Wash Plant Design Studies and Equipment Examples

Construction of Settling Ponds

Rehabilitation of Fish Streams after Mining

for

Placer Miners

Thirteen Reports Assembled by Kerwin Krause

May 2004

The Ten Commandments in Placer Exploration and Operation

Ron Stokes, P.E.
President, Stokes Exploration Management Company, Ltd., Vancouver, B.C.

We're going to talk about the 10 Commandments of Placer Exploration and Production this morning. The 10 Commandments, however, reminds me about the vicar in England who was really upset that somebody had stolen his bicycle. He said to his curate, his helper, "You know, I just can't understand why anybody in a small English village like this should steal somebody's bicycle." He said, "I could understand it if it was Los Angeles or New York or Vancouver, but it shouldn't happen in a small English village." So he said, "I've decided what to do. I'm going to preach a sermon on the 10 Commandments and when I get to 'Thou shalt not steal' I'm going to look around the congregation and I'll know who stole my bicycle." After the sermon the curate came up to him and said, "Vicar, it was wonderful. I have never heard such a sermon. It was just incredible. You turned me over inside. I was inspired. But sir, you didn't finish." The Vicar said, "Well, as a matter of fact, when I got to 'Thou shalt not commit adultery,' I remembered where I left my bicycle."

The 10 Commandments for Placer Examination

1. Take BIG samples. Reduce them in the field if necessary by washing and screening.
2. NEVER reduce your sample to assay size by splitting down.
3. Placer Miners recover AVAILABLE GOLD in their washing plants. Your testing method should match this.
4. Weigh your sample AND measure its volume.
5. Remember that the BOULDERS you didn't include in the sample are part of yardage estimate. Apply the BOULDER FACTOR.
6. Placer material SWELLS 20 - 30% when excavated. Include that in your INPLACE YARDAGE ESTIMATE.
7. Gold is weighed in TROY OUNCES. There are 31.10 grams in 1 Troy Ounce. There are 28.35 grams in an AVOIRDUPOIS ounce. 1 ounce of gold is 10% heavier than 1 ounce of feathers.
8. DO NOT FIRE ASSAY PLACER SAMPLES (except when checking for TOTAL gold content). You will get a substantial over valuation. The FIRE ASSAY sample is usually 10 to 15 grams, not all the 2000 gram sample of concentrate you submitted. TELL THE ASSAYER these are placer samples to be treated in the way YOU designate.
9. SALTING is easy to do in placer sampling. One speck can influence the values. Most likely it will be accidental, caused by YOUR poor procedure. Take BLANK SAMPLES.
10. One placer sample is almost useless as a value estimate. Ten give you a guide. Fifty give you a statistical pattern. (Only 6% are likely to be close to the true grade.) BUT two good colors in a panned sample are of economic interest.

10 Commandments for Production in Placer

1. FINE GOLD RECOVERY has to be an ECONOMIC exercise.
2. When operations are going smoothly, keep going. You will get enough breaks during the winter months. Down time is expensive. Preventative maintenance is a MUST.
3. Screen off the oversize that doesn't carry gold. CLOSE SIZING is the key to good gravity recovery.
4. Monitor your sluice box (or other recovery plant) closely. Know its SLOPE, the VOLUME of water, the VOLUME AND RATE of feed. (Do not guess.)
5. Prewash and condition your feed, liberate gold from the gravel and clay. Feed the box with a CONSTANT input.
6. "LOADS" vary. Do you REALLY KNOW how much feed you put through?
7. STRIPPING RATIOS are important. It costs money to remove overburden; critical when you have a marginal pay channel grade.
8. Clean to bedrock and beyond. (3-4 feet sometimes.)
9. The gold you weigh is not pure. It may be 700 - 900 fine. 70% to 90% gold content.
10. Thou shalt not POLLUTE, or silt-up downstream waters. Settling ponds are mandatory.

A Comparison of Sluicebox Riffle Systems

Abstract

The Sluicebox

The sluicebox has been used for the recovery of placer gold since ancient Greece (Jason's golden fleece) and it is still the most important placer gold concentrator in Alaska and the Yukon Territory. A sluicebox is a rectangular flume containing riffles through which a dilute slurry of water and alluvial gravel flows.

Sluiceboxes can provide a much higher concentration ratio (20,000:1) than most other gravity concentrators such as jigs and spirals. Sluiceboxes operate well with dilute slurries (5 to 15% density by volume) which are common to land-based washing plants and suction dredges. They tolerate minor variations in feed rates and solids density better than any other gold recovery equipment. Sluiceboxes are also very reliable, inexpensive, and simple to operate. They often process pay gravels with grades from as little as 0.2 g/m3 (0.005 oz/yd3) and commonly about 0.5 to 1.0 g/m3 at high volumes (up to 300 cubic metres per hour).

Riffles

The most common sluice riffles include expanded metal, angle iron (Hungarian) and flat bar riffles. Matting (Nomad) is usually placed under the riffles to help retain the gold particles. Clarkson (1994, 1997, and 1998) conducted field and laboratory research using conventional and radiotracer testing to determine optimum design and operating parameters for expanded metal and one inch angle iron riffle systems. Clarkson reported extremely high gold recovery efficiencies for systems designed and operated strictly according to his recommendations.

In recent years, many placer miners have tried alternative systems including oscillating riffles and hydraulic riffles. In an oscillating riffle, sluice runs fitted with expanded metal and matting are suspended from cables and oscillated in a high frequency circular motion.

Hydraulic riffles were originally developed in New Zealand as a kind of "poor mans jig". These riffles consist of alternating 2 inch vertical flat bar and perforated 1 inch square tubing. Low pressure, filtered water is injected through holes in bottom of the tubing to help keep the riffles dilated and soft. The author is unaware of any comprehensive comparison of these two systems but some operators have reported "better fine gold recovery" and lower water requirements.

Radiotracer Testing Technology

Walsh (1986) was the first to research and develop the use of radioactive gold (Au198) to test gold recovery equipment. Clarkson further developed field testing procedures using radioactivated gold as tracers to provide a statistically valid, rapid, simple, cost-effective and safe method of evaluating the gold recover efficiency of virtually any device which recovers or samples gold. These include sluiceboxes, jigs and drills. Radiotracer testing

technology avoids both the high costs and the unpredictable error levels common when conventional testing procedures are applied to ores containing free gold particles.

When gold particles are placed in a nuclear reactor, some of their nuclei capture an extra neutron to forms gold's radioactive isotope (Au198) which emits gamma and X-rays. These rays are readily identified with a scintillometer. Radioactive gold has a very short half-life of 2.7 days and rapidly disintegrates to normal background levels of radiation within a few weeks, thus eliminating the long term storage problems associated with other radioactive materials. Radiotracers can easily be protected from intentional contamination because the use of even mildly radioactive gold is licensed and strictly controlled by regulatory agencies.

The Comparison

In response to claims and concerns about oscillating and hydraulic riffles, the author compiled and analyzed data from private and publicly funded radiotracer testing programs for the period 1989 to 1997. The gold recovery performance of dozens of sluicing systems which were similarly equipped but were fitted with various riffle systems were compared. Recommendations for the optimum operating conditions for each riffle type were developed.

The results are best summarized on the graphs (figures 1, 2 and 3) which display the gold recovery in percent for a variety of gold particle sizes ranging from 100 mesh (150 microns) to 14 mesh (1.2 mm). Hydraulic riffles had the lowest fine gold recovery and had extremely low gold recoveries when unit feed rates (cubic yards per foot of width) approached those commonly used with conventional sluices (figures 1 and 2). Hydraulic riffles may be suitable for coarse gold recovery (nugget trap) and for use at very low unit feed rates.

Oscillating sluices generally had lower fine gold (100 and 48 mesh, 150 and 300 micron) and lower coarse gold (14 mesh, 1.2 mm) recovery than conventional systems which were designed and operated strictly in compliance with Clarkson's recommendations (figure 1). The coarse gold recovery of oscillating sluices can be improved with the addition of one inch angle iron riffles to the sluice runs (figure 3). In addition, it was concluded that oscillating sluices required the same operating and design parameters as conventional sluices. The use of low water quantities and low slurry velocities resulted in even lower gold recoveries, especially in the finer sizes.

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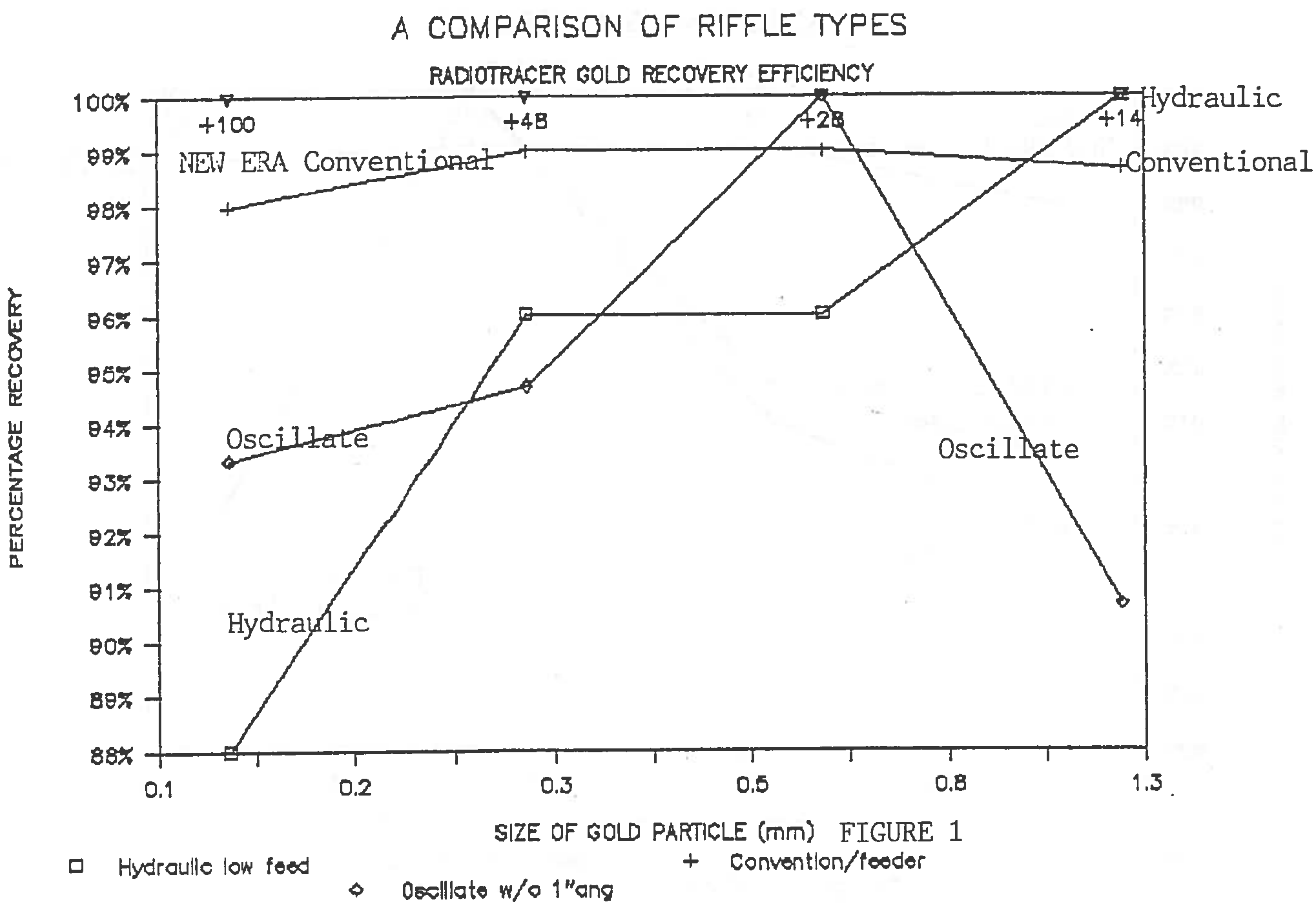

