014).

To avoid losses a generous cut was taken, rejecting only the lightest portion to tailings and only partially upgrading the concentrate where possible. Coarse gravels would be removed through wet sieving to -8#. Oversize materials would be investigated for nuggets and discarded. The high density middling materials were usually dominated by a certain type of occluding high density mineral, commonly cassiterite, garnet, galena, hematite, ilmenite or magnetite. Heavy mineral content varied between operations mining different creeks, and the main difficult mineral was identified. Prior to grinding, the volume and weight of each sample were taken and the bulk SG calculated. The material would then be loaded into the mill in small batches, typically less than 1.5 kg or ½ L by volume.

Figure 3.5: (left) Keene table on portable frame used to upgrade concentrates where possible. (Right) "Rough cut" example where little material is fully rejected to tailings to minimize losses in upgrading difficult concentrates (photo credit Gavin Clarkson 2014).

Water was added to an approximately 1:1 ratio by volume with the solids, after which the mill was sealed, levelled and activated. For each new material encountered, milling would proceed at approximately 1-minute intervals. After each interval, the grind products would be carefully poured over a 50# sieve and investigated. This would continue until the contents remaining on the screen were approximately 90 wt% gold with limited waste, which would be considered a successful test (Figure 3.6). If screen products were greater than 90% gold, the test was marked as overground. In some cases, the -50# fines would be reground in the rod mill for an extra 1-2 minutes and the grind products poured over a 70# screen, again considered successful if approximately 90 wt% gold remained on the screen. The oversize screen material would be quickly panned to remove any remnant waste particles and the gold dried and weighed.

For tests designed to determine overgrinding of contained gold, the grind feed would be classified to +50#. This ensured that any gold in the sample naturally finer than the collecting screen would be removed and therefore any gold in the product fines was reduced as a direct result of grinding. These were considered losses. Some samples were not classified to +50# and ground as a whole with fines included, done to better estimate the overall fine gold content and appropriateness of the collection screen size.

To quantify the loss, gold was separated from the fines by panning when possible. If unable to fully clean gold through panning, mercury would be introduced for amalgamation, and occasionally tailings were sent in for fire assay. Some cases allowed whole sample mercury amalgamation, while others required splitting the fines to a more manageable representative sample. Whole amalgamation was preferred as it minimized any errors introduced by the coarse gold nugget effect, although fire assay results were consistent with amalgamations. For sample splits, the gold grade of the split could be determined and back calculated to the weight of the whole sample, approximating overall gold losses to the fines. Amalgamation was done manually with vigorous shaking of the pan and hand mixing to ensure complete gold adsorption, using minimal beads of mercury. Mercury parting was performed with concentrated nitric acid, removing any contaminants that may have been included with the amalgamation. Panning of amalgamation tailings confirmed all contained gold had been adsorbed.

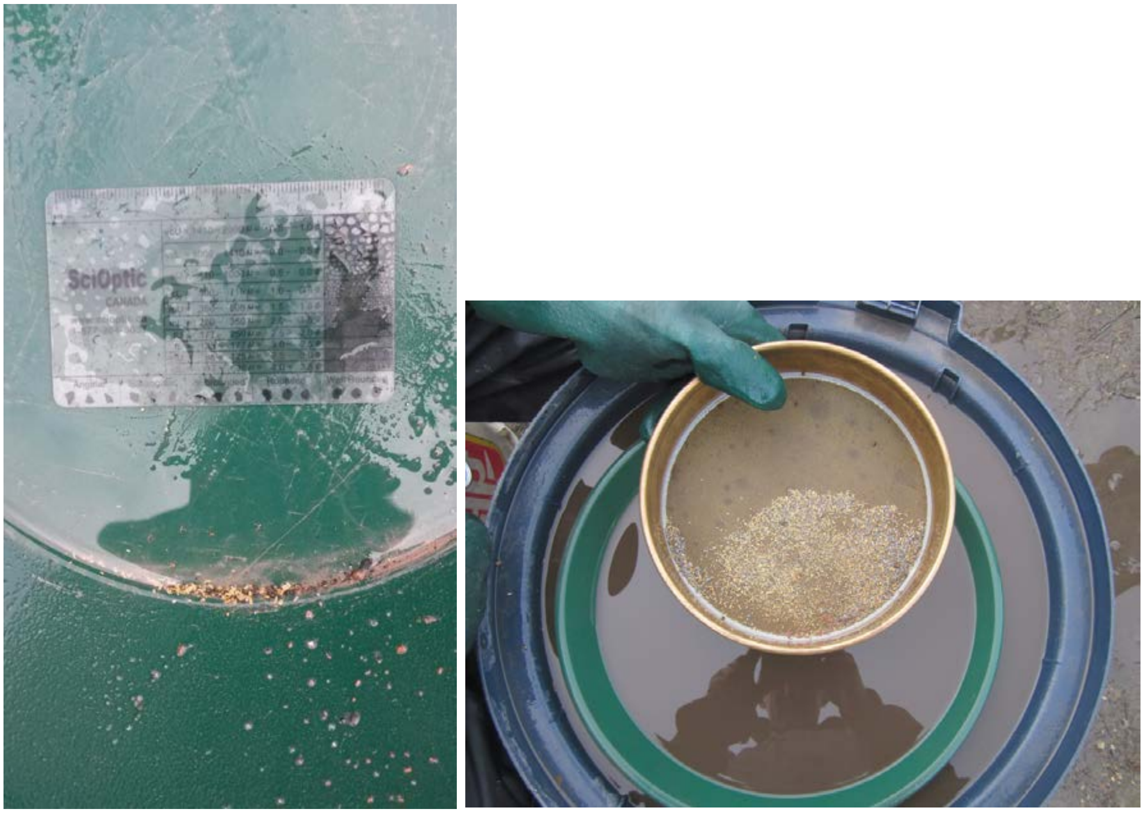


Figure 3.6: Typical +50# screen products for a completed test (photo credit Gavin Clarkson 2015).

Where sample splits were required, reproducibility was ensured through thorough mixing and careful use of a laboratory splitter. Mixing was performed over 15 minutes of folding using a rubber sheet, illustrated in Figure 3.7. Splitting was performed on the entirety of the mixed sample and repeated until the necessary sample size was obtained. During lab tests, particle size distributions were performed using a stack of standard sieves and classified at 8#, 16#, 30#, 50#, and 70#, shaken dry for no more than 15 minutes to ensure complete size sorting. Beyond this time period, the fines proportion separated was insignificant and attributed to reduction through sieve abrasion.

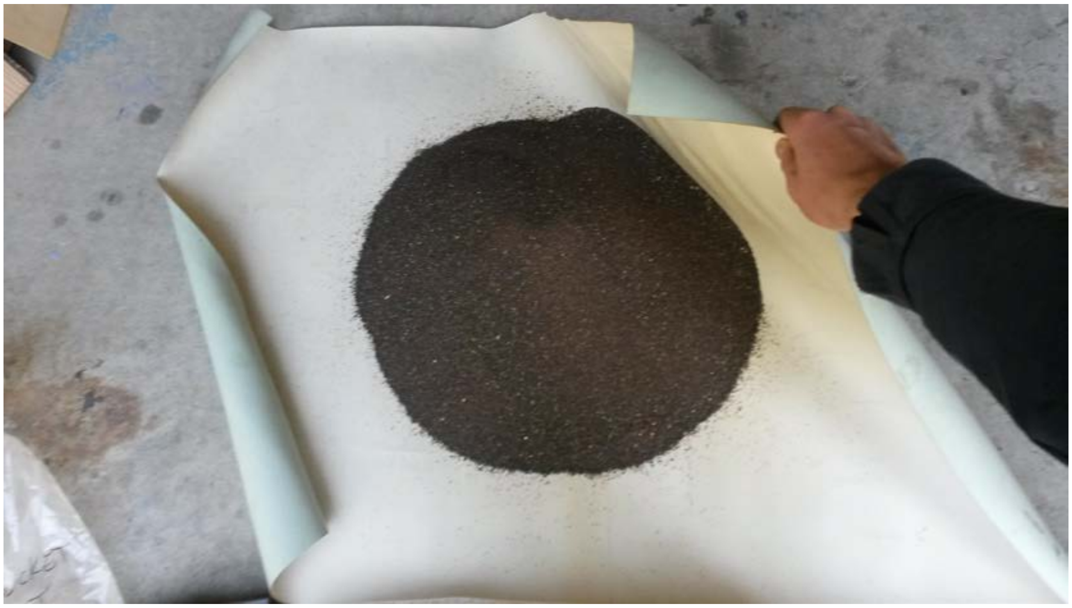


Figure 3.7: Mixing of bulk concentrate samples through repeated folding (photo credit Gavin Clarkson 2015).

Lab tests were often salted to ensure test concentrates contained some portion of recoverable gold. The salting was done with +50# flakes recovered by grinding during the field testing portion of the study (Figure 3.8). A small laboratory splitter was used to evenly split the flakes into samples of approximately 120 mg each. Given the small sample sizes being used during lab testing (<1 kg), this provided a sufficiently high grade for reliable testing even when discounting any possible natural gold occurring in the donated concentrates.

Figure 3.8: +50 gold recovered using rod mill, used for salting in lab experiments (photo credit Gavin Clarkson 2015).

Chapter 4 - Results

The following section summarizes the results of the three different testing phases: Pre field lab testing, field tour, and post field lab testing. Also included are results from a 2,000 oz t trial run reported by a miner that adapted the grinding mill as his main concentrator device. Before grind tests were done during field testing, a series of primary gravity concentrates from various Yukon locations were obtained and geologically analyzed to determine common heavy waste materials, the results of which are included with the field testing section.

Pre-field tests performed at the University of Alaska, Fairbanks, proved the ability to extract gold using grinding. The field testing phase yielded better recoveries than those done in the pre-field 20x20cm rod mill, and allowed this method to be attempted in a variety of settings and operations. Variability between testing properties during the field tour made correlating recovery sensitivities difficult, which motivated the post-field lab tests, designed to examine recovery sensitivities. During the post-field lab testing phase, batch samples from two properties were collected and split to identical representative tests that could be performed under varied conditions. Although during field tests the dominant occluding heavy mineral varied between properties, the majority of tests performed in lab settings were donated by two different operators, one dominant in cassiterite and the other rich in garnet.

For all results, locations and operators are confidential, and sample identification codes in no way indicate the source of the concentrates used. Prior to attempting gold recovery through grinding, a simple analysis of various primary gravity placer concentrates were analyzed to determine common possible occluding sulphides that occur in the Yukon gold fields and surrounding areas.

Before commencing the field testing tour, representative processed concentrate samples from multiple properties were obtained and analyzed. The purpose was to investigate common high density minerals that occur in primary concentrates that may accumulate in downstream processing. All samples had been collected in the field, run through a long tom, and hand panned. The pan concentrates were then dried and split to representative samples each approximately 40 g. These representative splits were visually examined with the aid of a microscope and the minerals present were identified and assigned a visually estimated abundance percentage (Appendix A). Mineral identification was aided with the use of an X-ray Fluorescence machine. Exact collection locations are kept confidential but all originated from the Klondike placer fields, the Mayo/Duncan creek mining region, and the Atlin mining region. Though proportions varied considerably between creeks, common high density minerals included pyrite, magnetite, garnet, cassiterite, ilmenite, hematite, and minor heavy phosphate minerals like apatite. One heavily mined creek in the Klondike placer fields contained high amounts of garnet, to the point where primary concentrates were greater than 50% garnet grains. Apart from garnet, the high density gangue minerals are dominantly soft and brittle sulphides rather than strongly bonded silicates (Table 4.1). Lower density waste minerals such as calcite, quartz, and rock fragments are easily discarded in the downstream gravity processing steps. Brittle waste materials bode well for the grinding extraction method as the high grindability of the waste would allow for faster grind times and ideally less overgrinding of gold particles into the undersize.

Table 4.1: Common high density waste minerals occurring in Yukon placers with quartz for comparison (compiled from (Nesse, 2000).

Mineral	Density	Hardness
Pyrite	4.9-5.2	6.0-6.5
Magnetite	4.9-5.2	5.5-6.5
Garnet	3.6-4.3	6.5-7.5
Cassiterite	6.8-7.1	6.0-7.0
Ilmenite	4.5-4.7	5.0-6.0
Hematite	4.9-5.3	5.0-6.0
Quartz	2.65	7

Overall, recoveries of pre-field laboratory tests were relatively low (Appendix B). In attempt to improve losses, changes in the charge size, mill type and rotational speed were attempted but yielded no solid correlations. Variability in size distributions, mineralogical characteristics, extent of previous processing, and uneven gold salting between tests are examples of why inter-sample correlation proved challenging. The variability of results occluded any obvious quantitative trends (Table 4.2). Losses to the undersize were as high as 70%, although tests were not always classified at +50# before grinding, allowing naturally fine gold to be included in the fines used for overgrinding loss calculations.