FIGURE 1

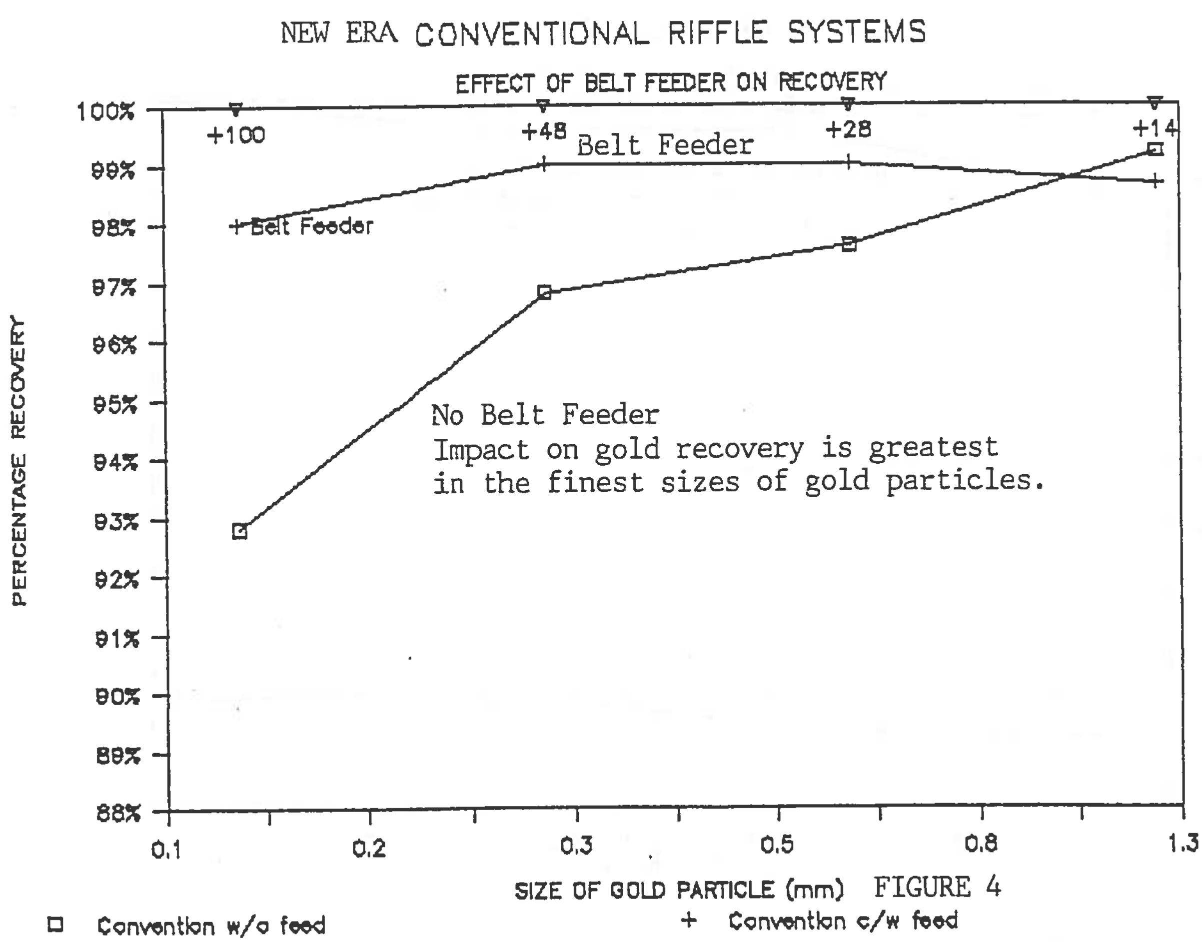

FIGURE 4

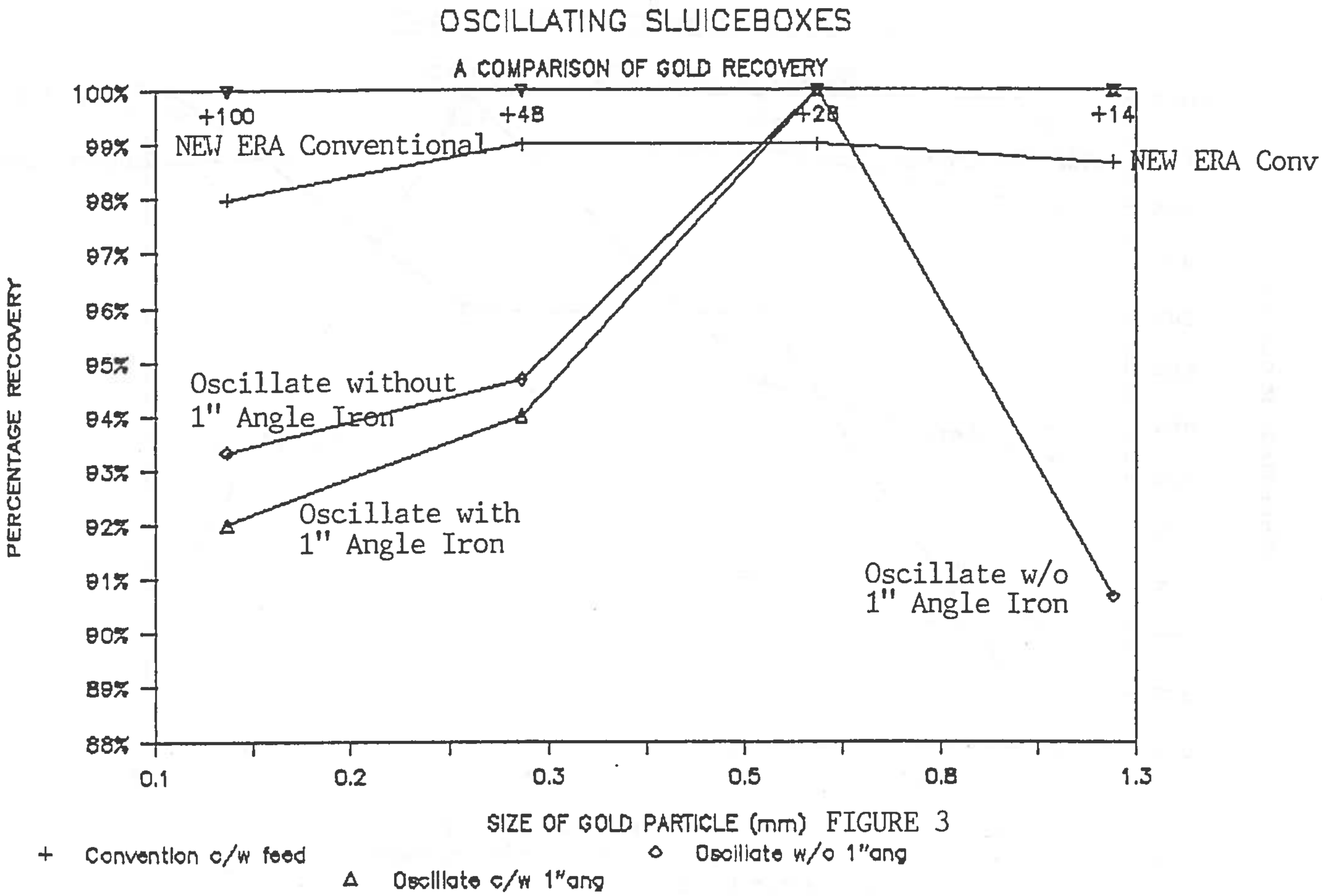

FIGURE 3

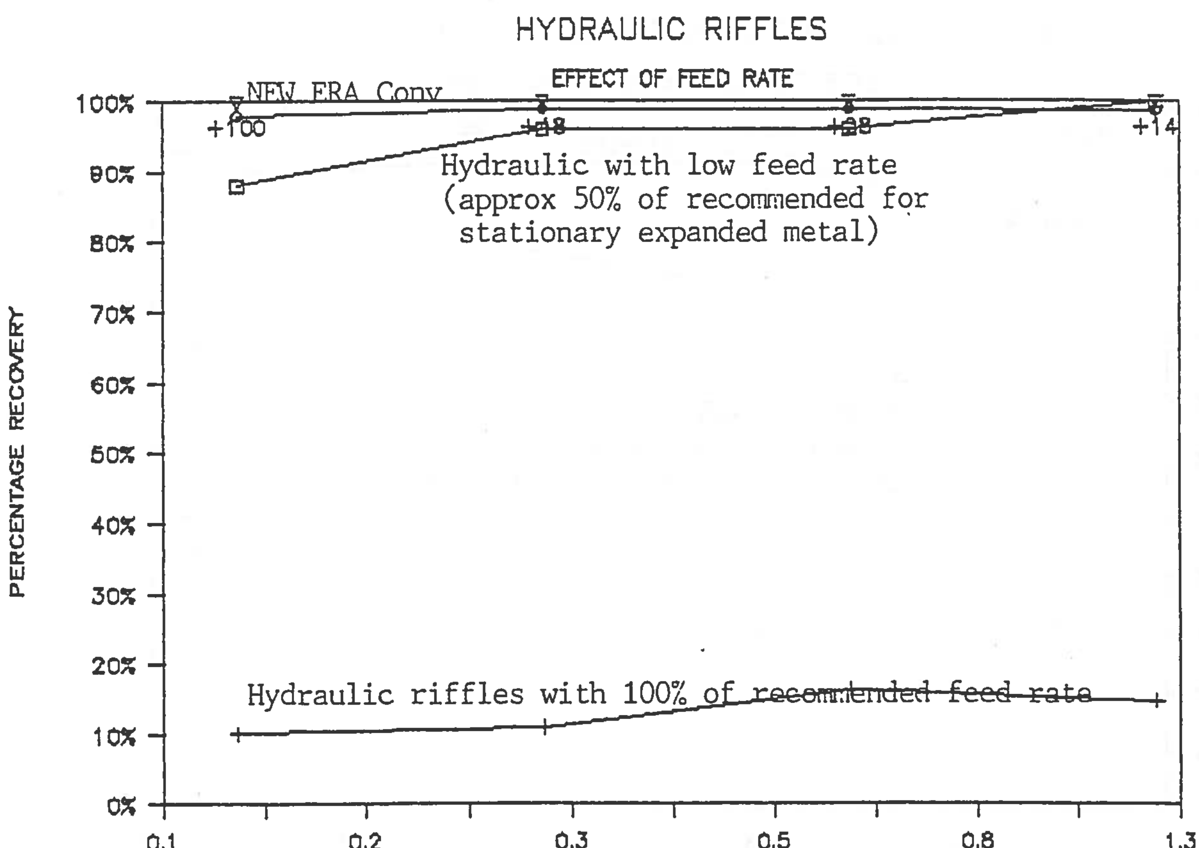

FIGURE 2

The Use of Spray Flotation for the Recovery of Fine Gold, Frazier River, B.C.

Joe Bork, Vice President, Business Development, AFT, Inc.

AFT has recently completed a successful field test with our mobile pilot plant processing the tailings stream from a British Columbia placer gold operation. The main objective of this test was to demonstrate the ability of the AFT patented Sprayflot™ flotation system to efficiently recover fine gold from placer tailings. Figure 1 pictorially shows the distinction between AFT's spray flotation process and two other flotation processes; conventional subaeration type flotation and column flotation. Spray flotation has been demonstrated effective in the flotation of fine coal, copper, molybdenum and nickle.

Several key features of the AFT spray flotation cell design include:

1. The surface feed application provides that;
   a. The cleaned product does not travel through gangue laden material.
   b. The recoverable product has a higher probability of floating.
2. The high shear nozzles;
   a. Break flocs and improve cleaning.
   b. Create microbubbles.
3. The internal recycle allows for;
   a. Additional froth generation.
   b. A "second chance" for recovering product.
4. The skimmers and froth drain plate control froth height, solids content, and drainage time.

AFT recently employed a flotation flowsheet to process placer gold tailings from a gravity plant, operated on the Frazier River, B.C., Canada. The feed to the flotation pilot plant was tailings from a table circuit and the middlings and tailings from a spiral circuit. Both feeds were -10 mesh with 5-20% +28 mesh and 30-50% -100 mesh. Silica and garnet constituted the major portion of the large size fraction. Black sands were the primary constituents of the finer fraction. The gold recovered was more than 50% -100 mesh; mostly of a flat, plate-like nature. AFT's flotation cells operated very well even with the large size fraction material. Because the material is fed through a spray system on top of the cell, there was no sanding problem generally inherent with processing large high density material via conventional flotation cells.

A key component in determining how efficient a separation device operates is the content of the precious metals in the tails of the circuit. The AFT system consistently recovered precious metals while discharging a tails stream with grades as low as .002-.004 oz gold/ton. This equates to approximately 0.1 part per million of gold in the tails stream.

A second measure of efficiency for separating devices is the ability to increase the grade of the precious metal from the feed to the concentrate. The AFT system consistently upgraded the feed, averaging 0.17 oz gold/ton, to a concentrate averaging over 600 oz gold/ton. Based on the assays of our flotation feed and tails, we recovered over 97% of the gold from the feed.

These continuous pilot plant test results are equal in recovery to those obtained in lab tests with one of AFT's batch cells, but the field grade achieved is four times higher. The large particle size of the sand was not an influencing parameter in the larger continuous pilot plant operation. Given these results, AFT believes they have demonstrated the capability of their flotation system both as a potential precious metal recovering device and as an efficient mineral upgrader. This system can complement placer gravity separation systems and may be able to increase the overall recovery of precious metals to the 99% level.

AFT conducts laboratory testing of submitted samples to determine their amenability to spray flotation processing. Lab test data provided, cost and time schedule are summarized in Table 1. AFT also has the capability to provide continuous on-site sampling and processing via their Sprayflot™ flotation equipment. Table 2 summarizes the 1990 costs of such test work.

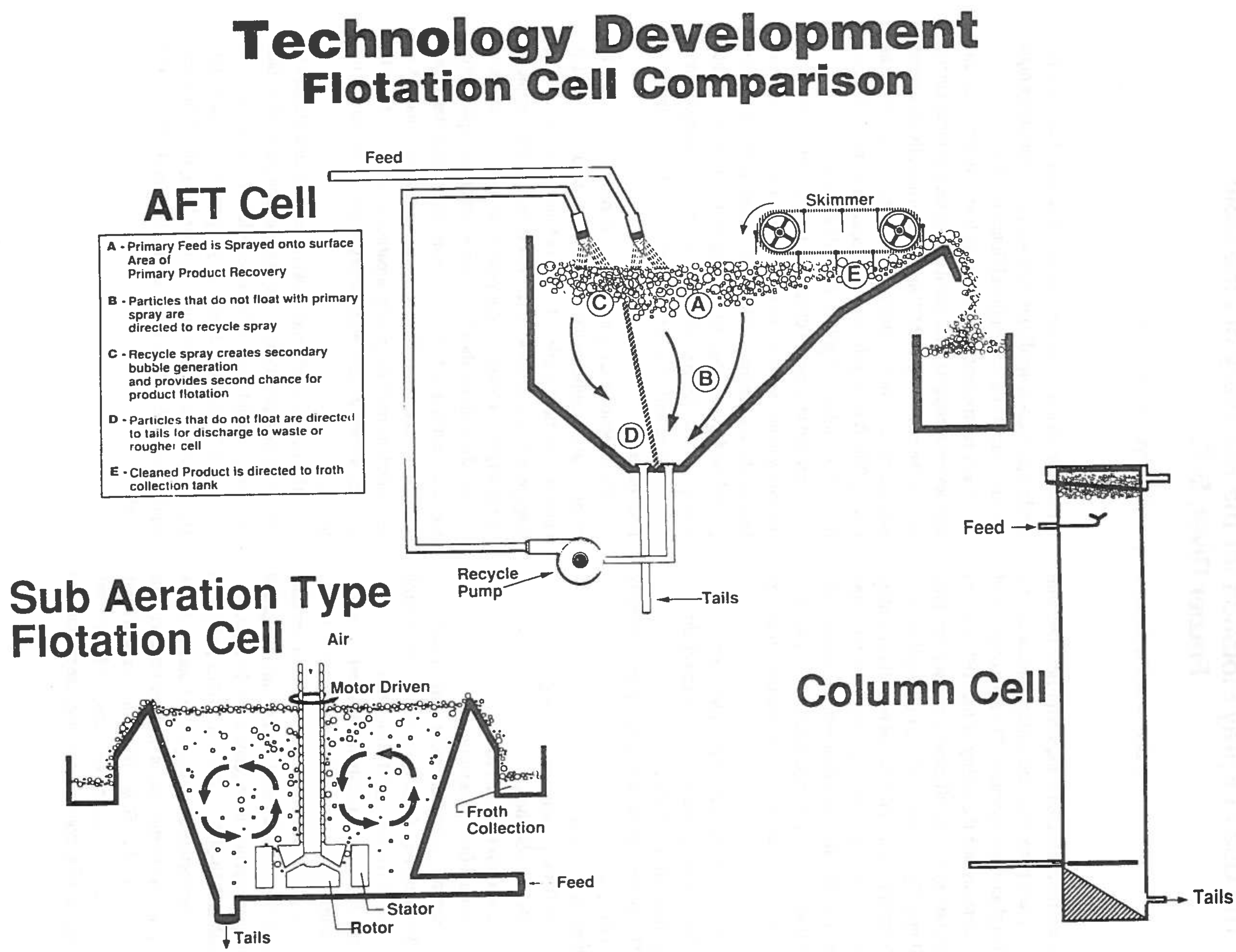

Figure 1. Comparison of flotation cell technology.

Table 1. 1990 Laboratory Costs, Timing and Tests Provided by AFT, Inc.

CUSTOMER INFORMATION :

- o Objective of Flotation Tests
  - Target Product Grade / Recovery
  - Target Tails Grade
- o Sample Data and Operating Conditions
  - Grind
  - Reagents / Dosage
  - pH
  - Percent Solids
  - Special Conditioning Requirements
- o Sample Required by AFT
  - 5 - 10 kg (dry solids) in Sealed Package
  - Special Packaging Suggested if Necessary to Preserve Sample
- o Additional Plant Background (Optional)

AFT WILL PROVIDE:

- o Flotation Test Report Including:
  - Summarized Grade / Recovery Curves
  - Flotation Kinectics
  - Metallurgical Performance Based on Customer's Objectives

TIMING:

- o Standard Scheduling - 4 to 5 Weeks

COST:

- o Standard Rate - $2500 to $3000

Table 2. 1990 Field Tests, Costs and Processing Options Provided by AFT, Inc.