Table 4.2: Summary of pre-field lab testing.

Test Name	Grind Time (min)	Sample Size (kg)	Au Rec'd (g)	Au Loss to -50# (g)	Loss	Total Au (g)	Feed Grade (g/t)
CassiterL01	10	2.424	0.05	0.03	41.6%	0.08	32
CassiterL02	15	2.882	0.27	0.06	18.2%	0.33	113
CassiterL05	10	2.322	1.04	1.89	64.5%	2.93	1262
CassiterL06	20	2.222	0.73	1.41	65.8%	2.14	963
CassiterL07	20	1.000	0.74	0.30	29.0%	1.04	1044
CassiterL08	15	1.000	0.41	0.92	69.3%	1.33	1333
CassiterL09	10	1.000	0.28	0.32	53.1%	0.79	789
CassiterL10	10	1.000	0.22	0.06	21.7%	0.28	280
CassiterL11	10	1.000	0.29	0.02	5.8%	0.31	308
Garnet L03	10	1.944	0.51	0.16	23.8%	0.66	342
Garnet L04	20	1.976	0.25	0.25	50.5%	0.50	253
Garnet L12	15	1.000	0.32	0.09	22.1%	0.41	415
Garnet L13	40	1.000	0.36	0.06	14.2%	0.42	423

Despite high gold losses, these tests did manage to extract clean gold from otherwise unresponsive gravity concentrates in a short amount of time, especially when compared to the current processing method of tedious hand picking. This had the potential to unlock significant value, which was proven with high gold recoveries experienced during field testing of much richer concentrates.

Field testing phase results, using the 20x30 cm rod mill, were more positive (Appendix C). Upgrading attempts with the Keene table before grinding had varying success, ranging from as high as 96% reduction to none at all without unacceptable gold losses. Difficulty in upgrading these concentrates with a gravity based shaker table was expected, as these materials have already undergone extensive gravity processing with no improvement. It is important to note that even materials that were more responsive to the shaker table still failed to separate the gold out to a high enough grade for smelting. Even with 96% reduction, the concentrate left was too low grade, as miners needed greater than 70% gold by weight in their concentrates to justify smelting or sale. Further density-based upgrading was not possible without gold particles "surfing" over waste minerals into middlings and tailings. It was at this stage of processing the 20x30 cm rod mill was substituted.

Table 4.3: Volume reduction of difficult concentrates by tabling.

Test	Table feed (kg)	Table Con (kg)	Reduction (%)
Galena F1	4.478	1.932	57%
Cassiter F1	54.19	11.89	78%
Garnet F3	22.3	2.24	90%
SGMg/HeF2	42.34	1.764	96%
HBWC F1	15.36	1.55	90%
HMg/He F1	24.12	12.25	49%
Garnet F2	7.13	7.13	0%
Garnet F1	22.3	2.24	90%
Galena F3	4.478	1.932	57%

The gold recovered through grinding during the field tour was quite encouraging. For example, one set of concentrate which could only be reduced by 50% on the Keene table yielded over 111g (3.6 oz t) of clean gold after grinding just over 1 kg of feed. This is a value of $4,320 at $1,200/oz t from material otherwise stockpiled with no plans to upgrade, extracted and ready for sale in just 6 minutes of grinding. Gold losses to fines were also low, especially when compared to pre-field lab tests. Throughout the field tour, losses to the undersize ranged from less than 1% to 15%, excepting one outlier property requiring longer grind times and undersize losses of 60%. Table 4.4 summarizes the portion of grind tests classified to +50# prior to grinding, where exact overgrinding losses could be calculated.

Table 4.4: Field test results. Each test below was screened at 50# prior to grinding, ensuring that gold particles occurring in the fines portions were overground and reduced by milling. Italics indicate recoveries improved by secondary screening over 70#.

Test Name	Grind Time (min)	Dom High Density Waste	Au% +50# Recovery	Au% +70# Recovery	Overall Recovery
Galena F1	6	Galena	99%	N/A	99%
Cassiter F1	*7*	*Cassiterite*	*60%*	*29%*	*90%*
Cassiter F3	*7*	*Cassiterite*	*86%*	*8%*	*94%*
Garnet F3	5	Garnet	98%	N/A	98%
HBWC F1	6	Lead	83%	N/A	83%
HMg/He F1	9	Hem/Mag	88%	N/A	88%
SGMg/HeF2	12	Hem/Mag	38%	N/A	38%

Note that in the majority of field locations, typically less than 10% of the gold in the +50# portion was overground into the undersize, while the majority was preserved and flattened into recoverable grains which settled on the collection screen. In tests Cassiter F1 and Cassiter F3, recoveries were lower than typical so concentrate was reground for approximately 2 minutes and collected on a 70# screen. In one case, this single added step improved recovery from 60% to 90%. This indicates that gold can potentially be ground and recovered down to nearly any available sieve size, though further testing indicated there is little to be gained in pursuing gold particles below 70#. It is of further interest to note that during regrinding of the fines, some gold particles formerly -50# become recoverable on a 50# screen, actually increasing in size during grinding. This reinforces the results found by Noaparast, 2004, in which gold size reduction is often by folding rather than failure, after which further grinding can cause an increase in classification size from being flattened again (Chapter 2).

With concentrates that were not classified to +50 prior to grinding, recoveries remained high, indicating only a small amount of gold occurred as natural -50# fines. Table 4.5 illustrates the results of field grind tests performed without classifying the feed at 50# prior to grinding. If a significant proportion of the existing gold was less than the 50# collection size, the overall recovery would be expected to suffer as the majority of gold would occur in the undersize. Despite this, the majority of field tests still boasted recoveries in excess of 90%, excepting one case of overall sieve recovery being only 41%. This indicates a minimal amount of fine gold (<70#) naturally occurring in the gold room concentrates at these locations, and efforts expended in collecting gold on finer sieves to be inefficient. For a more qualitative measure of natural -50# gold particles, the unground fines removed prior to field tests from Table 4.4 were split and assayed via mercury. The results are indicated in Table 4.6. As can be expected, the distribution for each property varies considerably as each location mines different creeks and uses different primary recovery and concentrating systems.

Table 4.5: Field grind tests which were not classified at 50#.

Test Name	Grind Time(min)	Dom High Density Waste	Au% +50# Recovery	Au% +70# Recovery	Overall Recovery %
Garnet F2	7	Garnet	96%		96%
Cassiter F2	7	Cassit	77%	+16%	93%
Galena F3	6.67	Galena	95%		95%
Garnet F1	9	Garnet	41%		41%

The percentage of natural fines varied from under 2% to almost 80%, so it must be emphasized that grinding by recovery should not be viewed as a replacement to gravity upgrading unless there is great confidence that the collection sieve is properly sized to the property's gold content.

Table 4.6: Percentage of gold present in unground -50# fines.
N/A where gold fines proportions not measured during testing.

Test Name	Au Rec'd (g)	Au Loss to -50# (g)	Natural fines (g)	Total Au (g)	%Natural -50#
Galena F1	111.27	0.80	7.82	119.94	7%
Cassiter F1	4.37	0.50	N/A	4.87	N/A
Cassiter F3	9.81	0.35	4.57	14.62	31%
Garnet F3	78.95	2.20	56.25	136.49	41%
HBWC F1	92.14	22.67	18.72	130.27	14%
HMg/He F1	40.28	26.06	180.20	225.73	80%
SGMg/HeF2	15.84	27.36	0.70	43.83	2%

Using the total gold contents of the materials tested during the field tour, the grades of the feed material were calculated. These high grades emphasize the need for an automated way to process these stockpiled middlings, as they contain significant gold, especially considering they currently have very low processing priority. Note the concentrate grades in Table 4.7. Due to the difficulty of processing, material upwards of 180,000 g/t gold is stockpiled by miners, some of which have been storing these concentrates for more than a generation with little being processed in the off season. During the field tour, some of the miners were impressed enough by our testing to adapt their own rod mills to their processing methods. One miner used the mill as the primary concentrator, rather than as a substitute to upgrade middlings from other density based equipment. The results of his trial run using a rod mill similar to the field test mill are outlined below.

Table 4.7: Summary of field test results including calculated grade of feed material.

Test	Grind Time (min)	Grind Feed (kg)	Au Rec'd (g)	Au Loss to -50# (g)	Feed Grade (g/t)
Galena F1	6	1.04	111.27	0.80	108,249
Cassiter F1	7	1.13	4.37	0.50	4,302
Cassiter F3	7	0.58	9.81	0.35	17,584
Garnet F3	5	0.44	78.95	2.20	183,130
HBWC F1	6	0.74	92.14	22.67	154,724
HMg/He F1	9	1.19	40.28	5.26	38,110
SGMg/HeF2	12	1.37	15.84	27.36	31,598
Garnet F2	7	1.25	48.97	N/A	39,335
Cassiter F2	7	0.93	29.19	N/A	31,514
Galena F3	6.67	1.10	38.5	N/A	34,895
Garnet F1	9	1.12	44.1	N/A	39,295

At this property, the only secondary concentration used before direct milling is a long tom. The long tom middlings are then classified into -3# /+12#, -12#/+20#, -20#/+30#, and -30#/+50#. These discrete size ranges are milled in lots less than 2.25 kg for approximately 7 minutes, and ground slightly longer if the concentrate settling on the collection screen is not clean enough. The grind slurry is then run through a 20# and 50# screen, the material on the sieves greater than

85 wt% gold. The oversize is quickly tabled to remove any excess waste and demagnetized to nearly pure gold. This is the only current example of an operation using the mill as its main concentrate upgrading device, with the shaker table only being used to clean the oversized grind concentrates. Occasionally, the fines are reground and run over a 70# screen in an attempt to recover any fine or overly abraded gold particles. These attempts have had mixed success, reportedly extracting a maximum of ¾ of the contained gold from the fines.

The operator saved the -50# unprocessed grind slimes over the duration of a 2,000 oz t gold extraction trial. Once 2,000 oz t had been recovered using grinding and sieving, the fines were delivered to a local concentrate processor known for his expertise with a large Deister-style shaker table. After 8.25 hours of careful, well attended processing, a mere 26 further ounces were extracted. This is a loss of less than 1.3% of gold to the undersize, including any naturally occurring fine gold particles. The trial run at this operation has proven the potential grinding has in gold processing, being used as the main upgrading device rather than complimentary to a classic upgrader such as a wheel, jig or table. It is important to emphasize however that although this type of application proved fruitful for this particular property, not all locations will boast coarse enough gold or a high enough proportion of dense minerals to justify replacing gravity with malleability-based extraction.

Field and pre-field tests indicated one of the greatest factors affecting recovery was the mill charge load, a hypothesis further supported by correlating test results of similar material from the two first testing phases (Figure 4.1). Although a negative correlation between mill load size and gold recovery is indicated, the data must be interpreted carefully as it is compiled from the same property at different times, with possible variations in size distribution and extent of gravity processing. Furthermore, the pre-field lab tests used different grinding equipment as outlined in Chapter 3. Despite this, the data does indicate overall recovery seems to be sensitive to the amount of material being ground in the mill, which supports experience during the field tour. To fully understand the factors effecting gold recovery of the grinding extraction method, careful repeatable lab testing needed to be done. Below are the results of the post-field tests, designed for repeatability and recovery correlation.

Field testing was done on location, allowing for a diverse set of samples to be experimented on. Due to time and operator constraints, only a limited amount of tests were able to be performed at each property. Since geology, mineralogy, stream maturity, and processing equipment are just some of the factors that can change significantly between neighbouring properties, correlating test results to determine recovery sensitivities were restricted. Post-field lab tests sought to perform repeated tests with identical materials under varying conditions (Appendix D).

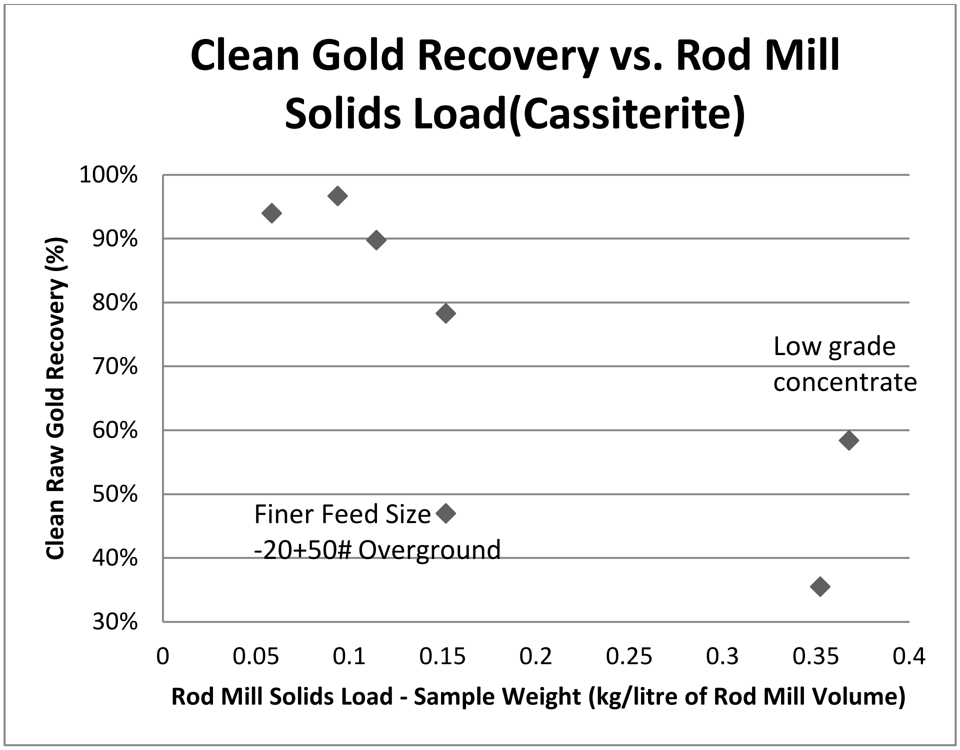


Figure 4.1: Decrease in gold recovery vs. rod mill load, compiled from field tests and pre-field lab test data from testing at the University of Alaska Fairbanks.

Two separate operators donated bulk samples for this testing, one dominantly garnet and the other cassiterite. Primarily, sensitivity to charge load was to be determined. The reaction to differing water content was also to be investigated, as most prior tests neglected to measure water addition, simply approximating a 1:1 ratio of water and solids by volume. Under optimal controlled conditions, the post field lab tests were meant to determine recovery trends in terms of load size and liquid content using an identical evenly sized charge load. Post-field lab testing sought to minimize errors introduced in previous tests by sample variability, uneven gold salting during the pre-field lab tests, and using different grind equipment between tests.

To avoid geologic variability, the bulk samples were separately evenly mixed and split, confirming sample reproducibility with matching particle size distribution (PSD) tests. Salting was done evenly and carefully measured to ensure an even minimum of gold in the samples to recover. Grinding was performed in the same apparatus as the field tests, where higher recoveries were achieved. Grind times were carefully controlled and performed iteratively until visual inspection confirmed the sieve collection contained approximately 90% gold with some waste. Tests with only gold on the screen were considered over-ground and rejected as outliers.

Results of the garnet concentrate will be summarized first. This was a relatively well-mixed fine to lower coarse sand between 50# and 16#, with very little fines or oversize material (Figure 4.2). This is appropriately sized for rod mill grinding, since larger particles will be preferentially reduced resulting in an even grind to -50# without needing to overgrind and chip the contained gold particles. After PSD analysis, any -50# remaining was discarded.

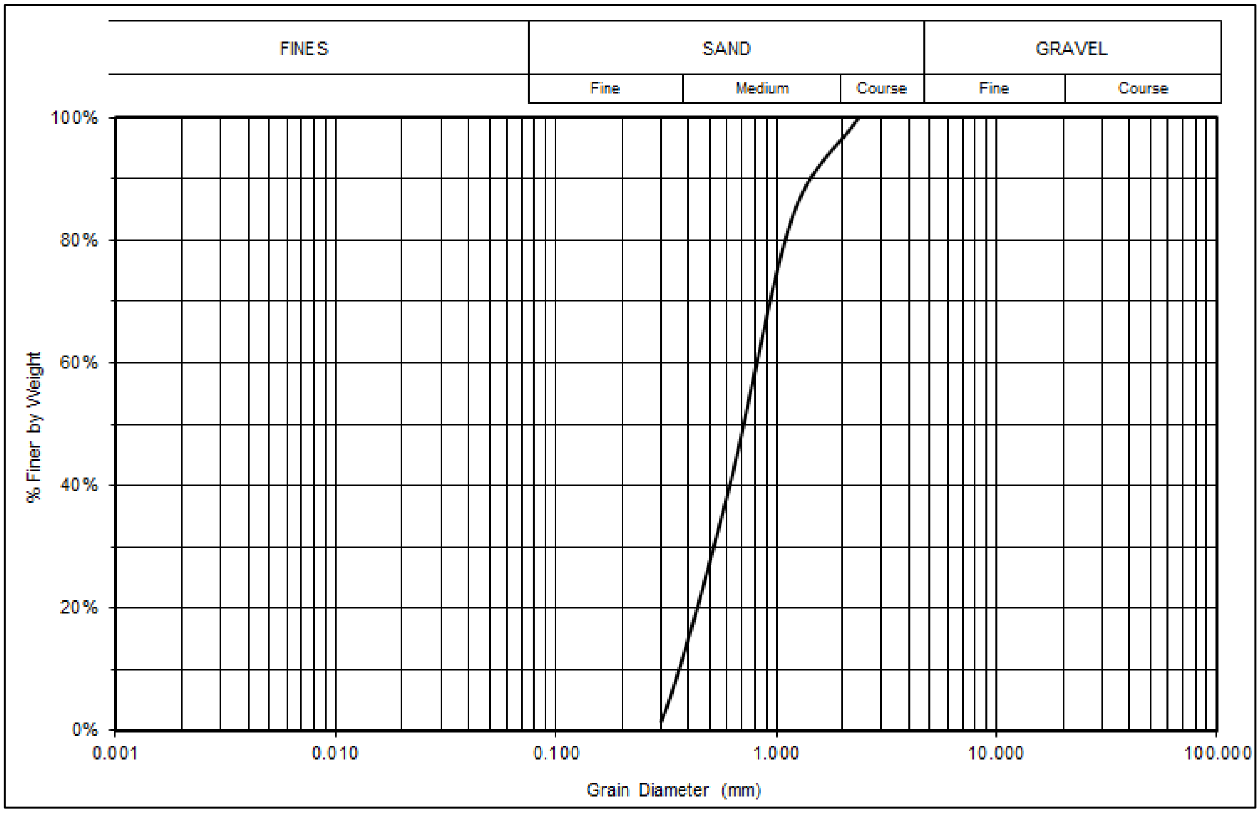


Figure 4.2: PSD of Garnet test material.

Loss estimates were carried out between each grind. It quickly became evident that the high portion of the hard orthosilicate was detrimental to grind recovery, and the material tested in the lab was different from the garnet concentrate field-tested at the same property where recoveries exceeded 90%. It is likely this batch went through further gravity processing, and contained a higher concentration of hard garnet than those tested in the field. Even at half the typical field charge (456 g), grind time exceeded 10 minutes and recovered only 73% of the contained gold, compared to field results where garnet concentrate grind time was as low as 5 minutes and +50# recoveries greater than

95% (Table 4.4). Increasing the mass to match field tests (1 kg) resulted in greater losses, losing nearly half to overgrinding. In an attempt to improve recoveries, water content was increased but showed no change, actually increasing the grind time by 90 seconds and causing greater losses. Although gold extraction from this material was possible, the hard waste minerals proved to be too resistant to grinding to provide an idealized sample for loss correlation and focus was switched to the cassiterite material.

The cassiterite concentrate yielded results akin to those measured during field tests, as the bulk sample used had likely undergone the same degree of processing. Furthermore, sulphide minerals like cassiterite have a weaker ionic character than silicates like garnet, the weaker bonds contributing to a greater crushability (Nesse, 2000). Adjusting the grind conditions for each identical cassiterite sample yielded some revealing trends. Like in field tests, losses to overgrinding typically were less than 10% for mill charges less than 0.1 to 0.15 kg/L. As predicted, losses began to suffer for larger mill loads. In an attempt to reduce gold over-grinding in larger mill charges, water content was increased to reduce slurry density. However, reducing slurry density also tended to increase the grind time necessary for the reduction required and consequently increase losses.

Data from lab testing is summarized below. Although the garnet tests were not considered representative and not tested as thoroughly, the results are included for completeness and often indicate trends similar to the more thorough cassiterite testing.

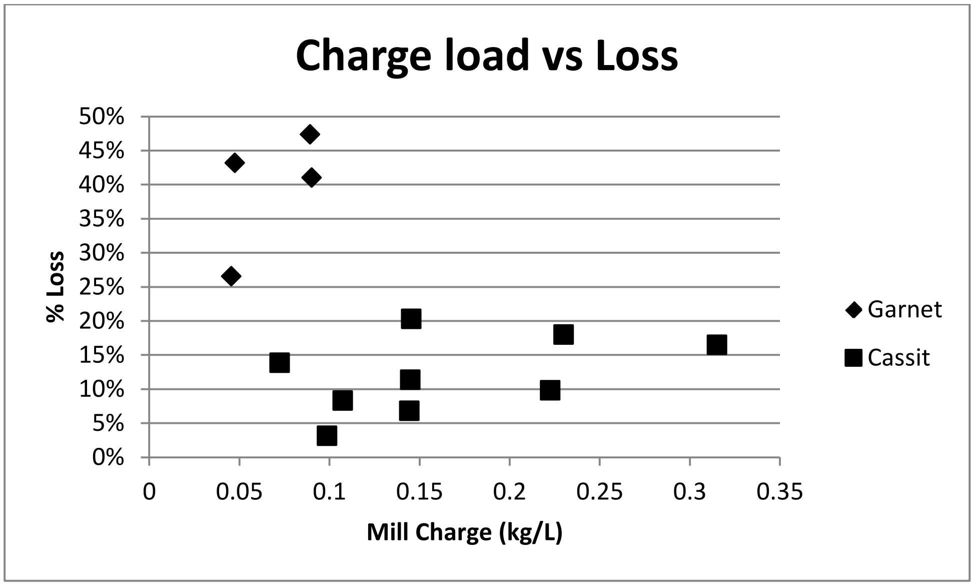


Figure 4.3: Mill charge load vs. losses.

Figure 4.3 indicates what was predicted during field testing: Overall gold recoveries are sensitive to the size of the load in mill. The recovery sensitivity of charge load and grind time are very similar, as can be seen in Figure 4.4. As can be expected, mill charge and grind time are closely correlated (Figure 4.5).

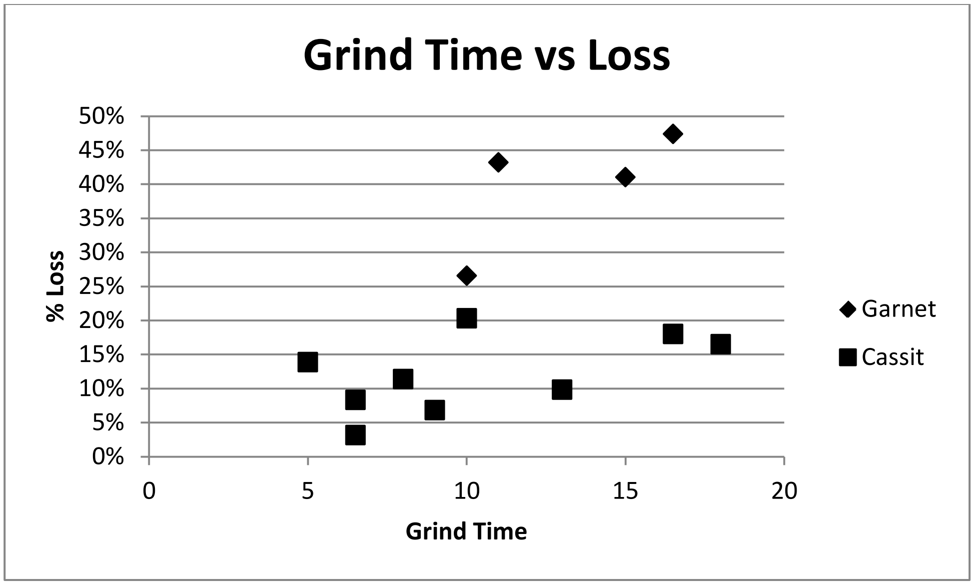


Figure 4.4: Grind time vs. Loss.