		5 DAY TEST	10 DAY TEST
7-Liter Batch			
Equipment Lease	@ $175/day	875	1,750
Transportation	@ $350	350	350
AFT Engineering	@ $500/day (1/2 day set up/1/2 day takedown)	3,000	5,500
	Plus expenses ($100/day + air fare)	1,675	2,100
		$ 5,900	$ 9,700
20-Liter Batch Test			
Equipment Lease (2 units)	@ $500/day	2,500	5,000
Transportation	@ $750	750	750
AFT Engineering (2)	@ $500/day (1 day set up + 1 day takedown)	7,000	12,000
	Plus expenses ($100/day + air fare)	3,450	4,450
		$13,700	$22,200

		5 DAYS	10 DAYS	30 DAYS
2-Ton/Hr Mobile Pilot Plant				
Equipment Lease	@ $1000/day	5,000	10,000	30,000
Transportation	@ $4000	4,000	4,000	4,000
AFT Engineering (2)	@ $500/day (3 days set up/ 1 day takedown)	9,000	14,000	34,000
	Plus expenses ($100/day + air fare)	3,800	4,800	8,000
		$21,800	$32,800	$76,000

Mining on the Middle Fork, Koyukuk River, Alaska, Using the MI-LO Alluvial Separation System

Lonnie McClung, President, Material Separation, Inc.

Wash Plant and Alluvial System

A wash plant was designed to maximize the cleaning of placer material, while using less water and providing a uniform feed to the alluvial separation system. Placer material was first fed to an 8' diameter mixing zone, at the head of the trommel, by way of a plate feeder and feed conveyor. Water was added at this location to bring the material to a consistency of soupy ready-mix concrete. The recycled water added contained approximately 30% solids (ultra light sands and settleable solids) and provided an abrasive medium. The slurried material was lifted into a 4' diameter trommel by a lifting trough and tumbled approximately 210 linear feet at a rate of 2.7 feet per second. The material was then rinsed with clean water and screened over a 1-1/2" bar grizzly with the oversized being conveyed to oversize tailings. The minus 1-1/2" material was screened on a 1/4" screen. The 1-1/2" x 1/4" material was processed through a 30" wide by 14' long sluice box with clean water.

The -1/4" slurry was pumped to the top of the alluvial processing tower (an elutriator), where the -1/4" slurry was classified into five fractions according to settling velocity:

1. Waste slurry, which was removed with an ascending water flow rate equal to approximately the settling velocity of a 600 mesh gold sphere. (Corey shape factor of 1.)
2. Ultra Light Sands, which were removed from the system with an ascending water flow rate equal to the settling velocity of a 200 mesh gold sphere. The ultra light sands were processed through a sluice box 5' long and 12" wide.
3. Light Sands, which were removed with an ascending flow rate equal to the settling velocity of a 100 mesh gold sphere and processed through a sluice box 14' long and 18" wide.
4. Coarse Sands, which were removed with an ascending flow rate equal to a 30 mesh gold sphere and processed through a sluice box 14' long and 18" wide.
5. A gravel fraction was removed and processed through a sluice box 14' long and 30" wide. This fraction contained gold having a settling velocity greater than that of a 30 mesh gold sphere.

Examination of the oversize coarse tailings and the water quality at the end of each sluice box showed the effectiveness of the washing plant. The oversize rock was washed clean, with no material deposits in cracks or crevices, even while running large amounts of clay through the plant. The discharge water, at the end of each sluice box, remained clean even after the material had tumbled the length of the riffles. While running large quantities of clay through the plant, no deposits of clay formed in the sluice boxes.

The wash plant was field tested at Tramway Bar mine on the middle fork of the Koyukuk River during the summer of 1989. The mine owners are Glen and Lela Bouton. The unit had a design capacity of 100, bank run, cubic yards per hour and most of the fabrication was done on site. The construction and fabrication of the washing plant was by Hector's Welding with Lonnie McClung, design engineer, in attendance to supervise the construction and fabrication.

The washing plant was conceived by Mickey McClung. The trommel, which was fed by a feed conveyor by way of a dozer trap and plate feeder, was built and designed by Hector's Welding. Oversize material was also stacked by a conveyor. Sluice box tailings were removed by a front-end loader. The entire plant, from feeder to conveyors, was electric powered. A 50 kw Emerson diesel generator supplied the electricity.

A rock kicker was used on the feed conveyor to remove large rocks (+20"). The washing plant handled rocks up to 20" in diameter. During the testing program one rock stuck in the trommel lifting trough. This was easily removed. Two rocks, that got past the rock kicker, but which were too large to be picked up by the lifting trough, had to be removed from the trommel lifting trough by hand. During the field testing program a total of 42,000 cubic yards of bank run material were processed.

Ground Support Equipment

Ground support equipment included:

Loader - Alice Chalmers 745, to remove sluice box tails.
CAT - John Deere 750, for striping and pushing to feeder.
Scraper - 22 Yard D-W 21, to remove oversize tailings.
4" x 3" Solids Pump, 10 hp.
6" pump, 10 hp, Gorman Rupp.

4" pump, 10 hp, Gorman Rupp.
Generator - Emerson 50 kw.

Personnel:

1. A hired hand to operate the Cat and Loader
2. Glen Bouton, Co-owner of mine, general superintendent. Oversaw the project and assisted with tailings removal and feeding the plant.
3. Lela Bouton, Co-owner of mine. In charge of camp, safety regulations and training. Assisted and supervised gold cleanups.
4. Mickey McClung, overseer of testing program. Recorded test results and assisted Lonnie McClung. Assisted Lela Bouton with camp supervision and gold cleanup.

Material Characteristics

The bank run material gave us a wide range of materials for processing. The material included:

1. Dry to wet muck.
2. Material with a high percent of sand.
3. Silts.
4. Clays (1-1/2 foot layer).
5. Large rocks up to 5 ft. diameter.
6. Layers of concentrated black sands.
7. Flat, round and fractured bedrock.
8. Topsoil with leaves, organics, twigs, branches, etc.

The average grain size distribution of the bank run material was:

Size Fraction	Fractional %
1) +20"	1%
2) 20" x 1-1/2"	24%
3) 1-1/2" x 1/4"	24%
4) 1/4" x 1/8"	12%
5) 1/8" x 1/16"	7%
6) 1/16" x 1/32"	5%
7) 1/32" x 1/64"	12%
8) -1/64"	15%
Total	100%

51% of the bank run material was -1/4" and was processed through the alluvial separation system.

Water Quality Control and Usage

Sluice box process water was drawn from a recycle pond approximately 60' in diameter by 3' deep. Sluicing water was allowed to flow back into this pond after processing the material, with continuous recycling of this water. The slimes (organics, clay-size particles, ultra light sands, floatables, suspended solids, and settleable solids) were discharged to a separate settling pond approximately 150' in diameter by 6' deep. Make-up water was drawn from the middle fork of the Koyukuk River.

Water samples were taken from the settling pond each day prior to start up, during the day, and at the end of the day. The results of this testing and the tests conducted by the State of Alaska Department of Environmental Conservation clearly show that removing the slimes in a concentrated form prior to discharging them to the settling pond, causes them to settle from the water at a much faster rate, leaving the water cleaner than our water source.

Over the summer 42,000 yds^3 of material were processed with the entire waste stream going to a pond 150' in diameter by 6' deep. There was no overflow back to the river and no recycling of this water. Waste slurry was concentrated. Waste water to the settling pond, to remove all of the slimes that would normally be discharged to a settling pond, was approximately 108 gallons of water per cubic yard processed. A conventional operation might discharge 1630 gallons per cubic yard processed to the waste pond; a reduction of 93.8%, using the MI-LO alluvial separation system.

Compared to recycled plant process water, as per measurements taken at the feed water manifold, sluice box discharge water streams showed increases in turbidity averaging 24.5 NTU, with a high of 42 NTU and a low of 15 NTU. However, T.S.S. in the sluice box discharge water streams were less than those found in the manifold water, the average being 57.3 mg/l, which was 107.7 mg/l less than the water entering the system.

Riffle and Sluice Box Design

During the summer of 1989, we tested numerous riffle designs, including California Riffles, Straight Riffles, Dredge Riffles, Hungarian Riffles, and Expanded Metal. Various sluice box water flow rates and sluice box slopes were also studied.

We preferred expanded metal over 3/4" carpeting with a 2-1/4" per foot slope on the boxes. There was very little difference with the different riffle designs, but a large influence by the slope of the box. With anything less than 2-1/4" per foot, we experienced riffle packing, even though we removed all fines going to the sluice box. Riffles would still pack with black sands from the top, down, leaving a loose free bed of larger rock underneath the packed riffle, even though the black sands were much smaller than the material underneath. (See Figure 1)

We also found that our sluice box worked backwards

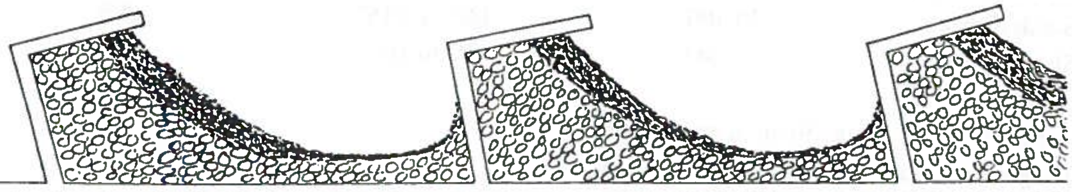

Figure 1. Material layering observed in sluice riffles.

compared to what would normally be found or thought. Our smallest gold was found at the head of the box in the first riffles. Gold size increased towards the end of the box. We theorize that small pieces of gold did not have to compete with a bed of sand, smaller in size, in order to find a place to come to rest. All of the smaller rocks were removed by the MI-LO alluvial separation system prior to reaching the sluice. As the gold increased in size, there was more competition between the small gold and black sands to find a place to come to rest.

The dimensions of the five sluice boxes are given below.

Ultra Light Sands box	12" wide by 5' long. The slope of box was 1-1/2" per foot. This box angle could not be changed.
Light Sands box	18" wide by 14' long
Coarse Sands box	18" wide by 14' long
Gravel box	30" wide by 14' long
+1/4" to -1-1/2" box	30" wide by 14' long

Total width of sluice boxes was 9'(108")

Material and Water Balance per Sluice Box:

	Water	Material
Ultra Light Sands	—	—
Light Sands	136 gpm	5 yards per hour
Coarse Sands	190 gpm	7 yards per hour
Gravel	326 gpm	12 yards per hour
+1/4" to -1-1/2"	652 gpm	24 yards per hour

Incorporated into the plant's coarse sand sluice was a 6" wide by 3' long clean-up sluice box (Figure 2). The coarse sands sluice box, which was capturing the highest percentage of gold, was adapted with a hopper approximately 30" from the head of the box. This hopper was the same width as the sluice box and 12" long (18" wide by 12" long). Material trapped in the hopper was drained from the bottom at a constant rate of 3 gpm to the 6" by 3' sluice box. Water was added to the sluice box to give a 4:1 water vs. material ratio.

The hopper and sluice box were designed to be able to regulate the flow of material to the sluice and/or stop the flow at will. The sluice box could be removed and cleaned without shutting down the plant and/or operation. This proved to be a valuable tool in the operation and testing program. By taking ten second samples being discharged from the hopper, the owner estimated what his ground grade was running at that time. The owner came to depend on these samples which, in fact, allowed him to avoid running areas of the cut which were very low grade.

Gold Recovery

It was not possible to compare the percentage of gold recovered during 1989 to previous years because the plant was operated in a new area. Also, the owners had never run a grain size distribution on gold recovered in previous mining seasons.

The 1-1/2" x 1/4" material was processed through a 30" wide by 14' long sluice box at a 4:1 water : material ratio. Gold recovery was efficient because gold did not have to compete with small rocks for a place to settle. The absence of fine gold in this sluice run and the clean water at the end of the sluice box demonstrated the efficiency of the washing plant.

Table 1 shows the gold size vs. material size and the percentage of gold recovered in each sluice box. These may be compared with other operations and areas. Gold sizes are based on settling velocities of gold with a Corey shape factor of 1.

Table 1. Gold Size and Distribution Among Sluices

	Gold Size (mesh)	Waste Material Size	% of Total Gold Recovered
Ultra Light Sands Sluice	200-600	1/32" to 1/64"	3%
Light Sands Sluice	100-200	1/16" to 1/32"	7%
Coarse Sands Sluice	30-100	1/8" to 1/16"	69%
Gravel Sluice	30-1/4"	1/4" to 1//8"	21%

79% of the gold recovered was 30 mesh and smaller.
21% of the gold recovered was coarser than 30 mesh.
(You should note that this does not consider the +1/4" gold, if any.)

For those readers wishing greater detail, a final report of the project is available.

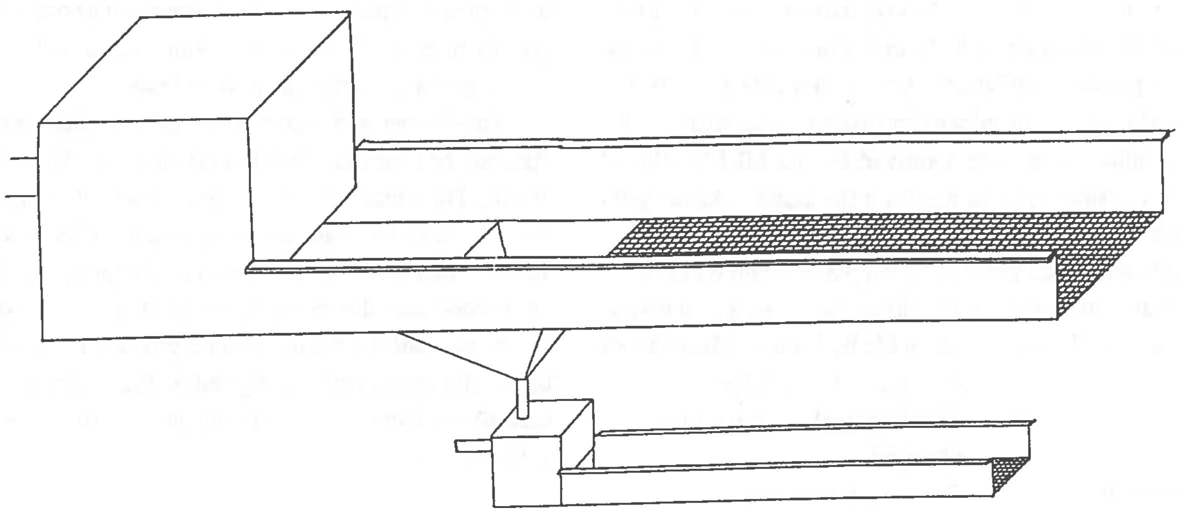

Figure 2. Short, narrow clean-up sluice positioned below coarse sand sluice.

Review of a Laboratory Investigation of Riffle and Punch Plate Performance[1]

Owen Peer, Research Assistant, KPMA Gold Recovery Project, Whitehorse, Y.T.

Introduction

The sluicebox is still the most popular placer gold recovery device in the Yukon because of its simplicity, reliability, low cost and very high concentration ratio. A sluicebox is a rectangular flume containing riffles on matting, through which a dilute slurry of water and alluvial gravel flows.