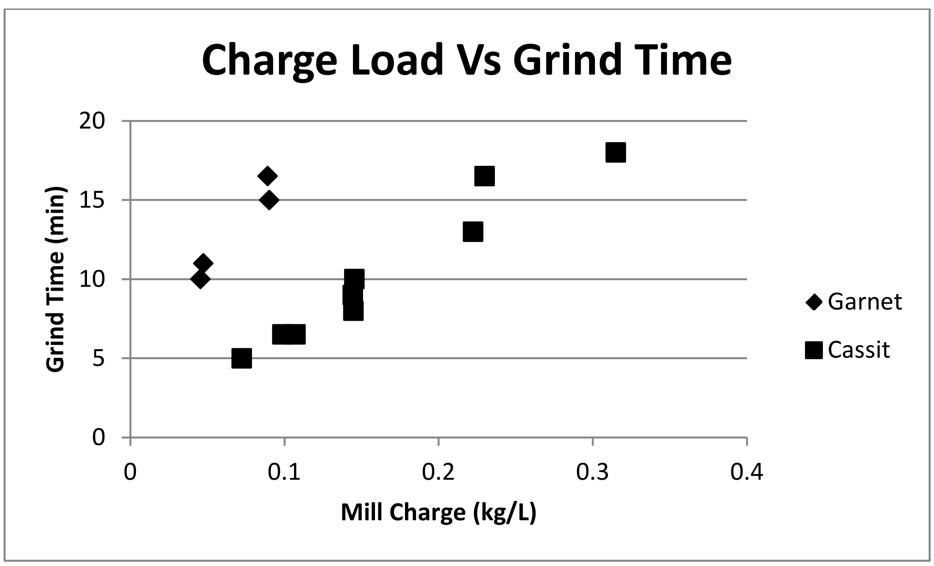


Figure 4.5: Charge load vs. grind time.

Water content was increased in larger loads in an attempt to reduce the losses observed, though water content appeared to be inconsequential to recovery

(Figure 4.6). Overall grind time and charge size appear to be the inter-related factors that most influence gold loss to the fines.

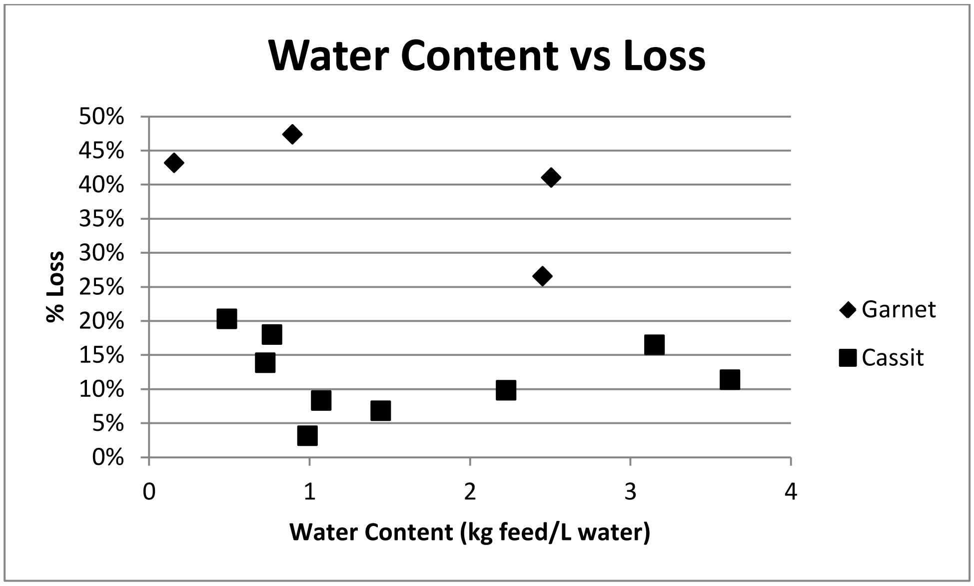


Figure 4.6: Water content vs. gold loss.

Chapter 5 - Discussion

Grinding gravity concentrates for gold extraction has shown to have multiple advantages, as well as being simple, effective and reliable. Field testing of various ores managed to extract gold previously inaccessible by gravity concentration methods. The discussion section below emphasizes the potential value that can now be extracted using this method, as well as the observed benefits of improved gravity recovery of grind products. A simple flowsheet is presented of a placer recovery setup which integrates the rod mill, with grades, recoveries and reductions estimated from experience and raw field data. Recovery sensitivity trends produced during post-field lab testing are analyzed, and the main recovery sensitivities determined. Since maximizing recovery must also be balanced with production efficiency, trend line equations from test results are used to simulate the interaction of gold production, charge size and acceptable losses for a cassiterite concentrate.

Figure 5.1 illustrates a simple flowsheet of a placer gold processing circuit which includes the rod mill as a processing stage. In this setup, the primary sluice concentrate is upgraded with a long tom, the clean gold from which is extracted and the middlings processed over a table. The clean concentrate produced by the table is sent for smelting, while the table middlings are submitted to grinding and sieving to extract the difficult to upgrade gold particles.

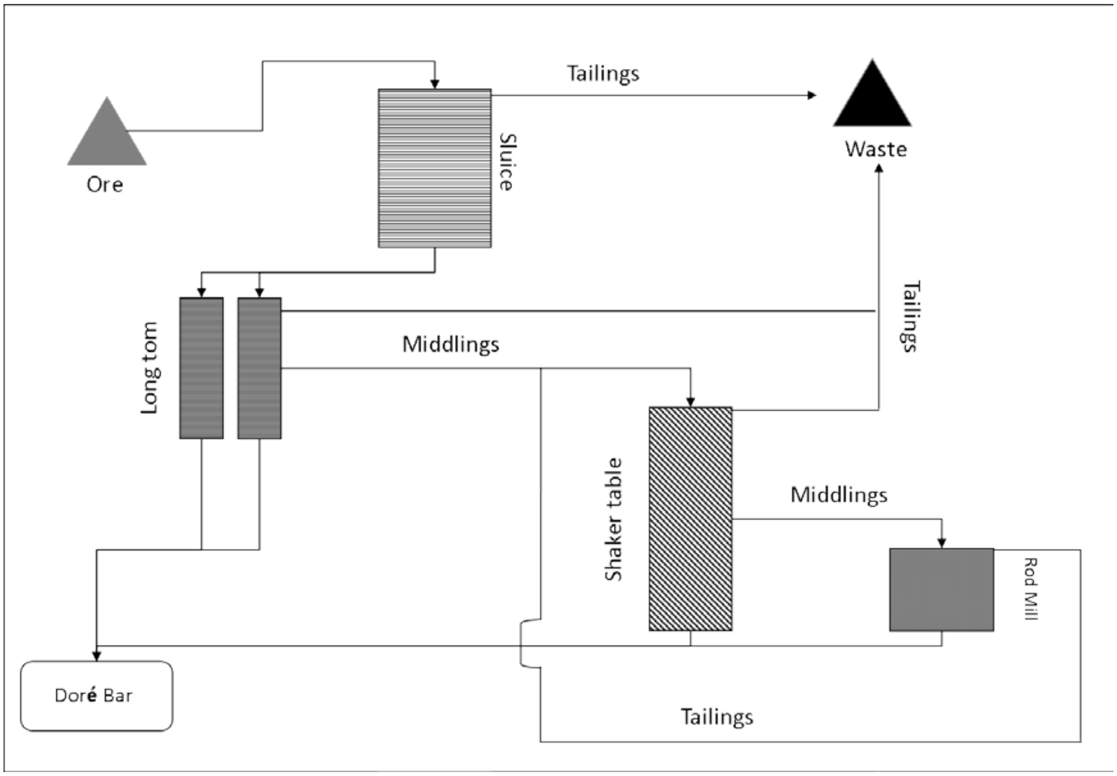


Figure 5.1: Flowsheet of placer gold processing circuit including rod mill.

Note the tailings of the rod mill are recirculated and processed on the table to maximize efficiency. Detailed lab results for reductions and recoveries of the mill and table are available from the field tour, but earlier stages such as the sluice and long tom needed to be estimated and back calculated based on results observed in the literature. Calculations and assumptions are included in Table 5.1 below. Although the mill itself extracted greater than 95% of contained gold from the difficult middlings, it is important to understand the overall losses of the entire circuit.

As mentioned in Chapter 2, it is more important than ever to maximize gold extraction while minimizing processing time in the modern Klondike gold fields.

This would improve profit margins for operators, allowing increased resources to be dedicated to exploration and potential future discoveries. The amount of gold being lost to the middlings and unable to be further upgraded is surprising.

Table 5.1: Metal balance calculations used for flowsheet in Figure 5.1.

	Recovery to Doré	Feed Amount	Tails Amount	Feed Gold	Feed Grade
	%	kg	kg	g	g/t
Sluice	95%	14400000	14399712	2400	0.17
Long tom	70%	288	172.86	2280	7916.67
Table	90%	115.13	92.11	570	4950.66
Mill	95%	23.01	23.012	51.30	2228.79

	Gold to Tails	est. Loss to Tails	est. Loss to Mids	Approx. Reduction	Gold to Doré	Waste to Doré
	g	%		%	g	kg
Sluice	120	5%	N/A	0.002%		
Long tom	114	5%	25%	40%	1596	0.16
Table	5.7	1%	9%	40%	513	0.05
Mill	2.57	5%	N/A	20%	48.735	0.0048

While currently impossible to tell exact numbers, some estimates of potential value can be inferred with the test results done during the field season. One example in particular is sample Cassiter F3, with a mill feed grade of 17,584 g/t. This particular concentrate was not able to be upgraded at all with a table, meaning this property has been stockpiling unprocessed concentrates that is more than 1.75% gold. Over decades of stockpiling, untold amounts of this material have been building up. If even a single 159 L barrel has been saved, that's approximately 423 kg of concentrate containing 7.4 kg of gold, or 238 oz t for a potential value of $285,600 at $1,200/oz t. This is significant value to ignore, especially in the realm of the small scale independent miner that dominants the

Klondike placer gold fields. Using the grind for extraction method maximizes the gold production of a mine without the costly need for stockpiling and storage of rich concentrates. This also reduces the security risk introduced by stockpiling, , requiring the storage of valuable materials year-round despite mines only operating approximately 4 months of the year.

Despite less than optimal pre-field test results, with grind times often exceeding 15 minutes and recoveries rarely exceeding 50%, the subsequent field tour and post field testing phases with the new rod mill equipment were a success. In all, during field testing 7 different properties were visited, each with unique upgrading systems and mineral assemblages. Each of these properties identified concentrates that were being stockpiled with no imminent plans on how to extract the contained gold values. Using grinding and sieving, clean sellable gold was able to be removed from the concentrates in less than 10 minutes of grinding per approximately 1 kg sample (Table 4.7). The heavy occluding minerals were able to be effectively reduced to less than the collection sieve size (50#) while gold particles were preserved and rarely folded or chipped to a smaller size classification, with on-screen recoveries exceeding 90% for the majority of field testing. One limitation of this technique is the recovery of very fine gold, sized less than the collection sieve. Recoveries for coarse particles were high, but one project in particular had almost 80% of the contained gold in the middling concentrate sized lower than the collection screen, with other properties ranging from 15% to 40% naturally occurring fine gold (Table 4.6). Grind extraction still has a place at these types of properties, as it quickly extracts the coarse gold content, and also improves subsequent gravity recovery of the newly ground fines.

Depending on the material being processed, grinding and sieve collection alone can extract greater than 80% of the contained gold in difficult locked concentrates. The grind fines can then be tabled, and table separation from the -50# grind fines was significantly easier than attempting to remove gold from the unground -50# fines. During field tests, visual comparison during table processing indicated less particles "surfing" over the high density waste bed into the middlings or tails when compared to unground classified -50# fines. This implies that size classification alone is not the only factor influencing the improved table recovery, and that exposure to a comminution environment increases gravity recovery. This is likely a result of the even sizing of waste afforded by the grinding environment. In unground fines, gold particles have trouble penetrating the interlocking thick bed of dense uneven shaped minerals and were more likely to wander to the middlings or tailings. The freshly ground fines are not only well classified but also blocky and evenly shaped due to exposure to the same grinding environment. Evenly sized blocky fines allow greater bed penetration than interlocking uneven grains, as well as being more amenable to rolling off the shaker table into the waste. Furthermore, gold particles in ground fines have been flattened into flakes which perform better during table separation than rounded waste particles. Ground fines are a more ideal feed for further gravity concentration compared to the unground portion, allowing table recoveries to approach the 90%+ recoveries they are capable of (Table 2.1). Unground fines, even when well classified, tended to still reject fine gold to tailings. Even in properties with the majority of gold -50#, grinding concentrates can still benefit overall recoveries.