Two field sampling programs sponsored by the Klondike Placer Miners Association (Clarkson, 1989, 1990) provided valuable data regarding the gold recovery efficiency of a variety of sluiceboxes. However, they also posed many questions which could not be answered without resorting to testing under more controlled pilot-scale conditions.

A pilot scale testing facility was constructed at the Yukon College in Whitehorse. It used a gravel pump and cyclone to continuously cycle -1/2 inch placer gravels through an 8 feet by 6 inch wide sluice run. The sluice run was constructed with Plexiglas sides to allow visual interpretation. Several sizes, types, spacings and orientations of riffles were tested under a variety of feed rates, water rates and sluice run slopes to determine the optimal scour and deposition patterns.

Once the optimum conditions had been observed, the feed was salted with irradiated gold particles to confirm the riffle's effectiveness. In addition, the effects of Monsanto matting, suspended punch plate and the screening efficiency of stationary punch plate were also investigated.

Researchers disagree on the exact mechanism of gold recovery in a sluicebox and have related it strictly to settling velocity (Peterson, 1984), or to turbulence "required to keep riffles loose enough to trap gold particles" (Peterson, 1986). MacDonald attributed gold recovery to a combination of settling velocity and turbulence "such that the gold particles can sink to the bottom and not be disturbed by eddies having greater components of velocity in the vertical plane than the settling velocities of gold" (MacDonald, 1983).

Poling and Hamilton (1986) observed riffle action through the Plexiglas sides of a pilot scale sluicebox. They recombined and sluiced the same batch of Sulphur Creek gravels and placer gold particles several times to evaluate the effect of operation parameters such as feed and water rates, sluice run slope and screening on gold recovery. Poling stated that "turbulent eddies are formed in the slurry as it flows over and around the flow obstructions that comprise the riffles," and that "the interaction of these eddies with the particulate material that tend to collect around the riffles forms a dispersed shearing particle bed where particles of a high specific gravity are concentrated."

In 1988, Clarkson (1989) conducted a detailed tailings sampling program at six operating placer mines for the Klondike Placer Miners Association. His tests confirmed Poling's recommended gravel and water feed rates for expanded metal riffles and indicated that angle iron riffles were required to efficiently recover gold particles coarser than 1 mm. Clarkson also confirmed that sluiceboxes lose coarse gold particles and the presence or absence of one of these in a tailings sample can lead to high, unpredictable errors in conventional samples.

In 1989, Clarkson (1990) assessed the gold recovery efficiency of sluiceboxes at 11 operating mines with nuclear activated gold particles. These radiotracers were salted into the feed streams of the sluiceboxes and their recovery was related to the riffle design and operating parameters. He determined that:

a. in general, sluiceboxes operating at or below Poling's recommended feed rates had the highest gold recoveries, however angle iron riffles required higher slurry flows and/or steeper gradients than those recommended by Poling for expanded metal riffles;
b. water rates did not appear to affect gold recovery, provided that there was enough velocity (due to volume or gradient) to keep the riffles loose;
c. excessive scouring and high gold losses resulted when expanded metal riffles were warped above the matting;
d. doubled expanded metal riffles were much more sensitive to riffle packing than single expanded metal riffles;
e. flat bar riffles remained free from packing at very high feed rates but created excessive turbulence;
f. both cocoa and Monsanto matting appeared to be unable to retain fine -0.3 mm (-48 mesh) gold as

1. This is an abbreviated version of a final report available from New Era Engineering Corp., Whitehorse, Y.T., Canada.

effectively as Nomad matting; and

g. gold from the Yukon was most commonly between 0.3 and 1 mm in size, less than one percent of the gold was finer than 0.15 mm, and the gold size distribution and shape factors could change dramatically throughout any given placer deposit.

The authors have been unable to find any information regarding the spacing and orientations of riffles. MacDonald suggested that "trial and error as to their spacings will soon determine the optimum spacing and height for the particular material being processed." This trial and error process would appear to be a process destined for error, considering the extremely difficult logistics and large errors that Clarkson (1989) associated with sampling placer gravels.

Laboratory Procedure

The short-term testing facility was constructed at the Yukon College in Whitehorse. Six hundred liters of extremely clay-rich pay gravels and 600 liters of sandy pay gravels were collected and transported to the college where they were screened to -1/2 inch for the testwork.

All of the placer gravels and process water were continuously cycled through the 8 feet long by 6 inch wide sluice run with a trash pump. The 4 inch trash pump transported approximately 700 USgpm of slurry ranging from 6 to 22 percent solids by weight through 4 inch PVC pipe to a cyclone. The 10 inch polyurethane Krebbs cyclone dewatered the slurry and discharged solids from its underflow into a narrow hopper/slick plate area above the sluice run. A butterfly valve controlled the allocation of process water from the cyclone overflow to the sluice run. The excess water volumes bypassed directly to the pump box for make-up water (Figure 1).

One side of the sluice run was constructed of Plexiglas to allow observations of the hydraulic flow patterns. Three sizes (1, 2 and 3 inch) of angle iron, modified angle iron and flat bar riffles, and two sizes of expanded metal (10H and 4 lbs/ft^2) were obtained. These riffles, doubled sections of the expanded metal riffles, Nomad matting and Monsanto matting were observed in the sluice run under a variety of slopes, water flow and feed rate conditions. The corresponding feed and water rates were determined by timing the period required to fill a bucket under the sluice discharge.

Once the optimum scour conditions had been determined for various riffle designs, the feed was salted with irradiated gold particles to confirm the riffle's effectiveness. Scintillometers were used to determine the exact location of each irradiated particle in each riffle.

A full scale section of punch plate was constructed of Plexiglas. Two different lengths of punch plate were positioned at various heights above the riffles to test the corresponding screening efficiency and distribution of undersize gravels.

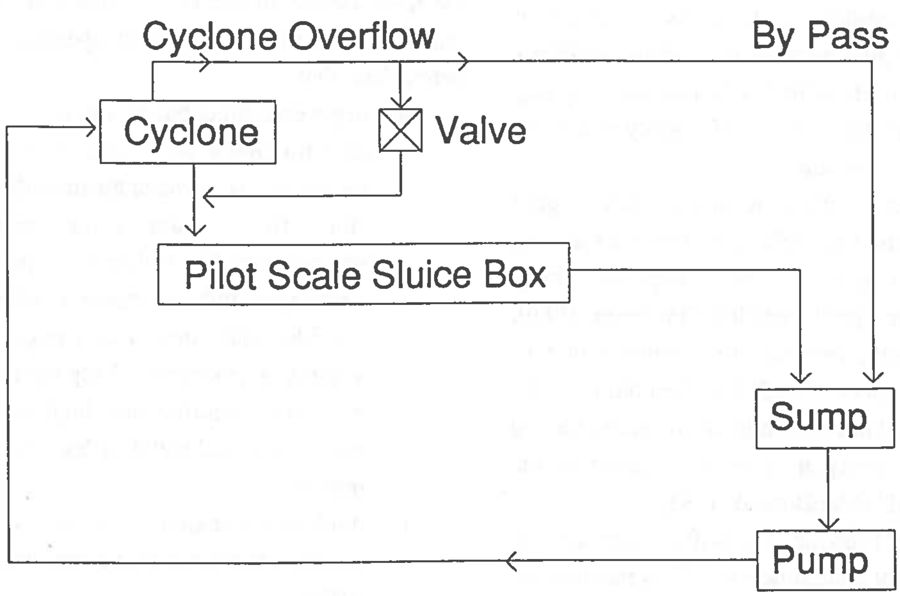

Figure 1. Pilot scale, closed-circuit sluice system.

Discussion

Sluiceboxes are actually centrifugal concentrators and settling velocity plays only a minor role in the gold recovery mechanism of a riffle. Gold's greater settling velocity allows a gold particle to descend to the bottom of the slurry column where it is preferentially cut into the streamline feeding a riffle's vortex (Figure 2). As the segregated slurry flow approaches the open space between the riffles it encounters a low pressure zone of separation which draws up to 0.25 inches of the slurry column down into the riffle. Under ideal conditions this distinct portion of slurry flow will be overturned as it flows down the rear of the following riffle and will continue flowing in a circular path to form a vortex.

The energy of this vortex is derived from the velocity of the slurry above the riffle and is slowly reduced due to friction as it flows down the back of the riffle, across the matting and up the live sorting crescent in its oval path. The gold contained in the streamline is driven by centrifugal force to the outside of the vortex. At the bottom of the vortex, centrifugal and gravitational forces combine to drive the gold particles into the matting.

If a gold particle cannot enter the matting it continues to a crescent of loose gravels which are continually being sorted by the reduced upward velocity of the vortex. Lighter weight particles continue flowing up and along the surface of this crescent and are ejected into the slurry flow above the vortex. Gold and heavier minerals which were not previously driven into the matting tend to remain near the bottom and inside of this sorting crescent.

When a sluicebox is shut down, the sorting crescent slumps into the area previously occupied by the vortex. This material is very well washed, loose and composed of heavier minerals. The volume under the riffle's horizontal lip, which is not occupied by the vortex and sorting crescent, is comprised of packed mineral particles which rarely contain gold. Gold particles are usually unable to penetrate into the packed solids under the riffle or a raised vortex because smaller heavy minerals fill the voids and harden the front of the solids.

The slurry velocity provides the energy which powers the vortex. If the velocity of the slurry is reduced through overloading with solids, insufficient water flow or shallow gradients, it may not sustain a vortex. If the riffles are too close, too tall, or if there is not enough energy available to the vortex, the vortex will not be formed properly. When the riffles are located too close, there is not a long enough contact between the slurry flow and the vortex to transfer the required energy. Under these conditions, the backside of the downstream riffle will begin to collect material and the bottom of the vortex will rise off the mat and may continue upwards until it disappears and the riffle is completely filled with material.

When the riffles are spaced too widely apart, the streamline which is drawn down into the riffle is not overturned and continues up and over the back of the next riffle. Under these conditions the space between the riffles fills up to form a shallow depression. Gold which is deposited in this depression is very sensitive to loss from scouring.

In a typical sluicing environment the maximum sized vortex which can be sustained is approximately one inch in diameter. If the riffles are taller than one inch, then the vortex will tend to rise up off the matting to be closer to the slurry flow, its power source. In tall riffles the vortex is extremely sensitive

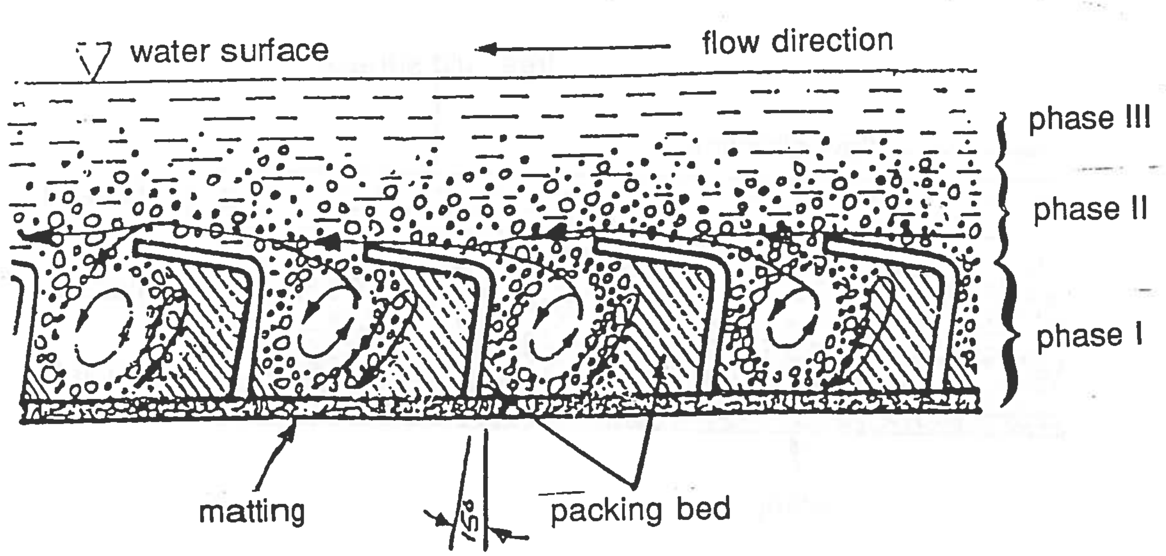

Figure 2. Hungarian riffle action.

to reductions in energy and will readily rise off the mat and pack the riffles with material.

Recommendations

Angle Iron Riffles

Clarkson (1989) recommended the use of angle iron riffles to retain gold particles coarser than 1 mm (14 mesh) and expanded metal riffles to retain gold finer than 1 mm. Angle iron riffles may require much steeper slopes (2.5 to 3 inches/ft) and can tolerate higher feed rates than expanded metal riffles.

Modified one inch angle iron riffles (top leg reduced from 1 to 1/2 inch in length) and ordinary one inch angle iron riffles were the most consistently efficient coarse riffles. The modified riffle has much smaller deposit of packed gravels and therefore higher proportion of clear matting because of its shorter top leg.

These riffles should have a two inch gap (3 inches on center line), be tilted at 15 degrees upstream (relative to the sluice run) in a sluice run with a slope of 3 inches/foot. The riffles have better performance at steeper slopes because the increased slurry velocity provides more energy for the vortex. The efficiency of the vertically aligned riffles is slightly lower.

Riffles with narrower spacings tend to fill up and isolate the gold concentrating vortex from the matting. Riffles with wider spacings form a shallow depression instead of vortices. Gold which is deposited in these depressions is very sensitive to loss from scouring. In riffles taller than one inch, the vortex is extremely sensitive to reductions in energy and will readily rise off the mat and pack the riffles with material.

Flat bar riffles are not recommended because they create excessive turbulence and reduce the vertical segregation of gold particles. The material rejected by a flat bar's vortex is launched up to the top of a turbulent slurry column instead of on to the next riffle. This severely reduces the opportunity for gravels and gold to enter the riffles.

Expanded Metal Riffles

Expanded metal riffles create vortices similar to those in the angle iron riffles but they cut a shorter height of the slurry column into their vortices (Figure 3). Due to its small size and shallow live sorting crescent, the expanded metal riffle is very sensitive to changes in slurry density, such as those caused by surging. Therefore, expanded metal riffles may require lower feed rates (8 loose cubic yards/hr per foot of sluice width) and shallower gradients than angle iron riffles. Expanded metal riffles must be kept tight to the matting to prevent high gold losses caused by excessive scour above the matting.

Doubled expanded metal riffles are not recommended because the bottom layer of expanded metal fills up and hardens with use. This prevents the gold particles from penetrating into the matting and makes the riffles even more sensitive to surging than single expanded metal riffles. When the doubled sections were separated with a 3/8 inch bar, the space eventually became clogged with gravels or they created hydraulic patterns which lowered recovery.