Post lab testing confirmed trends noticed during the field tour. As expected, the greatest factor in sieve recovery of the grind products was the charge load in the mill. As can be seen in Figure 4.3, there is a trend in increasing loss percentage vs. the mill charge amount. This is especially emphasized for the larger number of cassiterite tests but is also inferred for the difficult garnet samples. A very similar trend of increasing loss is seen when compared with grind time (Figure 4.4). It is intuitive to expect that the larger the charge load, the longer grind time needed to grind the majority of waste particles into less than the collection sieve size. This relationship between charge load and total grind time is emphasized by the results in Figure 4.5. It is clear these trends are inter-related, as larger loads necessitate longer grind times and therefore greater loss to fines. It appears that the longer grind times required to fully reduce the waste exposes the gold particles to greater abrasion and fracturing. Small loads are better suited to even grinding, quickly reducing the waste to fines before excessive gold particle abrasion with the grinding media occurs. In the rod mill, coarse particles are preferentially ground as their larger diameter prevents rod contact with the fines and gold particles (Figure 2.16). A larger charge results in extra grind time with only a small portion of coarse particles to be removed, allowing greater rod contact with gold particles and overgrinding of the fines. To maximize recovery, the grind time to reduce waste to below the collection size must be minimized.

Further experiments carefully varied the water content in an attempt to improve the recovery of larger loads. However, water content had no visible relation to loss percentage, as can be seen in Figure 4.5. In some cases, increasing the water content actually increased grind times up to 30 seconds longer than the 1:1 base water content (Appendix D). Based on this observation, a 1:1 solids/water ratio by volume, as performed for the majority of field tour tests with greater than 90% recoveries, is most appropriate.

Although charge size has the most effect on the recoveries of grinding for extraction, the grind fines are now more amenable to tabling to remove any lost gold particles. From an efficiency standpoint, it is important to analyze the benefits of sacrificing recovery in the rod mill stage with larger loads processing a greater amount of material.

There is a cost/benefit relationship in using greater load sizes within the mill, as despite higher gold losses a greater amount of gold can be recovered in a single run. This is especially important due to the non-continuous nature of the mill designed for this purpose, requiring time to stop, unload, and reload the mill. Gold ground to the undersize is now more responsive to separation by tabling, as the flattened gold particles are now easier to separate from fine, well classified, rounded particles that result from a mill grind. However, there is lost time in tabling these extra fines and it is of greater value to extract clean gold via sieving in the first step. To examine the cost/benefit relationship of using larger charges to extract more gold per test, a simulated scenario of processing a cassiterite concentrate similar to the one used in post-field testing was performed (Appendix E). This was done using recovery and grind time equations determined by trend lines fitted to the cassiterite recovery data.

It is clear that overgrinding losses are the most sensitive to the grind time. Grind time and mill charge size are well correlated as can be seen in Figure 4.5. Fitting a trend lines to this data yields the relation of grind time and charge size presented in Equation (5.1):

$$t = \frac{57.93x}{M_L} + .75 \qquad (5.1)$$

Where t = grind time in minutes, x = charge size in kg, and M_L is mill volume in litres. For loss estimation, the trend from grind time vs. loss tests (Figure 4.4) was used, as these results had a better correlation than charge load vs. loss (Table 5.1)

Table 5.2: R^2 values of trendlines from post-field test results used in extrapolation scenario.

Test	R^2
Charge Load vs. Grind Time	0.94
Grind Time vs. Loss	0.28
Charge Load vs. Loss	0.19

The trend line relating grind time and loss is presented in Equation (5.2), where $y =$ gold loss to undersize and $t =$ grind time.

$$y = .0065t + .054 \qquad (5.2)$$

Using these equations, a simulation was run using a cassiterite concentrate with 180,000 g/t contained gold, similar to grades obtained during field testing. For this analysis, mill charges started at 0.5 kg and moved incrementally up by 0.3 kg up to a maximum of 5 kg. For each load, grind times and gold losses were calculated using equations (5.1) and (5.2). By relating equations (5.1) and (5.2) and accounting for mill reload times and grade feed concentrate, clean gold production through grinding and sieve collection in g/min could be calculated using equation (5.3):

$$g/min = \frac{\left(-\frac{0.377x}{M_L} + 0.941\right) G_r x}{57.93x + r + 0.75} \qquad (5.3)$$

Where $G_r =$ grade of feed concentrate and $r =$ reload time between grinds.

As can be seen in Figure 5.1, the equation for g/min of gold produced is exponential, while losses are linearly dependent on charge size and grind time. From this relationship, maximum production can be achieved at 0.17 kg/L of mill capacity before no further efficiency can be gained to counter the increasing gold losses.

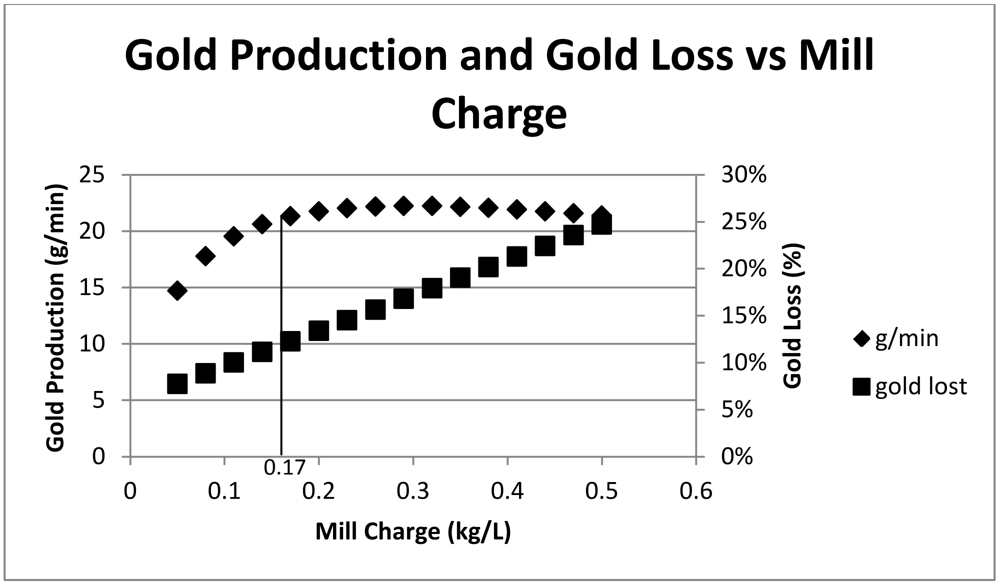


Figure 5.2: Gold production increase and loss comparison for different mill charges.

The difference in gold production efficiency from 0.05 to 0.17 kg/L is over 6.5 g/min, with an increase in loss of just 4.5% to a total of 12.2%. However, from 0.17 to 0.5 kg/L production only increases by .06 g/min, while losses increase by 12.4% to about ¼ of the contained gold being lost to the undersize and requiring further processing. For the 10L mill used during lab testing, this equates to using less than 1.7 kg of charge material to ensure high recoveries. This matches what was observed during field testing.

The example above and resulting equations are only representative for this single simulation of the same cassiterite concentrate that was used in the lab testing. Despite this, it does demonstrate how greater gold production efficiency is achieved with multiple smaller batches, rather than sacrificing more gold to overgrinding in an attempt to process greater amounts of material. Based on observations during field and lab testing, this tradeoff in production efficiency and loss would hold true for most concentrate types, though the representative equations would differ depending on concentrate size classification, sorting, and geological composition.

Chapter 6 - Conclusions and Recommendations

This thesis sought to alleviate the issue of the accumulation of high value middlings in the Yukon placer fields. Given the rising costs and declining grades of Yukon placer gold mines in recent times, maximizing the value of each claim is essential, and the current method of casually hand picking gold from concentrates is not an ideal solution. The grinding extraction method was designed to be a chemical-free, easy to use alternative to improve gold recovery of concentrates in the gold room. This method was meant to be easy to adapt and maintain, inexpensive, and require no further permitting that would otherwise be introduced by methods such as flotation, cyanidation or mercury amalgamation. Field results and lab testing have shown that adding a lab sized grinding mill as part of the concentrate upgrading process can have significant benefits.

In all, 7 different placer properties were tested during field application of the grinding for extraction method. Despite initial lab tests resulting in high overgrinding losses to the undersize, the field batch mill regularly managed to effectively reduce the brittle waste portion while preserving greater than 90% of gold particles for screen collection, with few exceptions. The concentrate extracted was typically greater than 90 wt% gold ready for immediate sale. Further benefits include the increased response of grind fines to gravity based methods like tabling when compared to unground fines. The potential value in formerly unprocessed middlings is surprising; with some mill feed grades as high as 180,000 g/t gold at mines that have been saving this portion of their concentrates for decades. This value can now be extracted through an efficient automated method that far outpaces inaccurate hand picking.

Miners were impressed with the application of grinding for gold recovery, and at the time of this writing more than 6 properties are adapting this technology to their cleanups. One property in particular reported less than 2% overall losses to the undersize when using the grinding mill as the only upgrading device downstream of a long tom.

Lab testing supported the field experience in that recovery was the most sensitive to the charge load and total grind time in the mill. Greater charges required longer grind times, exposing contained gold particles to longer periods of abrasion and possible reduction. As this is not a continuous unit, it is tempting to achieve greater throughput by sacrificing a portion of recovery for more efficient gold production. This is true to a point, though increase in gold production decreases exponentially for increasing charge loads, while gold losses increase linearly. Careful analysis of each differing concentrate type should be performed before considering over loading the mill beyond 0.15 kg/L mill volume.

Further research to be done on this subject would be the larger scale application of this technique. This research focused solely on lab-sized batch mills at independent placer gold mines. It would be of value to explore the potential this could have in larger mill circuits, and the design of a continuous throughput mill and classification system for the extraction of gravity recoverable gold at larger operations. Furthermore, although this research focuses on placer operations, it has a large potential application to any lode gold mine including gravity recovery in its processing circuit. Continuous gravity upgrading devices like Falcon and Knelson concentrators are often upgraded using tables, resulting in the same issue of middlings. Applying the grinding for extraction method could avoid the need to recirculate this material back into the circuit.

Can the malleability of gold be exploited to extract it from otherwise inaccessible placer middlings through grinding? Based on the results of field and lab testing, it can. Given the high recoveries of this method, the benefits to downstream gravity recovery, ease of use, and no further permitting required, a grinding mill could benefit nearly any gold room. It is no surprise that this technique is now being actively applied in at least half a dozen placer operations in the Yukon.

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Appendices

Appendix A: Mineral Abundance Estimation of Primary Gravity Concentrates

All samples run through test sluice, panned, dried and split into approximately 40g samples each. Examined under microscope as well as with portable xray fluorescence device on the Soil setting for rough elemental analysis.

- GC-13-08:
    - 5% magnetite
    - 50% Garnet
    - 15% Ilmenite
    - 25% White-light tan opaque dense minerals (Apatite/phosphates?
    - Minor Barite
    - Minor quartz
    - 5% Pyrite

  Red-brown lower coarse silty sand. Most grains rounded with aspect ratios .75 to .80, garnets particularly spherical. Portable XRF detects high P, likely from abundance of denser white opaque-clear grains of phosphates (Apatite). High titanium also detected from ilmenite grains. Majority of grains (90%+) .5-2mm diameter

- GC-13-22:
    - 20% Magnetite
    - 20% Hematite
    - 10% Pyrite
    - 15% White-light tan opaque dense minerals (Apatite/phosphates?)
    - 30% Ilmenite
    - 5% Shales/mixed lithology fragments

  Min grain size <.25mm, max grain size 7 mm diameter, average 1.0 mm diameter. Coarse dark brown silty sand. Grain shape is varied from rounded and partially spherical to flattened and shaley. SRF spike in P implies light colored opaque to clear white grains phosphates, like apatite.

- GC-13-32
    - Trace magnetite
    - Trace ilmenite

- Minor Hematite
- 95% Calcite, quartz and schist sand

Min grain size very fine silt, max 2-3mm, average <1mm. Dominantly white silty sand with occasionally larger grains. Sand composed of white calcite, quartz and dark green muscovite schist fragments.

- GC-13-30
  - Trace magnetite
  - Minor garnet
  - Trace ilmenite
  - Almost 100% calcite, quartz and schist sand
  - Trace hematite?