Matting

Monsanto matting is not recommended because its long needles protruded between the expanded metal riffles, disrupt-

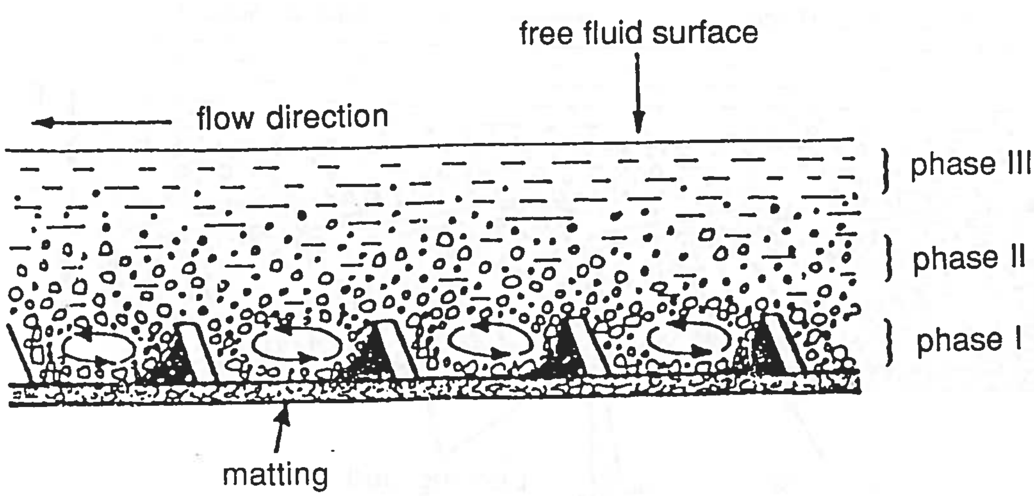

Figure 3. Expanded metal riffle action.

ing the formation of regular large vortices. The bottom 70 percent of the Monsanto matting tends to fill and pack hard, leaving only the tops of the needles to spin small irregular vortices. Clarkson (1990) indicated from radiotracer data that Monsanto matting appeared to be unable to retain fine (-48 mesh) gold particles effectively.

Punch Plate

A section of Plexiglas plate was drilled with 1/2 inch diameter holes, staggered at 1.5 inches on center line. The plate was elevated at 1 and at 2.5 inches above the riffles in the pilot sluice run. The punch plates were aligned at various slopes and elevations above the riffles and subjected to various slurry velocities. Table 1 displays the proportion of -1/2 inch gravels which passed through the eight and four feet sections of punch plate.

Stationary punch plate is not recommended because it is a very inefficient screen and it reduces the velocity of the slurry above the riffles. Its efficiency is even lower at steeper (3 in/ft) slopes and/or high slurry velocities. Sections of punch plate shorter than two feet are almost completely useless.

If the punch plate is too close to the riffles, the slurry velocity becomes too slow to power a vortex and the riffles will fill and pack. Riffles located below punch plate are much more sensitive to changes in slurry velocity and once filled (i.e. due to surging), take a long time to clear. Riffles which are located below punch plate are impossible to monitor for their effectiveness.

Oscillating Sluiceboxes

Even with these recommendations, pay gravels containing a high proportion of high specific gravity minerals, such as magnetite, or a high percentage of clay may be susceptible to riffle packing. Extreme gold losses occur when a sluice's riffles become packed because the gold is unable to get through to the matting. For these deposits, Clarkson (1990) suggests oscillating sluiceboxes as advisable alternatives.

Standard Recommendations

Pilot scale and field testwork (Clarkson, 1990) has indicated that sluicebox runs should be designed and operated at the following specifications for optimum recovery levels:

a. Pay gravels should be prescreened to at least -1 inch, washed thoroughly prior to sluicing, and feed rates should be controlled with mechanical feeders and/or manually operated wash monitors;

b. Every sluice run should have a sixteen foot long section of coarse expanded metal riffles (4-6 lbs/ft^2) which is wide enough to process 8 loose cubic yards/hr/ft with at least 160 Igpm of process water per foot of sluice width. The riffles must be tight against the Nomad matting to prevent scouring between the riffles and the matting;

c. Optimum slopes for the expanded metal riffles section will range from 1.5 to 2.5 inches/foot and should be set at a slope at which they do NOT pack and DO tend to deposit a crescent of heavy minerals and gold directly downstream of each individual riffle (loose gravels may partially fill the rest of the riffle);

d. The expanded metal section of the sluicebox should be followed or preceded by a narrower eight foot length of sluice run fitted with one inch angle iron riffles. At least 360 Igpm of slurry per foot of sluice width is required to operate the angle iron riffles. Try

Table 1. Punch Plate Screening Efficiency

Punch Plate			Above Punch Plate		Below Punch Plate	
Length (ft)	Slope (in/ft)	Elev (in)	Slurry Speed (ft/s)	Proportion of Solids	Slurry Speed (ft/s)	Proportion of Solids
8	2	1.0	12.7	37%	14.2	63%
8	3	1.0	17.5	45%	14.4	55%
8	2	2.5	11.5	21%	6.2	79%
8	3	2.5	18.1	67%	6.3	33%
4	2	1.0	16.0	69%	18.6	31%
4	3	1.0	17.5	86%	18.8	14%
4	2	2.5	14.6	64%	6.5	36%
4	3	2.5	15.5	87%	6.6	13%

to reduce or avoid rooster tails by gradually narrowing runs or by using baffles;

e. The one inch angle iron riffles should be aligned at 15 degrees from the vertical towards the top of the box, located with a gap of 2 inches between each riffle and mounted above Nomad matting;

f. The angle iron riffle section may have to be set at a steeper gradient of up to 3 inches/foot to avoid packing;

g. Riffles and matting must be easily removed so that more frequent cleanups (every 24 hours) will be performed (tracers which are not retained in matting will move down the sluice run, especially during start up periods); and

h. A section of slick plate should be placed in front of riffle sections to allow gold segregation in the slurry.

Once the equipment is in operation, periodic tests should be conducted to detect the extent and causes of gold losses.

Conclusions

Properly functioning riffles are actually centrifugal concentrators with settling velocity playing a minor role in gold recovery. Gold particles tend to segregate to the bottom of the slurry flow where they form a streamline that is diverted by a low pressure zone into a riffle. Under ideal conditions the rear of the downstream riffle overturns the streamline and it continues flowing in a circular path to form a vortex. At the bottom of the vortex, centrifugal and gravitational forces combine to drive the gold particles into the matting. Gold particles which are caught in the matting are very resistant to scouring losses.

If a gold particle cannot enter the matting, it tends to remain near the bottom of a crescent of loose particles. The reduced upward velocity of the vortex pushes lower density particles up and along the surface of this crescent and ejects them into the slurry flow above the vortex. Small gold particles which are deposited in the live sorting crescent may be washed away with excessive scouring. Gold particles retained by the longer and steeper sorting crescent of an angle iron riffle were less likely to be removed with scouring than those retained in expanded metal riffles.

When a sluicebox is shut down, the sorting crescent slumps into the area previously occupied by the vortex. This material is very well washed, loose and composed of heavier minerals. The volume under the riffle's horizontal lip which is not occupied by the vortex and sorting crescent is comprised of packed mineral particles which rarely contain gold. Gold particles are usually unable to penetrate into the packed solids under a riffle or under a raised vortex.

The slurry velocity (momentum) provides the energy which powers the vortex. If the velocity of the slurry is reduced through overloading with solids, insufficient water flow or shallow gradients, it may not sustain a vortex. If the riffles are too close, too tall, or if there is not enough energy available to the vortex, the vortex will not be formed properly and gold recovery efficiency will be reduced.

Slick plates are important in reducing turbulence and improving the vertical segregation of gold particles. Smaller gold particles and those located higher above the riffles in turbulent slurry flows will travel further before entrapment in a riffle.

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Evaluation of Yukon Sluicing Operations Using Radiotracer Gold: KPMA Gold Recovery Project Update[1]

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Introduction

The Yukon's numerous (185) placer gold mines are still a major social and economic force in this developing economy. Gold recovery at each of the various sluiceboxes is not optimized due to extremely difficult sampling logistics and a shortage of technical expertise and industrial research.

Sluiceboxes are very simple, reliable, inexpensive and yield very high concentration ratios, typically in excess of 10,000:1. This combination is very difficult to beat and explains why the sluicebox is still the only device used in the Yukon to recover placer gold.

Testing sluiceboxes with conventional sampling and evaluation techniques is very costly, time consuming, and problematic. Most placer gold ores are of very low value and contain a very limited number of distinct gold particles in a large volume of pay gravels. The effect of a single gold particle can cause large random errors (nugget effect). Standard errors are unpredictable and ranged from 7 to 50%, even when large sample volumes (2 to 7 cubic yards) consisting of hundreds of increments are processed, due to the frequent occurrence of coarse gold particles in the tailings from conventional sluiceboxes (Clarkson, 1989).

Nuclear tracer tests are more accurate (standard errors ranged from 2 to 5%), faster, cheaper, and safer than conventional sampling. In 1989, the recovery efficiency at several sluicing systems were determined by mixing gold tracers into the feed streams of 11 placer mines (G through Q) in the Yukon Territory. Four distinct sizes of radiotracers were used and their recovery was related to the design and operational characteristics of the individual sluiceboxes and their pay gravels.

Most of the sites which were tested in 1989 used bulldozer-fed, triple run sluiceboxes; only 3 of the 11 sites prescreened their pay gravels. The manually monitored sluiceboxes which were equipped with large distributors and sluice gates (H, I and L) had much higher screening efficiencies than the Ross Boxes (O and P). However, none of the distributors in the triple run boxes could match vibrating screening equipment for screening efficiency and consistent feed rates, important factors in efficient gold recovery. In many cases the poor screening efficiency of triple run boxes reduced gold recovery by underutilizing the side runs and overloading the center run with fine pay gravels, boulders and excess water volumes.

The final report, of which this paper is a summary, presents the existing and potential gold recoveries, and recommends sluicebox designs and operating parameters based on the results of the nuclear tracer test work in 1989 (sites G through Q) and the conventional sampling program of 1988 (Clarkson, 1989, sites A through F).

The Sluicebox

The sluicebox has been used in the Yukon since the Klondike Gold Rush and, with very few exceptions, is still the only device used for primary placer gold concentration. A sluicebox is a rectangular flume containing riffles on matting, through which a dilute slurry of water and alluvial gravel flows.

As the slurry passes over riffles, it forms a vortex which scours the placer gravels and drives the washed gold particles into the matting (Figure 1). The most popular sluice riffles include expanded metal, angle iron (Hungarian) and flat bar. Cocoa matting, or the more effective synthetic "Nomad" matting, is placed under the riffles to retain the gold particles. To remove the gold concentrates, the sluiceboxes are shut down and the riffles and matting are taken apart and cleaned.

Sluiceboxes are very simple, reliable, inexpensive and yield very high concentration ratios, typically in excess of 10,000:1. This combination is very difficult to beat and explains why the sluice is still very popular.

Triple Run Sluiceboxes

Several larger Yukon placer operations use triple run sluiceboxes consisting of a slick plate, dump box recovery area, distributor, center run, undercurrent run and side runs (Figure 2). The slick plates are mixing areas where the pay gravels are washed with either stationary or manually operated water monitors. Manually operated monitors provide better washing and help control surges of pay gravels.

Dump boxes are sections of punch plate suspended above

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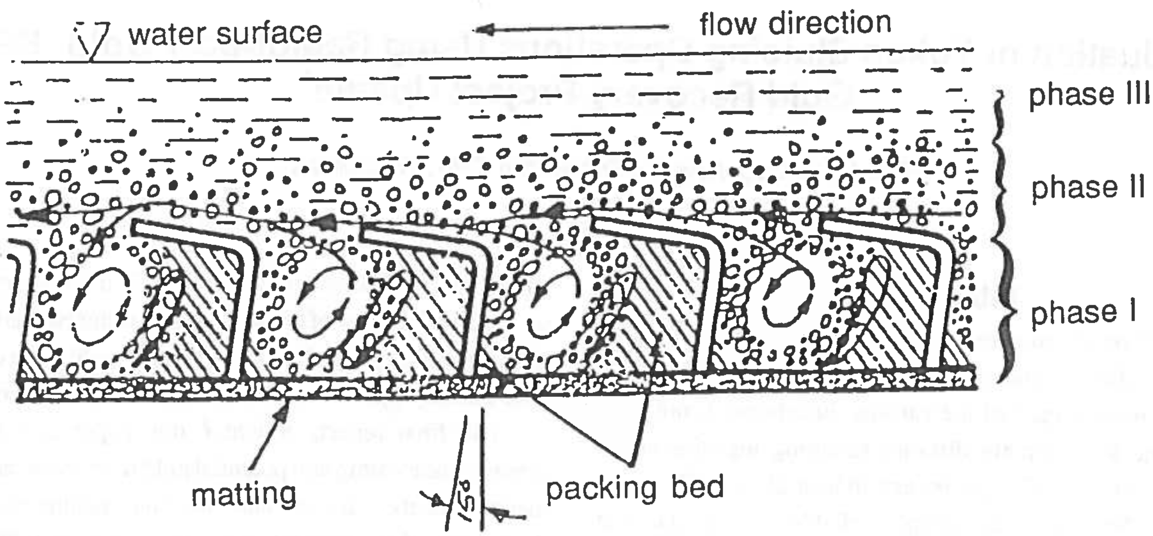

Figure 1. Hungarian riffle action.

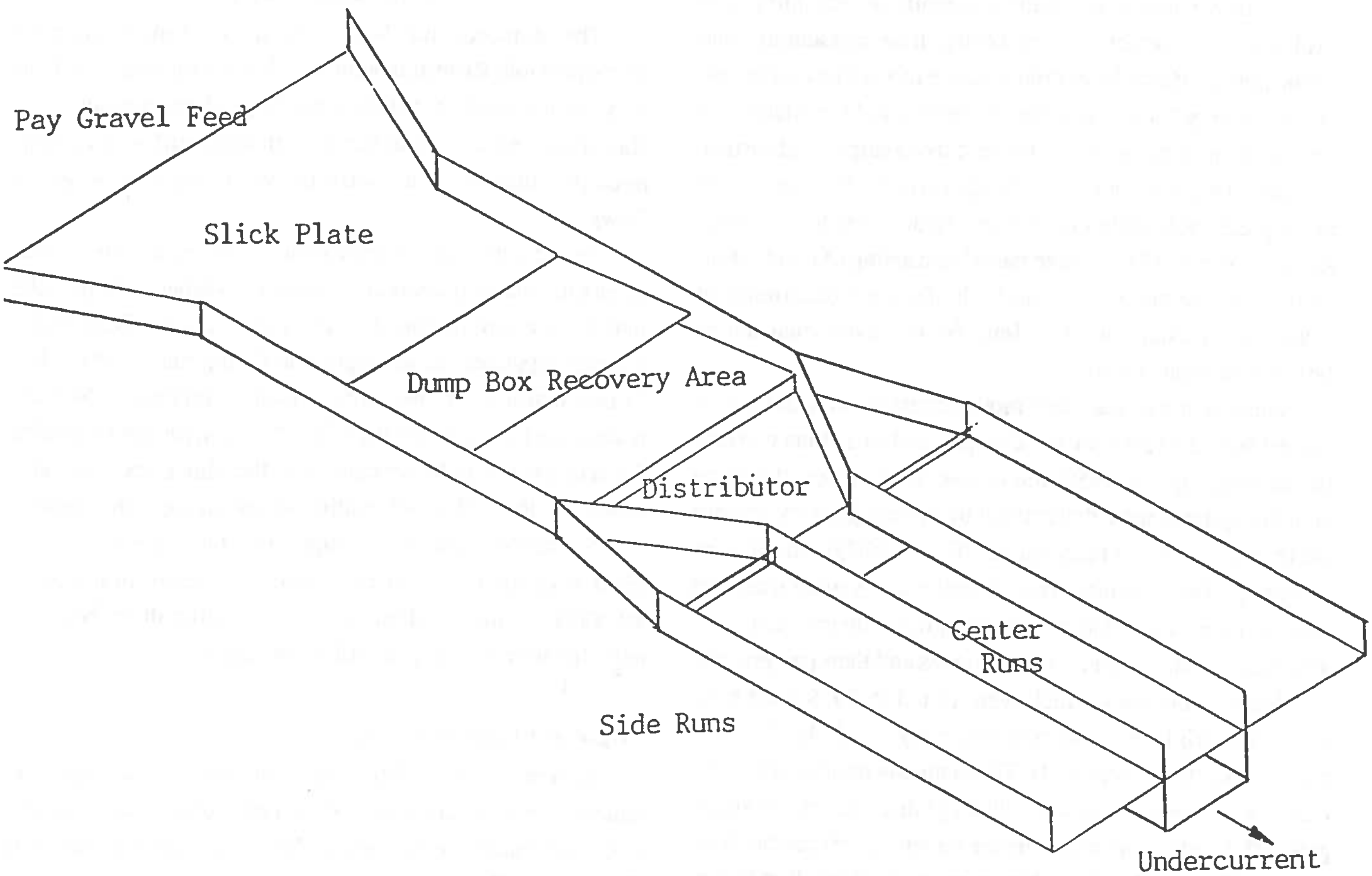

Figure 2. Typical three run sluicebox with small distributor, center undercurrent run and dump box recovery area.

a set of riffles and/or matting. Some of the fine pay gravels will pass through these punch plate holes and have their free gold collected by these riffles. The heavy sections of punch plate above the riffles in a dump box must be raised before its riffles and matting can be cleaned.

The distributor is a crude screening device which uses stationary punch plate to screen out fine pay gravels and direct them on to an undercurrent or set of side runs. Most of the water entering the distributor must stay on top of the distributor so that large rocks and boulders will be pushed along. Fine pay gravels and gold are inevitably trapped in these turbulent, excessive water flows (300% to 600% of the water required for optimum sluicing) and continue with the boulders at high speed down the center run.

Distributors are often too small (less than 100 ft²) and are fitted with punch plates which contain too few holes or have small hoes (less than 3/4"). The high water velocity of the pay gravel slurry above a distributor reduces its efficiency and limits the top size of gravels which can get through its holes. Many distributors are counter productive to gold recovery because they overload the center run with fine pay gravels and underutilize the undercurrent and/or side runs.

An undercurrent is a flume located directly below the center run; side runs are located on either side of the main sluice run. An undercurrent's location makes observation of its riffle action and adjustment of its slope very difficult. The center run must be completely dismantled before an undercurrent can be cleaned, therefore undercurrents are not cleaned as often as they should be.

The allocation of water to the various runs is easily controlled if sluice gates are installed at the distributor's undersize discharge. If the slope of the side runs is easily adjustable, then a more shallow sluice gradient can be utilized to improve the recovery of placer gold particles.

Prescreened Sluiceboxes

When placer gravels are not screened, additional water and steeper sluicebox gradients are required to move the boulders and coarse gravels down the sluice run. The high water velocity and extreme turbulence created by boulder movement causes gold migration and loss. Screens also improve washing by breaking up clumps of clay and cemented particles. Inadequate washing is a very common cause of gold losses.

Some of the Yukon's placer operations use either a stationary grizzly or a moving deck grizzly (Derocker) to eliminate large rocks from the sluicebox feed. Dry grizzlies are easier to operate than wet grizzlies but they reject rock with fine gravels and gold adhered to their surfaces. The Derocker is a well known and reliable screening device which does a good job of washing and rejecting coarse rock and boulders. It can be fed with a bulldozer provided wings are added to its entrance. However, the Derocker is only capable of screening to 2.5 inches. This is too coarse for optimum gold recoveries at most placer operations.

A few other operations also screen pay gravels with vibrating deck screens, rotating trommel screens and the "Supersluice" hydraulic grizzly. Vibrating screens can screen the largest volume of gravels for a given screen area and have low capital costs. Trommels are very good at washing the pay gravels, but are large and relatively inefficient screens. The long, even gradient of a trommel screen also requires higher feed ramps.

Considering the size of its deck (8 by 20 feet), the Supersluice has a low capacity (reported at 250 cubic yards/hr). The Supersluice uses steel fingers to move pay gravels over three decks with bars set at a one inch spacing.

Oscillating Sluiceboxes

Pay gravels containing a high proportion of high specific gravity minerals such as magnetite, or a high percentage of clay are susceptible to riffle packing. Extreme gold losses occur when a sluice's riffles become packed because the gold is unable to get through to the matting. For these deposits, oscillating or live-bottom sluiceboxes may be advisable alternatives.

An oscillating sluicebox consists of a pair of sluice runs suspended from a frame with cables. A direct current electric motor is mounted in between and above the sluice runs and rotates a weighted bent shaft through an angle drive. The motor-drive combination imparts a horizontal circular "panning" motion with a 5/8 inch diameter circle oscillated at 130 to 180 rpm. Oscillating sluiceboxes should not be used for pay gravels which don't pack riffles, because conventional sluiceboxes have a slightly higher recovery of fine (-0.30 mm, -48 mesh) gold with normal pay gravels.

Conventional Sampling

The magnitude and causes of gold losses from the Yukon's placer operations were not established previously due to the extreme difficulty of obtaining representative samples of the low grade placer gravels. In addition, accurate information regarding the operation and recovery efficiency of existing concentrators is usually not available. Gold losses were expected to be high due to the application of inappropriate recovery equipment; the inability to optimize equipment parameters such as screening, feed rate, and water addition; and the lack of technical expertise and industrial research.

Testing of sluiceboxes with conventional sampling and evaluation techniques is very costly, time consuming, and problematic. Most placer gold ores are of very low value and contain a very limited number of distinct gold particles in a large volume of pay gravels. The effect of a single coarse gold particle can cause large random errors (nugget effect). In addition, the uneven distribution of gold particles in a placer deposit often produces large random sampling errors.

The 1984 Placer Sampling Program collected pay gravel and tailings samples from four Yukon placer mines. Berry analyzed the content and size distribution of the gold particles. Several assay anomalies attributed to the "nugget effect" indicated that the conventional laboratory splitting and sampling procedures employed were not appropriate for placer gold.

Poling (1986) performed an excellent evaluation of a pilot scale sluice under controlled laboratory conditions. The same batch of Sulphur Creek gravels and placer gold were recombined and sluiced several times in order to assess the effect of operating variables. The study determined that 95% of placer gold as fine as 0.15 mm (100 mesh) should be recovered by following recommended optimum processing parameters.

The US $2.7 million Alaskan Placer Mining Demonstration Grant Project (Peterson) was beneficial, however, many of the tests were performed without controlled conditions, over short durations, and with incomplete or faulty sampling. Despite these limitations, Peterson indicated that the most cost-effective alternative for improving fine gold recovery over a crude sluicebox is to provide screening, a thorough washing of pay dirt, and even feeding to a well designed sluicebox. He determined that gold recovery devices added to the end of a sluicebox, or used to replace a sluicebox, including jigs, Reichert spirals, and centrifuges, are usually less cost-effective compared to this proven alternative.

In 1988, Clarkson (1989) conducted a detailed tailings sampling program at six operating placer mines for the Klondike Placer Miners Association. His tests confirmed Poling's recommended gravel and water feed rates and indicated that angle iron riffles were required to efficiently recover gold particles coarser than 1 mm.

Clarkson collected hundreds of tailings sample increments in duplicate from across the full width of each sluicebox discharge, over a two to four day mining period. The entire volumes (2 to 7 cubic yards each) were screened and processed on a shaking table to determine gold losses. Despite the large size, numerous increments, and extreme care taken in the design and implementation of the program, the standard errors ranged from a low of 8%, for sites with low losses, to 50%, for sites with high and/or coarse gold losses.

Sluiceboxes lose coarse gold particles and the presence or absence of one of these in a tailings sample can lead to high unpredictable errors in conventional samples. The collection of head samples is even more impractical than tailings samples due to the more frequent occurrence of coarse gold particles.

Testing sluiceboxes with conventional sampling is very costly, time consuming, and problematic. Some miners, geologists and engineers have tried to determine the relative recovery efficiency of a sluicebox with the following indicators which are erroneous and misleading:

a. PRESENCE OF FINE GOLD - The presence or absence of fine gold in a sluicebox is not a valid recovery test because even the crudest sluicebox will recover some proportion of the fine gold present in a placer deposit;

b. PRESENCE OF NUGGETS - The presence or absence of nuggets is not a valid recovery test because some of the coarse +1 mm gold particles are recovered in even the finest expanded metal riffles. However, site E (Clarkson, 1989) demonstrated that angle iron riffles are much more efficient at recovering coarse gold. Expanded metal mesh is more efficient at recovering gold finer than 1 mm, however, it can lose up to 70% of the gold coarser than 5 mm;

c. INITIAL CONCENTRATION - A high concentration of gold in the first few feet of sluice run is not a good indicator of recovery efficiency. Tracer tests revealed that sluiceboxes with overall recoveries of less than 30% still had most of the recovered gold in the first few feet of the sluice run;

d. TRIAL AND ERROR TESTS - False conclusions will result when the efficiency of sluicebox modifications are based on the quantity of gold recovered. This is due to the wide variations in the size distribution and quantities of gold present in different areas of a placer deposit;

e. GOLD PAN SAMPLES - A gold pan is a very small sample and prone to the "nugget" or coarse gold particle effect. Tailings piles are particularly difficult to sample due to gold segregation;

f. COMMON USAGE - Conventional sampling and radiotracer technology have indicated that many popular sluicebox designs and operating procedures are very wasteful. Often the long term survival of gold recovery devices has very little to do with their recovery efficiency;

g. LONG TERM SURVIVAL - The long term survival of a placer gold mine is dependent on many factors.

Operators with high grade gold deposits will survive even if they employ poor recovery and mining practices;

h. YOU CAN'T GET IT ALL - It is generally considered impossible to recover all of the gold in a placer deposit, however, that does not mean that an operator should be content with the amount of gold he is currently losing. Minor modifications doubled the overall recovery at site H and increased its profitability dramatically.

Nuclear Tracers

The high cost and unpredictable error levels in conventional testing led to the implementation of the 1989 radiotracer testing program. By 1949, nuclear tracers had been used extensively in mineral processing research. However, as late as 1985, Walsh was one of the first to use them to evaluate the efficiency of gravity concentration devices. Most heavy minerals (except placer gold and diamonds) can be accurately sampled and assayed with conventional techniques.

When placer gold particles are placed in a nuclear reactor, some of their nucleus' absorb an extra neutron and form gold's radioactive isotope (Au198) which can be used as a tracer. If these gold tracers have sufficient radioactivity and are relatively close, scintillometers can be used to identify and isolate the tracers. The number of tracers which are salted into the feed stream of a sluicebox must be large enough to produce test results with low error levels and yet be small enough to limit the time required to separate them from a concentrate.

The standard errors from these radiotracer tests are best represented by a binomial distribution. The maximum standard error occurs if the recovery is near 50%. For 100 tracers, the maximum standard error is 5%. The overall recovery estimates will usually be within one standard error of the true value (14 times out of 20) and almost always with two standard errors (19 times out of 20). To further reduce this maximum standard error, a much higher number of tracers would be required (i.e. for a standard error of 1%, 2500 tracers are required).

Nuclear tracers have increased the scope and safety for the field testing of sluiceboxes while reducing errors, costs and evaluation times dramatically. Each conventional tailings sampling test performed in 1988 cost the same as five radiotracer tests in 1989. When tracers are used, it is not necessary to take continuous tailings samples from the sluicebox's discharge while dodging boulders and heavy equipment. The gold tracers need only be irradiated to mild levels and exposure to them can be reduced with distance and protective aprons. Standard hygiene and handling procedures can eliminate the possibility of ingesting a tracer. Pocket dosimeters worn by Walsh and Clarkson to measure personal exposure to radiation did not detect dosages greater than normal background levels.

Conventional samples take a lot of time, money and effort to upgrade to raw gold. Every time they are upgraded, additional errors are introduced due to the inefficiency of recovery equipment. Significant losses are often discovered several months after testing, when it is too late for modifications and more tests in the same season.

With nuclear tracers, no assaying or upgrading is required, tests can be completed in 48 hours and this allows sluiceboxes to be modified and retested in the same week. Tracers can be used outdoors and in dirty gold rooms without introducing errors or worrying about tampering because the tracers can be readily identified with a scintillometer and are available only to licensed agents.

Testing Procedure

Representative placer gold particles were obtained from the various operator's concentrates and sieved into four distinct size fractions including: 1.4 mm (-10+14 mesh); 0.72 mm (-20+28 mesh); 0.36 mm (-35+48 mesh); and 0.18 mm (-65+100 mesh). The 0.18 mm was the finest placer gold commonly encountered in the Klondike. The 1.4 mm mesh gold was often coarse enough to demonstrate the requirement for angle iron riffles. These particles were packaged and sent out for irradiation at a nuclear reactor.

At each operation the irradiated gold particles were thoroughly mixed with moistened pay gravels. This mixture was split into identical volumes and added to the top section (slick plate) of the various sluiceboxes. At every site the material flow rates, water flows, processing parameters, equipment dimensions and operational characteristics were measured.

Once the operator had completed sluicing, a hand-held scintillometer was used to note the location of gold tracers in the sluice runs. Then the sluicebox was cleaned up and was checked for any remaining tracers. After the operator upgraded the concentrates, these tailings would be checked for tracers. The final concentrates were sieved and weighed. The gold tracers were removed from the concentrate and counted to determine the recovery efficiency of the sluicebox. The expired tracers were stored in a lead lined container until their radioactivity was near background levels (about 2 months).

These recovery data, in combination with weight and sieve data from the sluicebox concentrates, allowed the determination of the gold distribution curves for the head, concentrate, and tailings.. The metallurgical efficiency and operating parameters of the various sluiceboxes were compared with the recommendations. At two sites (G and H), the sluice runs were modified and retested to confirm the recovery improvements.

Tables 1, 2, and 3 present relevant information concerning the processing equipment, sluicebox design, and sluicing parameters at each of the mine sites tested.

Conclusions

Overall losses ranged between 71 and 0%, or from $2.5 million (site H) to less than $1000 (site J) per 1200 hour season and averaged $385,000. These losses could be reduced to a maximum of 11% (site H) and an average of 4% by screening the pay gravels to -1 inch and modifying the sluiceboxes according to section 3's recommendations. This would result in an average increase in revenue of $306,000 per site (Figure 3). In many cases, screening systems would also lower operating costs by reducing labor, water pumping and/or heavy equipment requirements.

Site H and P had the highest gold losses due to inappropriate riffle and matting design. During the testing program the overall recovery at site H was increased from 29% to 62% (Figure 4) by replacing its doubled expanded metal riffles and cocoa matting with single expanded metal riffles and Nomad matting. These minor modifications resulted in additional revenues of $1.2 million per sluicebox/season.