  Min grain size very fine silt, max 2-3mm diameter, average <1mm. Dominantly white silty sand with occasional larger grains. Sand composed of white calcite, quartz and dark green muscovite schist fragments. Occasional garnet and ilmenite grain observed, trace hematite assumed due to high Fe detected on portable XRF and presence in tails sample. Not directly observed.

- GC-13-25
  - 15% magnetite
  - Trace barite, scheelite
  - Trace pyrite
  - Minor ilmenite
  - Dominantly (95%) medium-coarse mixed lithology sand, including quartz, calcite, schist, shale, and other mixed lithics.

  Min grain size .25 mm, max 5.0mm, average 1.5 mm across. Dominantly medium-coarse grained dark brown mixed lithology sand consisting of shales, schists, quartz grains, calcite grains and well mixed lithics. Grains range from flat and shaley to blocky and rectangular. Trace rusted pyrite, trace <1mm size grains of Scheelite or Baryte detected using UV light. Minor ilmenite present, grains reaching up to 1 mm across, cleavage easily visible.

- GC-13-36
  - 5% magnetite
  - 1% pyrite
  - 7% garnet
  - 5% ilmenite

- 82% Quartz/calcite/phosphate/schist/minor mixed lithics sand
- Trace talc

Min grain size .1 mm, max 3.0 mm, average 1 mm across. Dominantly tan, mature quartz, calcite, phosphate, schist and minor mixed lithics sand. Grains range from flat and shaley to blocky to subangular spherical. Trace <1mm barite or scheelite grains detected using UV light. Metallic minerals almost exclusively magnetite and ilmenite. Ti did not show on initial portable XRF scans.

- GC-13-42
  - Trace magnetite
  - 3% Garnet
  - 4% Hematite
  - 4% Ilmenite
  - Trace pyrite
  - 89% Quartz, calcite, muscovite schist, biotite sand

  Min grain size fine silt, max 3.0 mm across, average 1.5 mm. Silty medium coarse tan to light brown sand with dominantly blocky angular grains. Sand consists of quartz, calcite, muscovite schist, biotite and occasional mixed lithics. Garnets fractured, few complete crystals. Hematite easily spotted due to red rusting/streaking. Occasional golden-yellow micas look very similar to gold, like phlogopite. Extreme density difference will prevent confusion.
- GC-13-43:
  - 5% magnetite
  - 15% Ilmenite
  - 7% Hematite
  - 5% Garnet
  - Trace pyrite
  - 68% Quartz, calcite, phosphate, muscovite schist, biotite sand

  Min grain size fine silt, max 3.0 mm across, average 1.5mm. Silty medium coarse tan to light brown sand with dominantly blocky angular grains. Sand consists of dominantly quartz and calcite followed by phosphates, muscovite schist and biotite with occasional mixed lithics. Garnets are fractured with few complete crystals. Hematite easily spotted due to extensive red rusting on metallic surface. Occasional golden yellow micas (phlogopite) look very similar to gold.

- GC-13-38:
  - Trace scheelite

- 1% magnetite
- 10% Pyrite
- 5% Hematite
- 3% Ilmenite
- 3% Garnet
- 78% Evenly mixed quartz, calcite, muscovite and biotite schist sand with occasional mixed lithic grains

Min grain size .25mm, max 5.0mm across, average 2.0m. Silty lower coarse tan to light brown mixed sand with dominantly blocky, angular grains. Sand consist of evenly mixed fragments of quartz, calcite, muscovite and biotite schist with the occasional other lithic grains. Schist fragments common. Trace scheelite grains detected with UV light. Pyrites are fresh with minimal rusting. Hematites moderately rusted and rounded. Ilmenites lack rusting and are rounded.

- GC-13-40:
  - Trace magnetite
  - Trace ilmenite
  - Trace pyrite
  - 5% garnet
  - 10% Hematite
  - 85% Rusty Biotite schist, quartz, muscovite schist sand

  Min grain size silt, max grain size 3.0 mm across, average 1.0mm across. Very silty medium coarse rusty red sand with dominantly blocky, angular grains. Sand consists of dominantly rusty light and dark micaceous shales with mixed quartz, all grains effected by red-brown iron oxide rust staining. Differentiating stained quartz and red garnet grains difficult. Hematite severely oxidized. Trace pyrite grains appear fresh.
- GC-13-37:
  - Trace magnetite
  - Pyrite
  - 2% Garnet
  - 2% Trace Ilmenite
  - 3% Hematite
  - 93% muscovite schist, quartz, phosphates, biotite sand.

  Min grain size silty, max 3.0 mm, average 1.5mm. Silty upper coarse light brown sand with a mix of blocky and flat shaley angular grains. Sand consists of mostly partially rusted micaceous shale fragments with subordinate quartz, phosphates and biotite. Partial to little rust staining.

- GC-13-44
  - Trace pyrite
  - Trace magnetite
  - 7% Hematite
  - 2% Garnet
  - Trace Ilmenite
  - 91% mixed quartz, muscovite and biotite schist sand with occasional lithic grains

  Min size silt, max size 5.0 mm across, avg 1.5 mm across. Silty upper coarse brown sand with a mix of blocky and flat shaley grains. Sand consists of mixed quartz and micaceous schist with biotite and phlogopite. Phlogopite color easily mistaken for gold. Trace pyrites present severely rusted. Little rusting on other grains, including minimal red on hematite grains, generally fresh grains.
- GC-13-45:
  - 10% Magnetite
  - Trace pyrite, severely rusted
  - 3% Garnet
  - Trace Ilmenite, weakly magnetic
  - 2% Hematite
  - 85% Mixed quartz, micaceous shale, phosphates and occasional lithics

  Min size .25 mm, max 7.0 mm, average 1.5 mm across. Silty coarse brown sand with dominantly subangular blocky grains. Sand consists of relatively even mix of quartz, muscovite and biotite schist, phosphates and occasional lithic grains. Mild rust staining, excepting pyrites which are severely rusted. Of the heavy minerals, hematite is mostly in the coarse upper grain size range. Other heavies are finer.
- GC-13-46
  - 20% Magnetite
  - 3% Garnet
  - 1% Hematite
  - Trace pyrite, rusted
  - 75% Dominantly quartz sand with mixed micaceous shale and occasional lithic fragments

  Min size .25mm, max 3.0 mm, average 1mm across. Silty coarse brown sand with dominantly subangular blocky grains. Sand consists of dominantly subangular quartz with mixed muscovite and biotite shales and occasional lithic fragments. Little rust staining except on trace pyrite grains. Sand is dominantly subangular and blocky, though magnetite and hematite are mostly subrounded.

- GC-13-51
    - 2% Magnetite
    - Trace garnet
    - 4% Ilmenite
    - 6% Hematite
    - 88% Quartz, caclite, micaceous schist fragments, and other rust stained lithic sand

  Min size .25 mm, max 7mm, average 1.5mm across. Sandy coarse light brown sand with dominantly blocky subrounded grains and subordinate flat shaley subangular grains. Sand consists of mixed subrounded quartz and calcite grains with flat foliated micaceous schist fragments and other subrounded rust stained lithics. Hematite and ilmenite are the dominant heavy minerals. Moderate rust staining especially on hematites.
- GC-13-53
    - Magnetite 15%
    - 5% Hematite
    - 3% Ilmenite
    - Cassiterite
    - 1% garnet

  Min size .25 mm, max 5mm, average 1.0mm across. Sandy coarse light brown sand with dominantly blocky subrounded grains and subordinate flat shaley subangular grains. Sand consists of mixed subrounded quartz and calcite grains with flat foliated micaceous schist fragments and other subrounded rust stained lithics. Hematite and ilmenite are the dominant heavy minerals with occasional cassiterite noticeable. Moderate rust staining on most grains.
- GC-13-58
    - Trace magnetite
    - Trace ilmenite
    - Trace garnet
    - 99%+ white blocky subangular sand dominantly of quartz sand and white micaceous schist fragments with very occasional mixed lithic fragments. Occasional phlogopite booklets.

  Min grain size silt, Max grain size 7mm. Sample consists almost entirely of white silty medium coarse sand compose of dominantly blocky subangular quartz and white mica schist fragments. Heavies include trace magnetite, ilmenite and garnet grains.

- GC-13-58
    - 2% magnetite
    - Trace garnet
    - Trace pyrite
    - Trace hematite
    - Trace ilmenite
    - 98% white blocky subangular sand dominantly of quartz sand and white micaceous schist fragments with very occasional mixed lithic fragments. Occasional phlogopite booklets.

Min grain size silt, Max grain size 7mm. Sample consists almost entirely of white silty medium coarse sand compose of dominantly blocky subangular quartz and white mica schist fragments. Heavies include trace magnetite, ilmenite, garnet and fresh-rusty pyrite grains.

Appendix B: Pre-field lab test results

Test	Material / Minerals	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Dist Grnd Fines	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	total gold (g)	gold recove red (g)	gold loss - 50#	Au loss %
CassiterL01	Cassiterite	-8#	2.424	No	N/A	N/A	-8#	2.424	10	N/A	N/A	58%	N/A	42%	0.001%	2.25	93%	N/A	N/A	50%	80	8x8"	0.08	0.045	0.032	41.63%
Notes: The sample was coarse cassiterite concentrate with flat gold particles from a low gradient river placer deposit.																										
A total of 58% of clean +50# raw gold was recovered with 42% of low grade (0.001%) -50# slimes.																										
The sample was very low grade which may have influenced the results and grind sample may be too large for the small mill.																										
CassiterL02	Cassiterite	-8#	2.882	No	N/A	N/A	-8#	2.882	15	N/A	82%	N/A	N/A	18%	0.002%	2.81	97%	N/A	N/A	50%	80	8x8"	0.327	0.267	0.06	18.25%
Notes: The same was coarse cassiterite as in L01 but had 0.1507 g of (-14+30#) gold added to existing fine gold particles.																										
With the extra gold and longer grind time, a total of 82% of+40# clean raw salted and original gold was recovered.																										
The better results are probably due to the addition of coarser gold, the amount of -50# slimes was slightly larger - 2.8 kg vs. 2.2 kg for previous test.																										
CassiterL05	Cassiterite	-20#	2.322	No	N/A	N/A	-20#	2.322	10	11%	24%	N/A	N/A	65%	0.08%	2.24	96%	N/A	N/A	50%	80	8x8"	2.93	1.039	1.891	64.54%
Notes: This sample was fine cassiterite concentrate with flat gold particles from a low gradient river deposit.																										
A total of 35% of the clean raw +40# gold was recovered with 65% as dirty -40# gold at 0.1% raw gold.																										
There was too much material on the 50# sieve and so a 40# was used to obtain cleaner gold - need to grind longer?																										
CassiterL06	Cassiterite	-20#	2.222	No	N/A	N/A	-20#	2.222	20	N/A	15%	19%	N/A	66%	0.07%	2.11	95%	N/A	N/A	50%	80	8x8"	2.14	0.731	1.409	65.84%
Note: This sample is the same fine cassiterite with flat gold as in test L05 but was ground twice as long (20 minutes).																										
A total of 33% of the clean raw +50# gold was recovered with 67% as dirty -50# gold at 0.07% purity.																										
The overall recovery has not changed with increased grinding, however the gold appears to have been ground finer.																										
The amount of -40# slimes in L05 (2.24 kg) is similar to LO6 -50# fines (2.11 kg). Maybe too much material in the smaller rod mil?																										
CassiterL07	Cassiterite	-20#	1.000	No	N/A	N/A	-20#	1.000	20	N/A	N/A	71%	0.4%	29%	0.04%	N/A	N/A	N/A	N/A	50%	80	6x6"ba	1.044	0.741	0.303	29.01%
14-Sep-13	Note: This is the same fine cassiterite concentrate sample with flat gold as in test L5 through L9.																									
1 kg of this cassiterite concentrate was ground for 20 minutes in a 6" by 6" diameter ball mill to see if recovery would improve.																										
The recovery improved to 71% of 50# raw clean gold with a bit on gold on the 70# sieve (0.4%) and about 21% of the gold at -70# in low grade 0.04%.																										
This grind had much less material but also a smaller mill 6"by 6" and used balls instead of rods.																										
CassiterL08	Cassiterite	-20#	1.000	No	N/A	N/A	-20#	1.000	15	N/A	N/A	N/A	31%	69%	0.10%	0.91	91%	N/A	N/A	50%	80	8x8"	1.333	0.409	0.924	69.29%
15-Sep-13	Note: This is the same fine cassiterite concentrate sample with flat gold as in test L5 through L9.																									
Note: Only 1 kg of the fine cassiterite sample was ground and for 15 minutes in a rod mill - less than 1/2 of normal sample size.																										
Only 31% of +70# clean raw gold recovered - the sample was over ground considerably - try will less grinding time.																										
CassiterL09	Cassiterite	-20#	1.476	No	N/A	N/A	-20+50#	1.000	10	N/A	N/A	N/A	36%	41%	0.00%	0.94	94%	23%	0.05%	50%	80	8x8"	0.789	0.283	0.32	53.07%
15-Sep-13	Note: This is the same fine cassiterite concentrate sample with flat gold as in test L5 through L9.																									
Note: Only 1 kg of the fine cassiterite sample was ground and for 10 minutes - less than 1/2 of normal sample sieved at 50#.																										
The recovery of clean +70# gold is still only 36% of the total mass including the -50# unground split, that increases to 47% of just the +50# ground product.																										
The -70# material has a high percentage of the remaining clean raw gold 64% at a low gold grade 0.04% and is difficult to table.																										