The sluiceboxes with screened feed (G, J, and K) had the highest recoveries, but even these systems required one inch

Table 1. Processing Equipment at the Various Mine Sites.

FEEDING	Site	G	H	I	J	K	L	M	N	O	P	Q
Bulldozer			Yes	Yes	Yes		Yes			Yes	Yes	Yes
Wheeled Loader/Backhoe						Yes	Yes	Yes	Yes			
Hopper/Feeder		Yes				Yes						
GRIZZLY	**Site**	**G**	**H**	**I**	**J**	**K**	**L**	**M**	**N**	**O**	**P**	**Q**
Length ft		No	No	No	No	13	No	?	?	No	No	No
Width ft						8		?	?			
Area ft2						100		?	?			
Spacing in						6		6	2.5			
SCREEN	**Site**	**G**	**H**	**I**	**J**	**K**	**L**	**M**	**N**	**O**	**P**	**Q**
Type		VB	PP	PP	VB	VB	PP	PP	DR	PP	PP	PP
Manual Monitor			Yes	Yes	Yes		Yes					
No of Screen Decks		1			2	1						
Length ft		8	12	10	12	12	14	12		6	6	8
Width (Dia) ft		8	14	14	7	5	15	8		12	10	7
Area ft2		63	133	105	83	57	96	100		49	43	41
Final Opening in		0.75	0.50	0.50	0.39	0.75	0.50	0.50	2.50	0.50	0.50	0.75
Efficiency*		100%	79%	75%	100%	100%	64%	34%		39%	14%	69%

Legend: VB - vibrating screen deck; DR - Derocker grizzly;
PP - stationary punch plate in a triple run's distributor.

Notes: * Efficiency refers to the proportion of -1/4 inch gravels which reports to the underflow of a screen. The manually monitored triple run boxes which were equipped with large distributors (100 ft2) and sluice gates (sites H, I and L) had much higher screening efficiencies than the Ross Boxes at sites O and P. None of the distributors in the triple run boxes could match vibrating screening equipment for screening efficiency and consistent feed rates and these are important factors promoting gold recovery. In many cases the poor screening efficiency of triple run boxes reduced gold recovery because they underutilized the side runs and overloaded the center run with fine pay gravels, boulders and excessive water volumes.

Table 2. Sluicing Equipment at the Various Mine Sites.

	G	H	I	J	K	L	M	N	O	P	Q
OVERALL	G	H	I	J	K	L	M	N	O	P	Q
Total Length ft	20	20	20	59	17	18	28	17	33	25	32
Total Area ft2	120	219	222	1013	276	209	293	100	474	397	395
Matting	NM	CO	NM	NM	UN	NM	NM	CO	MO	MO	NM
Concentrate Ratio	7	22	23	4	35	29	8	?	48	?	37
DUMP BOX RECOVERY	G	H	I	J	K	L	M	N	O	P	Q
Riffle Type	No	No	No	AN	No	No	No	No	MO	MO	EX
Length ft				39					12	5	12
Width ft				4					12	10	8
Slope in/ft				1.6					2.3	2.8	1.8
DISTRIBUTOR	G	H	I	J	K	L	M	N	O	P	Q
Length ft	No	12	10	No	No	No	12	6	6	6	8
Area ft2		133	105				8	17	49	43	41
Slope in/ft		3.7	2.8				2.9	2.5	2.5	2.8	2.6
SIDE/MAIN RUN	G	H	I	J	K	L	M	N	O	P	Q
Riffle Type	EX	DE	EX	EX	EX	DE	DE	No	MO	EX/MO	EX
Length ft	8	20	20	20	18	18	16		21	20	20
Total Width ft	20	7	7	44	8	8	6		12	10	12
Slope in/ft	1.5	2.6	1.9	1.7	1.5	2.0	2.9		2.7	2.8	1.2
CENTER RUN	G	H	I	J	K	L	M	N	O	P	Q
Riffle Type	No	No	No	No	EX/AN	No	AN	FB/AN	FB	FB	AN/EX
Length ft					16		16	17	22	20	21
Total Width ft					4		3	3	4	4	3
UNDERCURRENT	G	H	I	J	K	L	M	N	O	P	Q
Riffle Type	No	DE	EX	No	EX	DE	EX	EX/AN	MO	MO	No
Length ft		20	20		16	4	16	17	22	20	
Total Width ft		4	4		4	4	3	3	4	4	
Slope in/ft		3.5	2.8		1.5	3.4	2.9	2.5	2.7	2.9	

Legend: SR - single run sluicebox; DR - double with/undercurrent;
TR - triple run sluicebox; OS - oscillating sluicebox;

EX - expanded metal riffles; DE - doubled expanded metal;
AN - angle iron riffles; FB - flat bar riffles;

CO - cocoa matting; NM - backed Nomad matting;
UN - unbacked Nomad matting; MO - Monsanto matting.

Notes: Sites O and P used only the coarse needled Monsanto matting in their side and undercurrent runs and used flat bar riffles in the center run. Site K screened the feed to their triple run Ross Box with a vibrating screen deck and used unbacked Nomad matting.

angle iron riffles to improve coarse (+1 mm) gold recovery. For example, site G's overall recovery efficiency was improved from 96 to 99% when the lower section of its sluice runs were narrowed to half the original width and one inch angle iron riffles were installed.

Virtually all of the sites without screening (H, I, L, M, O, P, and Q) will pay back the capital investment in screening equipment in less than one season. Many others should have additional revenue in the first season of operation. Site Q is committed to install a Supersluice diesel/hydraulic operated screening system capable of being fed with a D9 bulldozer. For a capital cost of less than $100,000, season revenues are estimated to increase by $345,000. Other placer operations are planning to implement similar improvements.

Table 3. Processing Parameters for the Various Mine Sites.

FEED RATE Site	G	H	I	J	K	L	M	N	O	P	Q
PAY GRAVELS											
Total Lyd3/hr	40	240	240	40	225	142	132	70	250	100	250
Total % of Optimum	69%	158%	177%		141%	104%	89%	101%	134%	57%	104%
Is Feed Surging?	No	No	No	No	No	No	Yes	No	Yes	Yes	Mod
Side Runs Lyd3/hr	40	109	121	26	110	45	29	No	68	9	87
Side Runs % Optimu	69%	189%	203%	7%	174%	72%	65%	No	70%	7%	34%
Center run Lyd3/hr	No			No	62		89	?	146	88	163
Center Run % Optimum					317%		389%	?	450%	291%	675%
Undercur Lyd3/hr		131	119	No	14	96	14	?	37	4	No
Undercur % Optimum		446%	394%		43%	305%	59%	?	115%	13%	

PROCESS WATER	G	H	I	J	K	L	M	N	O	P	Q
Total Igpm	1250	5000	4000	2834	4923	4500	2656	1500	6500	4000	5000
Total % Opt Width	129%	283%	222%		194%	236%	151%	162%	249%	183%	214%
Side Runs Igpm	1250	1541	1593	2834	2849	1696	619	No	2868	1358	2368
Side Speed f/s	4	7	7	4	7	8	6	No	6	7	6
Side % Opt Width	129%	131%	133%	40%	224%	134%	81%	No	149%	86%	124%
Center run Igpm	No			No	653		1468	875	3632	2642	2632
Center Speed f/s					5		10	7	11	14	13
Center % Opt Width					103%		294%	189%	528%	438%	627%
Undercur Igpm		3459	2407	No	1420	2804	569	625	996	663	No
Undercur Speed f/s		12	12		7	17	6	7	6	7	
Undercur % Opt Width		588%	399%		224%	441%	114%	135%	145%	110%	

Notes: -The short sections of punch plate which served the undercurrent runs in sites L, O, P, and Q were almost completely ineffective.

-The excessive water volumes (400 to 600%) and velocities (10 to 17 f/s) required to move boulders down the center run of a triple run sluicebox provide few opportunities for gold particles to work through the turbulent slurry flow and be retained by a riffle. As coarse rocks pass over the riffles they scour the riffles and often become wedged between the riffles and disrupt proper riffle action.

-Triple run boxes which were washed with stationary water manifolds always experienced feed surges coinciding with the push cycle or loading cycle of the equipment which was feeding the sluicebox (sites M, O, P and Q).

The above tabulation of the pay gravel and process water flowrates are compared to those recommended by Poling (1985) and confirmed by Clarkson in 1988 and 1989 field sampling:

a) Feed rateof 8 loose cubic yards/hr/ft of sluice width; and

b) Water rate of160 Imperial gpm per foot of sluice width (for expanded metal riffles). Water rate can be reduced by increasing the slope of the sluice run or by decreasing the height of the riffles.

The total recovery for four gold size fractions was determined at each site, a better indicator of sluicebox performance than the overall recovery. Estimated recovery improvements were based on comparisons between screened operations from the 1989 nuclear tracer and the 1988 conventional sampling programs. There is a very limited amount of reliable data regarding the recovery of jigs and other gravity concentration devices, but they are unlikely to provide cost-effective alternatives to sluiceboxes for primary placer gold recovery.

Recommendations

The highest gold recoveries occurred at sites which screened their feed to -1 inch, used both expanded metal and angle iron riffles on top of Nomad matting for every sluice run and fed their runs near optimum feed and water rates. Expanded metal riffles are efficient at recovering placer gold particles finer than 1 mm,

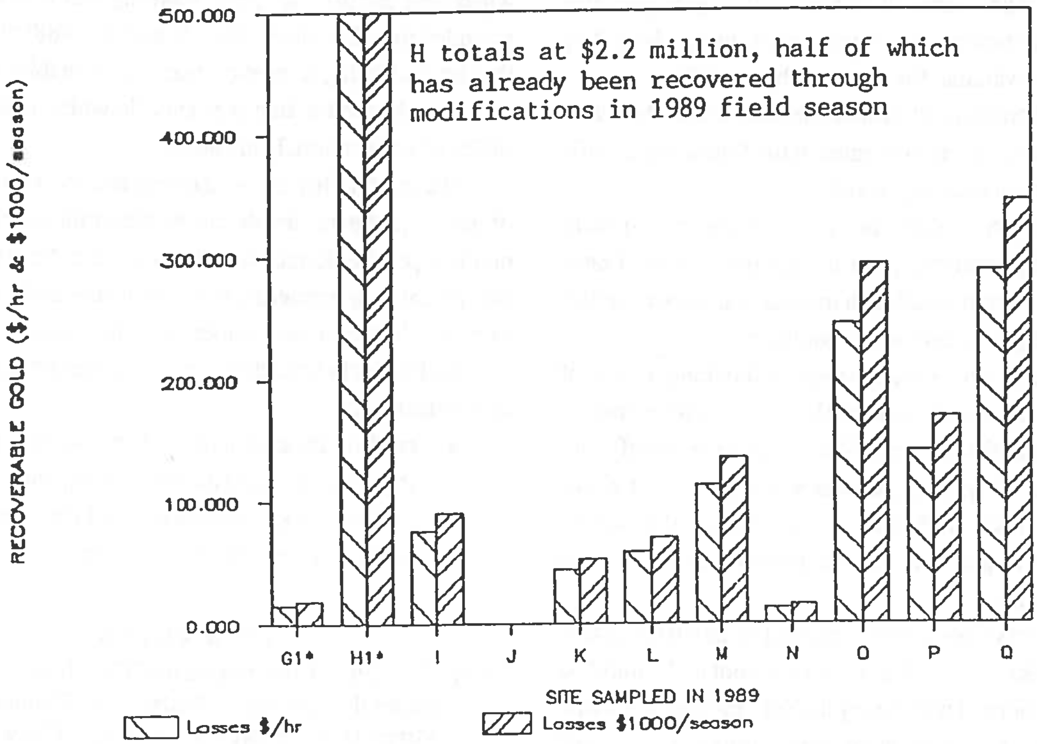

Figure 3. Value of recoverable gold losses

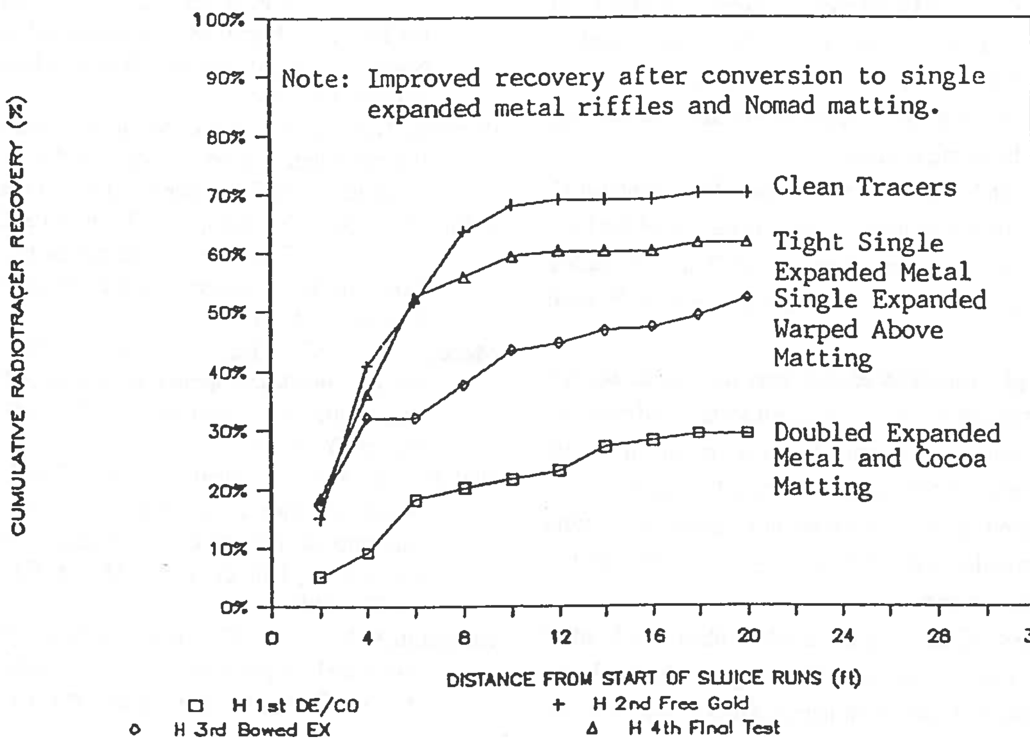

Figure 4. Cumulative tracer recovery vs distance from head of sluice at site H.

while angle iron riffles are more efficient at recovering those particles greater than 1 mm. Slick plates allow gold particles to segregate to the bottom of the pay gravel slurry where they are more readily available for recovery by the riffles.