Test	Material / Minerals	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Dist Grnd Fines	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	total gold (g)	gold recovered (g)	gold loss - 50#	Au loss %
CassiterL10	Cassiterite	-8+40#	1.000	No	N/A	N/A	-8+40#	1.000	10	48%	10%	13%	7%	22%	0.01%	0.89	89%	N/A	N/A	50%	80	8x8"	0.28	0.219	0.061	21.75%
Notes: This sample of cassiterite concentrate was salted with an additional 0.142 g of friable gold particles for this test.																										
The ground concentrate appeared over ground and had 29% of the gold distributed in the -50# size fractions																										
Overall recovery of clean +70# raw gold was 78%, the -70# was about 0.01% gold. The salted gold was coarse (-14+30#).																										
CassiterL11	Cassiterite	-8+40#	1.000	No	N/A	N/A	-8+40#	1.000	10	81%	8%	5%	N/A	6%	0.002%	0.93	93%	N/A	N/A	50%	40	8x8"	0.308	0.291	0.018	5.78%
Notes: This sample of cassiterite concentrate was salted with an additional 0.181 g of friable gold particles for this test.																										
The mill speed was slowed to 40 rpm instead of 80 rpm resulting in a coarser grind, the salted gold was coarse (-14+30#).																										
Overall recovery of clean +50# raw gold was 94%, the -50# ground material was very low grade (0.002%										Previous L10 test indicate that sample in L10 was over ground.																
Garnet L03	Garnet	-16#	1.944	No	N/A	N/A	-16#	1.944	10	49%	12%	15%	N/A	24%	0.01%	1.87	96%	N/A	N/A	50%	80	8x8"	0.664	0.506	0.158	23.81%
This sample is coarse garnet with minor tramp iron, magnetite and hematite and has very flat flakes of gold.																										
This sample had 76% recovery of +50# clean raw gold, but all sieves have lots of garnet to clean.																										
Cannot pan the garnet away. The -50# split is only 0.01% raw gold. Need to grind next sample longer to obtain cleaner gold on each sieve																										
Garnet L04	Garnet	-16#	1.976	No	N/A	N/A	-16#	1.976	20	10%	28%	11%	N/A	51%	0.01%	1.83	93%	N/A	N/A	50%	80	8x8"	0.501	0.248	0.253	50.54%
Notes: This is the same sample of garnet and coarse gold as in L03 but ground for 20 minutes instead of 10 minutes																										
The total +50# clean raw gold recovery is only 49% with 51% of the gold in the -50# split at a low grade of .01% raw gold.																										
Almost only gold on the 30 and 40# sieves, more +40# gold particles, size distribution appears smaller, less tramp iron & magnetics																										
Increasing the grind time results in finer cleaner gold on the sieves but a lower overall gold recovery.																										
Expect that the grind sample in both tests L03 and L04 is too large for the small 8x8" rod mill.																										
Garnet L12	Garnet	-20#	1.000	No	N/A	N/A	-20#	1.000	15	N/A	N/A	69%	9%	22%	0.01%	0.89	89%	N/A	N/A	50%	80	8x8"	0.41	0.323	0.092	22.14%
This sample is coarse garnet with minor tramp iron, magnetite and hematite and has very flat flakes of gold.																										
Note: This is a barren garnet concentrate sample salted with 0.416 g of "G" raw gold																										
The total +70# clean raw gold recovery was 78% with 22% of the gold in the -70# at very low grade of 0.01%																										
Checks at 8 and 10 minutes indicated more grinding needed, but perhaps 10 min would have been enough for 30& 40# sieves?																										
Garnet L13	Garnet	-20#	1.000	No	N/A	N/A	-20#	1.000	40	86%	N/A	N/A	N/A	14%	0.004%	0.79	79%	N/A	N/A	50%	40	8x8"	0.423	0.363	0.06	14.18%
This sample is the same as L12 -coarse garnet with minor tramp iron, magnetite and hematite and has very flat flakes of gold. The speed of the rod mill as 40 rpm.																										
The total +30# clean raw gold recovery was 86% with 14% in the -30# fraction at a very low grade of 0.004% raw gold. The grind time at low speed was 40 min.																										
The mill speed was slowed to 40 rpm but due to the hardness of the garnet it had to be run for 40 minutes to get clean gold on a 30# screen.																										
Gold on all sieves finer than 30# was very dirty and combined for fire assay.																										
This result may be misleading as the salted gold added was relatively coarse (-14+30#) and the material was not well ground.																										

Appendix C: Field test results

Test	Material / Minerals	S.G.	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Au Dist Grnd	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	total Au rec (g)
											595	420	297	210	-70#									
Galena F1	Galena	2.77	-8+50#	4.478	Keene	1.932	43%	-8+50#	1.035	6	71%	N/A	22%	N/A	0.7%	1.4%	0.81	78%	7%	7%	50%	72	8x12	119.94
Note: This is a glaciated steep gradient stream deposit with abundant galena that plugs sluicebox riffles and ends up in the concentrates.																								
This sample was difficult to table to clean conc - prescreened at 50# prior to grinding - distribution of cleaned gold includes pre-screened -50# gold																								
A total of 71% +22% = 93% was recovered as +50# clean raw gold - 5% of the gold was in the unground -50# split.																								
The fine -50# ground fraction is very low grade 1.4% and unground fraction is low grade 5% for a concentrate but could be further upgraded with grinding.																								
Galena F3	Galena	2.77	-8#	4.478	Keene	1.932	43%	-8#	1.103	6.667	66%	N/A	29%	N/A	4.7%	0.5%	0.91	82%	N/A	N/A	50%	72	8x12	40.42
Note: This is a glaciated steep gradient stream deposit with abundant galena that plugs sluicebox riffles and ends up in the concentrates.																								
Notes: This sample was difficult to table to clean conc - not prescreened prior to grinding - total ground -50# is the same (5%) as was in the unground previous sample																								
A total of 66% + 29% =95% was recovered as +50# clean raw gold.																								
The -50# ground and unground -50# galena from previous sample could be collected, ground and sieved at 70# and 100# to clean the -50# fine gold.																								
Cassiter F1	Cassiterite	2.64	-8#+50	54.19	Keene	11.89	22%	-8#	1.133	7	40%	N/A	21%	29%	10%	0.08%	0.93	82%	N/A	N/A	50%	72	8x12	4.8728
Notes: This is an unglaciated low gradient river deposit with abundant cassiterite, minor tramp steel/illmentite and very flat gold flakes.																								
These were Gold Wheel Tailings - they were tabled on Keene table to 22% of original weight, prescreened at 50# and ground for 7 minutes																								
A total of 61% of +50# clean raw gold was recovered with 29% of the gold in a -50+70# at 12 % raw gold and 10% of the gold in a -70# fraction at only 0.1% purity.																								
This fine gold fraction was combined with other -70# and -50# gold later and reground to clean the finer gold sizes.																								
Regrinding of the -50# fines in F4 would increase overall recovery of clean raw gold to									85.45%															
Cassiter F2	Cassiterite	2.66	-8#	0.927	No	N/A	N/A	-8#	0.927	7	37%	23%	17%	16%	7%	5%	0.42	46%	N/A	N/A	50%	72	8x12	31.26
											37%													
Notes: This is concentrate from live bottom long tom 3/4 of the way down (middlings)									77%	of the concentrate was clean +50# gold														
About 23% of the gold was -50# and ranged in purity from 8% (-50+70#) to 5% raw gold (-70#)																								
The higher clean +50# gold recovery is probably due to the smaller sample and nature of the concentrate.																								
Regrinding of the -50# fines in F4 would increase overall recovery of clean raw gold to									96.64%															
Cassiter F3	Cassiterite	2.66	-8+50#	0.76	No	N/A	N/A	-8#	0.578	7	N/A	N/A	59%	8%	2%	2%	0.40	69%	31%	15%	50%	72	8x12	36.13
Notes: This is concentrate from live bottom long tom 3/4 of the way down (middlings)									59%	of the concentrate was clean +50# gold including unground gold.														
This sample is identical to the above (Cassitrer F2) but was pre-screened at 50# and this split was removed										86%	of the screened concentrates was clean +50# gold													
About 10% of the gold was ground to -50# and ranged in purity from 7% (-50+70#) to 2% raw gold (-70#), the unground -50# gold was about 15% pure raw gold																								
Regrinding of the -50# fines in F4 would increase overall recovery of clean raw gold to									94%															
Cassiter F4	Cassiterite	2.66	-8#	0.110	No	N/A	N/A	-50#	0.110	2	N/A	N/A	17%	37%	15%	28%	N/A	N/A	N/A	N/A	50%	72	8x12	9.8116
Note: this is -50+70 ground and unground material from the previous 3 Cassiterite tests, all of the sieves including -70+100# had clean raw gold																								
Therefore the total +100# clean gold recovery was					85%	with	15%	of the gold in the -100 # @			28%	clean raw gold												
Note also that this material was all -50# but after grinding there was 17% of the gold +50# due to the flattening effect of the grinding.																								
Regrinding the -50# concentrates would result in recovering an additional 85% of the previous dirty -50# concentrates.																								
The dirty -100# product may have been clean on a 150# screen which we did not have, but at 28% raw gold it was direct smeltalbe in any event.																								

Test	Material / Minerals	S.G.	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Au Dist Grnd	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	total Au rec (g)
Garnet F1	Garnet	2.95	-20#	22.3	Keene	2.24	10%	-20#	1.12	9	22%	N/A	19%	N/A	59%	95%	1.07	96%	N/A	N/A	50%	72	8x12	107.56
Note: this is an unglaciated low gradient river deposit with abundant garnet, minor magnetite/hematite and very flat gold flakes.																								
Note: this is Gold Wheel/Dieseter table tailings with garnet - it was sieved at 20# to improve tabling and reduced to 10% of original -20# weight.																								
The gold is clean after 9 minutes of grinding even in the -50# fraction for close to 100% of the gold at 90% clean, lead was the main contaminant																								
Lead is impossible to separate from raw gold by grinding as both gold and lead have similar densities and are both malleable.																								
Garnet F2	Garnet	2.95	-20#	7.13	Keene	7.13	100%	-8+20#	1.25	7	91%	N/A	9%	N/A	0.5%	0.2%	1.19	95%	N/A	N/A	50%	72	8x12	49.20
Note: this is Gold Wheel/Deiseter table tailings with garnet - It did not table well and was reconstituted to grind without prior concentration by tabling.																								
This material is the -8+20# portion of the concentrate tailing and was ground for only 7 min and sieved at 50# and retabled to clean the ground minerals																								
The coarsest fraction (+20#) was the dirtiest with lead at 72% clean raw gold, the -20+30# was clean and the -30+50# was only 6% clean raw gold.																								
There was more lead in this size fraction (bullets and fragments of bullets from old timers) and that is why it was more difficult to clean.																								
Garnet F3	Garnet	2.95	-20+50#	22.3	Keene	2.24	10%	-20#	0.44	5	N/A	N/A	58%	N/A	1%	42%	0.36	81%	41%	90%	50%	72	8x12	136.49
Note: this is Gold Wheel/Deister table tailings with garnet - it was sieved at 20# to improve tabling and reduced to 10% of original -20# weight.																								
This is the same material as in the first garnet test but was sieved at 50# and the -50# removed prior to grinding																								
Only 1% of the gold in the original +50# ground material was -50% and it was 42% clean raw gold.																								
The -50# unground material was 90% clean raw gold. This test shows that much of the fine gold in the first test was already present before grinding.																								
SGMg/HeF1	Mag/Hem/Il	2.30	-12#	18.40	Keene	1.36	7%	-12#	**1.36**	**13**	23%	N/A	13%	N/A	**65%**	18%	1.04	77%	N/A	N/A	50%	72	8x12	23.106
Note: These are gold wheel tails from a glaciated deposit high in magnetite and hematite with minor pyrite and flattened gold particles																								
Fines table on Keene table moderately well to low grade concentrate only but with high concentration ratio, gold wheel tails are flat gold which tables okay.																								
This material was difficult to grind and resulted in a high proportion of high grade (18%) -50# fines - too much material in grinding mill																								
Only 36% of the concentrate was +50# clean gold - too much material and the grind time was much too long.																								
SGMg/HeF2	Mag/Hem/Il	2.65	-12#	42.34	Keene	1.764	4%	-12+50#	**1.37**	**12**	9%	N/A	28%	N/A	60%	0.9%	1.07	78%	2%	0.2%	50%	72	8x12	43.83
Note: These are 2nd Hutch Cleanup Jig concentrates - screened at 50# for grinding																								
The original gold in this jig hutch sample was much coarser than in the previous gold wheel tails.																								
The concentrates tabled well and concentrated to 4% of the original volume. However the -50# unground and ground material was impossible to table.																								
Only 37% of the gold was +50# clean gold, there was only 2% of the gold in the unground -50# concentrate at low concentration.																								
The sample was much too large for this material and was over ground from 2% -50# to 62% of the gold distributed in the -50# ground & unground product.																								
These samples should be ground again at very reduced volumes and reduced grinding times in the future.																								
HBWC F1	lead	1.92	-8#	15.36	Keene	1.55	10%	-8+50#	0.742	6	33%	N/A	38%	N/A	15%	14%	1.42284	92%	14%	15%	50%	72	8x12	130.27
Note: This gold wheel tailings sample is from a high bench white channel deposit with lead and minor other high density minerals.																								
This material tabled well on the Keene table with only 10% of the gold wheel tailings ending up as table concentrate for grinding.																								
This material was ground for only 6 minutes and the concentrates combined with the concentrates from the following longer test.																								

Test	Material / Minerals	S.G.	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Au Au Dist Grnd	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	total Au rec (g)
HBWC F2	lead	1.92	-8#	15.36	Keene	1.55	10%	-8+50#	1.48	10	33%	N/A	38%	N/A	15%	14%	average	92%	14%	15%	50%	72	8x12	
About 71% of the gold was concentrated to a +50# mixture of gold and lead (including the unground -50# material).																								
This is equivalent to 84% of the original +50# gold concentrated to a mixture of gold and lead (about 50% lead, 50% gold)																								
The lead had to be hand picked out of the concentrates as per usual, it cannot be separated by grinding or tabling.																								
The larger sample grind size 1.48 vs 0.742 lead to a much longer grind time 10 vs 6 minutes and to overgrinding of the second larger sample.																								
When the concentrates were combined it lead to the lower than optimal recoveries of clean +50# raw gold, abundant lead as also a problem.																								
HBWC F3	tramp iron	3.16	-8#	3.94	Keene	0.455	12%	-8#	3.94	2	N/A	N/A	100%	N/A	0%	0%	N/A	N/A	0	N/A	50%	72	8x12	3.9465
This rusted concentrate tailings sample was busted into chunks with a hammer and then ground for 2 minutes to liberate the free gold particles.																								
The final +50# concentrate was 82% clean raw gold due to the presence of lead in the original rusted concentrate.																								
This test demonstrates the efficiency of the rod mill and Keene table for recovery of gold from rusted/cemented concentrates.																								
Losses not calculated																								
Test	Material / Minerals	S.G.	Feed Size Range	Dry Wt kg	Tabled	Table Con kg	Table Con %	Grind Size	Grnd Wt kg	Grind Time min	Gold Dist +30#	Gold Dist +40 #	Gold Dist +50#	Gold Dist +70#	Dist Grnd Fines	Grnd Fines % Gold	Grnd Fines Wt Kg	Grnd Fines %	Dist Ungrnd Fines	Ungrnd % Gold	Solids %	RPM	Mill dia x len "	
HBWC F4	slag	N/A	-2#	3.20	No	N/A	N/A	-2#	3.20	8	21%	N/A	69%	N/A	11%	25%	2.70	84%	N/A	N/A	50%	72	8x12	310.78
This sample was smelter slag from a botched pour with lots of gold in it, it was broken with a hammer and put in rod mill for 8 minutes.																								
There were a total of 3 batches at 1.1 kg each ground																								
About 90% of the total gold was recovered as clean +50# gold, some lead remained in the -30+50# portion (23% lead), and																								
the -50# fraction was 25% raw gold which is a smeltable concentrate.																								
This test demonstrates the easy recovery of gold from smelter slag using only grinding and sieving.																								
HMg/He F1	Mag/Hem/Il	2.412	-16#	24.12	Keene	12.25	51%	-16+50#	1.19	9	8%	N/A	10%		2%	10%	N/A	N/A	80%	22%	50%	72	8x12	225.73
This sample was gold wheel tailings from a concentrate high in pyrite, lead, magnetite and hematite with minor illmenite and flat gold flakes.																								
The concentrate was difficult to table due to surfing of flat gold particles into middling ports, did two scavenger runs.																								
Most of the gold (80%) was in the unground -50# fraction at 22% raw gold - only a small amount of gold was ground to -50# in size.																								
For the -50# ground portion actual recovery of clean +50# raw gold with lead was 88%. The +30# fraction was 51% clean raw gold.																								
The -30+50# fraction was 95% clean raw gold.																								
To improve recovery of this very difficult sample it would be necessary to regrind the -50# unground and ground again and sieve at 70 & 100# to clean.																								

Appendix D: Post field lab results

				Avg Dens Garnet	2.47				Vol rod	9884	9.884	10	L					
Sample No.				Avg Dend Cassit	3.67													
	Initial	Vol	Charge %	Density	Added Au	Water	Water/ solid	Water/ solid	+50# Au	~% Au	Total Au	-50# Loss	%Loss	Total grind time	Feed grade	Grind Time	Picture	Comments
	g dry	ml dry	kg/L	g/ml	mg	ml	ratio (v	kg/L	mg		mg	mg		min	g/t	min		
Ruby 0.5	456	186	0.0456	2.45	130	186	1.00	2.45	332	90%	298.8	108	27%	10	892.11	0.5	cam 105-0709-10	still coarse not sieved
Ruby 0.5A	474	195	0.0474	2.43	NONE	3025	15.51	0.16	166	95%	157.7	120	43%	11	585.86	8	cell 0.5A 8 min	
Ruby 1	902	360	0.0902	2.51	108	360	1.00	2.51	412	90%	370.8	258	41%	15	697.12	5		Too coarse, longer than time in field. 1:1 not enough water? 50% too much?
Ruby 1B	893	360	0.0893	2.48	106	1000	2.78	0.89	294	90%	264.6	238	47%	16.5	562.82	5		still coarse like ruby 1. reduce water to 1.0 L for 5 minutes
Cassit 1	1443	405	0.1443	3.56	136	1000	2.47	1.44	202	95%	191.9	14	7%	9	142.69	3		too coarse
Cassit 0.5B	723	195	0.0723	3.71	136	1000	5.13	0.72	138	90%	124.2	20	14%	5	199.45	5	cell cassit 0.5B 5 min	40% gold on screen, easy pan.
Cassit 0.75	987	270	0.0987	3.66	0	1000	3.70	0.99	68	90%	61.2	2	3%	6.5	64.03	5	cell 0.75 Cassit 5 min	10% total volume still +50#
Cassit 0.75B	1073	290	0.1073	3.70	132	1000	3.45	1.07	302	95%	286.9	26	8%	6.5	291.61	6.5	cell 0.75 6.5 min	did not add gold. Have to repeat test. Panned and weighed.
Cassit 2	2224	600	0.2224	3.71	108	1000	1.67	2.22	302	85%	256.7	28	10%	13	128.01	9		next largest 1400 g smple took 9 min, too coarse about 50% oversize
Cassit 2B	3150	850	0.315	3.71	126	1000	1.18	3.15	180	90%	162	32	16%	18	61.59	13		still 30% o/s
Cassit 3A	1454	400	0.1454	3.64	112	3000	7.50	0.48	182	95%	172.9	44	20%	10	149.17	9	cell cassit 3A 9 min	same weight and time as cassit 1, not same reduction
Cassit 3B	2299	615	0.2299	3.74	102	3000	4.88	0.77	96	95%	91.2	20	18%	16.5	48.37	13	cell cassit 3B 13 min	3L water analog to Cassit 2. 10% still o/s
Cassit 4A	1448	400	0.1448	3.62	104	400	1.00	3.62	164	95%	155.8	20	11%	8	121.41	7.5		1:1 water analog of cassit 1 and cassit 3A. Close at 7.5 min
																8	cell cassit 4A 8 min	panned
Cassit 0.5	716	195	0.0716		124	1000	5.126		96	100%	96	46	32%	7	198.3	7	Cell: cassit 0.5 7 min	overground? Clean gold on sieve.

Overground. Clean gold on sieve, not representative of other

Appendix E: Efficiency simulation

Grade			Increment				Mill Volume			Reload Time			Goal		
18%			0.3	kg			10	L		2	minutes		500	g	
Mill Charge kg	Mill charge kg/L	Grind Time (min)	Loss	Recovery	Gold rec'd	Gold Lost	Total Gold (g)	Reloads needed	Reload time needed (min)	Grind time needed (min)	Total time (min)	g/min produced	gold lost		
0.5	0.05	3.64775	7.74%	92.26%	83.03	6.96693375	90	6.02169741	12.04339482	21.96565	34.00904	14.70197	41.95277	0.014699	14.69862
0.8	0.08	5.38556	8.87%	91.13%	131.23	12.7736842	144	3.810211365	7.62042273	20.52012	28.14054	17.76796	48.67044	0.017763	17.76268
1.1	0.11	7.12337	10.00%	90.00%	178.20	19.8003772	198	2.805842078	5.611684156	19.98705	25.59874	19.53221	55.55673	0.019526	19.52554
1.4	0.14	8.86118	11.13%	88.87%	223.95	28.0470128	252	2.232611435	4.46522287	19.78357	24.24879	20.61958	62.61808	0.020612	20.61187
1.7	0.17	10.59899	12.26%	87.74%	268.49	37.5135911	306	1.86229166	3.724583319	19.73841	23.46299	21.31015	69.86125	0.021302	21.30166
2	0.2	12.3368	13.39%	86.61%	311.80	48.200112	360	1.603592622	3.207185244	19.7832	22.99039	21.74822	77.29334	0.021739	21.73912
2.3	0.23	14.07461	14.52%	85.48%	353.89	60.1065755	414	1.412854734	2.825709467	19.88538	22.71109	22.01568	84.92186	0.022006	22.00609
2.6	0.26	15.81242	15.65%	84.35%	394.77	73.2329816	468	1.266569842	2.533139684	20.02753	22.56067	22.16246	92.75469	0.022152	22.15248
2.9	0.29	17.55023	16.78%	83.22%	434.42	87.5793304	522	1.150958126	2.301916253	20.19958	22.5015	22.22074	100.8001	0.02221	22.21044
3.2	0.32	19.28804	17.91%	82.09%	472.85	103.145622	576	1.057407995	2.11481599	20.39533	22.51014	22.21221	109.067	0.022202	22.20163
3.5	0.35	21.02585	19.04%	80.96%	510.07	119.931856	630	0.980261178	1.960522356	20.61082	22.57135	22.15198	117.5645	0.022141	22.14118
3.8	0.38	22.76366	20.17%	79.83%	546.06	137.938032	684	0.915646995	1.831293991	20.84348	22.67477	22.05094	126.3025	0.02204	22.03995
4.1	0.41	24.50147	21.30%	78.70%	580.84	157.164152	738	0.86082841	1.72165682	21.09156	22.81322	21.91712	135.2914	0.021906	21.90597
4.4	0.44	26.23928	22.43%	77.57%	614.39	177.610213	792	0.813815612	1.627631223	21.35394	22.98157	21.75657	144.542	0.021745	21.74528
4.7	0.47	27.97709	23.56%	76.44%	646.72	199.276218	846	0.773127591	1.546255183	21.62986	23.17612	21.57393	154.0659	0.021563	21.56253
5	0.5	29.7149	24.68%	75.32%	677.84	222.162165	900	0.737639557	1.475279113	21.91889	23.39416	21.37285	163.8756	0.021361	21.36134