Field and laboratory test work has indicated that sluicebox runs should be designed and operated at the following specifications for optimum recovery levels:

a. Pay gravels should be prescreened to at least -1 inch, washed thoroughly prior to sluicing, and feed rates should be controlled with mechanical feeders and/or manually operated wash monitors;
b. Every sluice run should have a 16 foot long section of coarse expanded metal riffles (4-6 lbs/ft^2) which is wide enough to process 8 loose cubic yards/hr/ft with at least 160 Igpm of process water per foot of sluice width. The riffles must be tight against the Nomad matting to prevent scouring between the riffles and the matting;
c. Optimum slopes for the expanded metal riffles section will range from 1.5 to 2.5 inches/foot and should be set at a slope at which they do NOT pack and DO tend to deposit a crescent of heavy minerals and gold directly downstream of each individual riffle (loose gravels may partially fill the rest of the riffle);
d. The expanded metal section of the sluicebox should be followed or preceded by a narrower 8 foot length of sluice run fitted with one inch angle iron riffles. At least 360 Igpm of slurry per foot of sluice width is required to operate the angle iron riffles. Try to reduce or avoid rooster tails by gradually narrowing runs or by using baffles;
e. The one inch angle iron riffles should be aligned at 15 degrees from the vertical towards the top of the box, located with a clear distance of 2 to 2.5 inches between each riffle and mounted above Nomad matting;
f. The angle iron riffle section may have to be set at a steeper gradient of up to 3 inches/foot to avoid packing;
g. Riffles and matting must be easily removed so that more frequent cleanups (every 24 hours) will be performed (tracers which are not retained in matting will move down the sluice run, especially during start up periods); and
h. A section of slick plate should be placed in front of riffle sections to allow gold segregation in the slurry.

Sites G, J and A (1988) demonstrated that a sluicebox can recover almost all of the placer gold in a Klondike deposit when feed control, adequate washing and fine screening are provided to a sluicebox. Sites G and B (1988) also illustrated that an oscillating sluicebox was a reasonably efficient gold recovery device for fine pay gravels which tend to pack the riffles of conventional sluiceboxes.

The washability of pay gravels and the size distributions of placer gold particles should be determined before deciding on the type of gold recovery equipment to be used. Once the equipment is in operation, periodic tests should be conducted to detect the extent and causes of gold losses.

Additional field testing of existing placer operations should be conducted to:

a. confirm the additional gold recoveries resulting from the recommended modifications; and
b. expand the knowledge of gold recovery at a greater variety of deposit types and recovery equipment.

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Gold Analyses - Myths, Frauds, and Truths[1]

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Abstract

"Unassayable gold and platinum group metals" have come into vogue in the 70s and 80s in certain jurisdictions as a means of perpetrating fraud. Usual arguments are that a particular ore is not amenable to "conventional fire assaying." Explanations for unassayable gold usually revolve around: evaporation of micron-size gold; vaporization of organic gold complexes; volatization of gold halides; alloying of gold with PGM's which prevents fusion or alloying which prevents collection.

This paper reviews several of the myths and truths of gold and PGM assaying with the knowledge that not a single mine operates in the free world producing gold from unassayable ore.

Introduction

Accurate analyses of exploration data and estimation of ore reserves are two of the most important functions of professional geologists and engineers in mine development. The problems encountered become particularly acute when dealing with precious metal values where very low grade deposits (in the range of 0.01 Troy oz per ton or ~0.2 ppm (g/t) by weight) sometimes are economic or constitute "ore." To emphasize this "scarcity effect," a gold assay of 0.4 g/t (0.01 T oz/st) would result from the presence of a single 420 μm (35 mesh) gold flake (which weighs 1 milligram) in a 5 kilogram siliceous sample.

The current high gold price and the prospects of high profit margins awaiting exploitation of even very low grade gold deposits, has spawned a sizeable group of incompetent or fraudulent gold assayers. These "assayers" will often report significant precious metal values where no such values occur. Failures of legitimate assay labs to confirm these "significant" assays are often attributed, by the charlatan, to their capability of detecting "unassayable gold" while the check assayer cannot. Geologists and engineers lacking detailed knowledge of sampling and assaying techniques are often hard-pressed to convince a gold-fever aroused company executive or investor-client to abandon a property on which attention-grabbing values have been reported, often by more than one laboratory. Professionals engaged in such evaluations must acquire an understanding of contemporary precious metal sampling and assaying techniques. Only then can they determine whether misrepresentation of values have been perpetrated.

The authors have independently investigated over 30 prospects and as many "assay" labs in which significant precious metal (gold, silver and platinum group elements) values have been reported when in fact near-zero values were present. We write this paper, with emphasis on fire assay techniques which are a well recognized analytical standard, or preconcentration technique, in the hope that the industry will learn to recognize and then ignore fraudulent assayers. At present, the prospects of seeing them out of business, or better yet behind bars, seem improbable.

This paper includes discussion of sample preparation, fire assay procedures, other noble metal analytical procedures and typical myths that have been used to justify fraudulent assay procedures to find "unassayable" gold.

Sampling of Gold Ores

Methods of Obtaining Samples

A wide variety of techniques have been used to obtain samples of gold ore. These include grab samples, chip samples, channel samples, panel samples, mine car samples, muckpile samples, core samples or sludge samples or cutting samples from diamond drills, rotary drills, percussion drills or reverse circulation drills, bulk samples taken by backhoes or bulldozers or by drilling and blasting. The application of these various sampling techniques to placer gold deposits is described in MacDonald (1983). Sampling techniques for lode gold deposits are described in McKinstry (1948), CIM Special Volume 9 (1968), Jones (1974) and Metz (1985). A comparatively recent innovation in hard-rock sampling technique is the use of a portable diamond-impregnated circular saw to cut the edges of channel samples so that the sample volume is more regular and uniform, Magri and McKenna (1986).

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2. Paper presented by G.W. Poling

Size of Samples

Relatively simple methods to calculate adequate sample size have been presented by Gy (1968, 1974, 1979), Clifton et al. (1969), Visman (1969) and Ingamells (1980). In sampling gold deposits containing relatively coarse gold, these techniques will all dictate that sometimes alarmingly large sample weights are required to obtain a representative sample, particularly of a low grade deposit. For example, a gold deposit containing 0.16 g/t (0.005 T oz/st) (250 mgm Au/m^3) with the coarsest gold being 1 mm diameter, would require a minimum sample size of 1.5 m^3 (or around 5000 lb) to achieve an accuracy of ±25 mgm/m^3 at a 95% confidence limit. A general conclusion of Clifton's work is that "the number of gold particles in the sample is the only factor controlling the precision of chemical analyses." A precision of ±50% is achieved at 95% certainty when samples for analyses each contain a minimum of 20 particles of gold (or ±20% @ 67% certainty). Although these relatively simple sampling theories depend on the gold particles being randomly distributed (which seldom occurs), they certainly provide very useful indications of adequate sample size. Implications of these sampling statistics have led at least one author to conclude that there are deficiencies in sampling placer gold deposits that are "perhaps insurmountable at reasonable cost," Fricker (1976). David (1977) gives a good review of the work of Gy and Ingamells.

Minimum Sample Size, Need for Pre-concentration

Clifton et al. (1969) nomographs, one of which is reproduced as Figure 1 in this paper, are based on simple statistics assuming randomly distributed gold. These nomographs are very useful in determining minimum sample weights and the possible need for physical preconcentration prior to analysis. Application of the Figure 1 nomograph should ensure that with 95% probability, the true gold content will be within ±50% of the gold content obtained by chemical or instrumental analysis of the sample. This degree of precision will be attained if the particular sample contains a minimum of 20 particles of gold. The original Clifton et al. paper provides additional conversion scales to provide sample sizes required for both higher and lower degrees of precision. Their paper also documents the high probability of certain samples mistakenly assaying zero gold content if fewer than five particles of gold are possible in any sample.

Table 1 tabulates the effect of gold particle size and the number of particles in a 4.5 kg (10 lb) sample and in other various assay sample sizes (weights). The implication is that small sample sizes or large gold particle size will invalidate an assay of small samples.

Table 2 tabulates the nugget effect for one particle of gold of a given size on an assay of a sample of given weight.

Preparation of Samples for Assay

Once a representative sample has been taken in the field, the sample must be prepared in the laboratory to produce a subsample that is both representative and of suitable size for assay.

Generally, samples weighing less than 10 kg will be processed as follows:

1. The entire sample is dried.
2. The sample is crushed to -6 mm.
3. The entire sample is riffle split to obtain one or more splits of approximately 250 g.
4. Each 250 g subsample is pulverized to produce an assay pulp of -150 µm.
5. The pulp can be rolled and sampled or better yet, obtained by using a micro riffle splitter.

Care must be taken to prevent cross-contamination of samples during crushing and pulverizing. When metallic gold is present, disc pulverization often leads to severe cross-contamination problems. Ring and puck pulverizers should then be used, Merks (1985).

Reference to Figure 1 (after Clifton et al.) will indicate that low grade samples containing relatively coarse gold cannot be represented by a pulverized subsample of a 250 g split. One partial solution is to add a "metallics assay." Following pulverization to -150 µm, the pulp is screened at -150 µm. Because gold is not readily pulverized, coarse gold will be retained on the -150 µm mesh screen. The entire material retained on this screen is weighed and assayed separately. Screen undersize is sampled and assayed separately. Total gold contained in the original sample is calculated from the combined weights and assays of the "metallics" and the screen fines.

Another procedure, designed to ensure that coarse gold, readily liberated gold and refractory — encapsulated gold are all included in assay results, is shown in Figure 2. This procedure, while costly, will not only give more accurate results but also indicates partial process requirements.

Fire Assaying for Gold Silver and Platinum Group Metals

Any modern review of the analytical chemistry of the noble metals must place particular emphasis on the fire assay. The fire assay is the most favored noble metal analytical technique both in history and at present. In the last decade or so, instrumental techniques have become more popular and, in

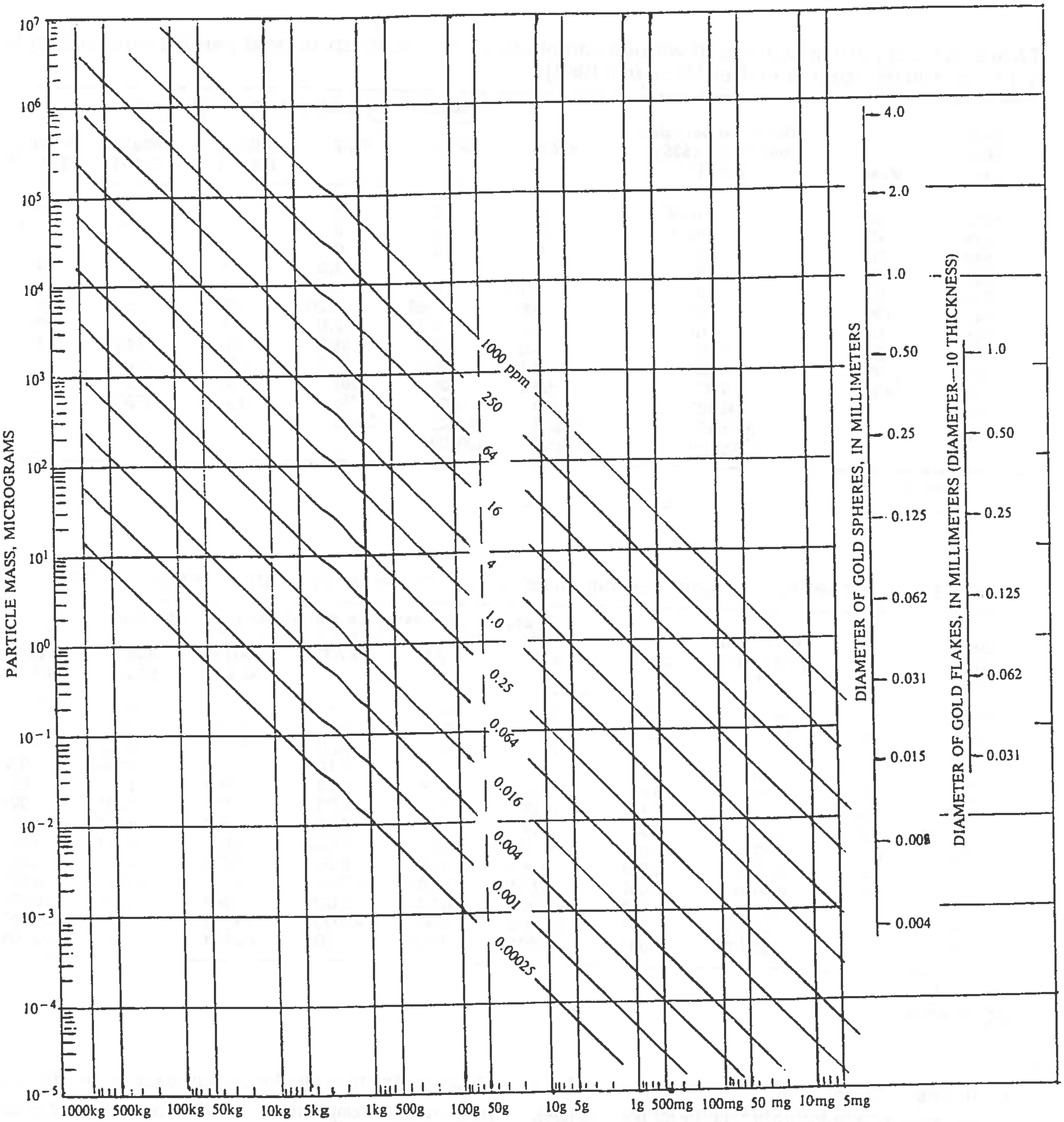

Figure 1. Size of sample required to contain an expected 20 particles of gold as a function of the combination of gold particle size and grade, assuming all gold particles to be of uniform size and randomly distributed in the deposit (after Clifton et al., 1969).

combination with the fire assay method, have supplanted the standard gravimetric finish-fire assay in many cases.

The classical lead crucible fusion assay is the most common and successful procedure for the concentration and collection of the noble metals.

One basic reason for the continued use of the fire assay technique is the relatively large sample size that can be treated by the technique, usually one assay ton or 29.1667 g. Most instrumental techniques do not allow sample sizes larger than a few grams, which are usually inadequate (Fig. 1). Another reason is that the fire assay technique is relatively forgiving and free of interferences.

The most common fire assay procedure for gold and silver analysis is:

TABLE 1. Gold particle content of various sample sizes as a function of gold particle size (cubes) for a 1.56 g/st (0.05 oz/st) ore (after McLean [1982])

Gold size μm	Mesh	No. of Au particles per (10 lb) 4535 g sample	Number of Au particles per assay sample 1 AT*	2 AT	5 AT	1000 g (33 AT)	2000 g (66 AT)	10 000 g (330 AT)
1650	10	0.17	0	0	0	0	0	0.
833	20	0.71	0	0	0	0	0	1.6
589	28	2	0	0	0	0	1	
295	48	16	0	0	0.5	3	6	35
208	65	46	0.30	0.60	1.5	10	20	101
147	100	128	0.84	1.68	4.20	28	56	281
104	150	370	2.4	4.8	12.0	81	163	368
74	200	1000	6.6	13.2	33.0	220	440	2193
45	325	4588	30.4	60.8	91.2	1011	2022	
38	400	7959	52.6	105	263	1736	3472	
20		49920	330	660	1650	10890	21780	
5		3276000	21671	43342	108355			
2		50000000	330000	660000				

* AT = Assay Ton

TABLE 2. "Nugget effect" of gold particle sizes versus sample weight (after McLean [1982])

Gold size μm	Mesh	Wt. of one gold particle (mg)	Change in gold assay per particle of gold · T.O. Ton 1/2 AT*	1 AT	2 AT	5 AT	1000 g 32 AT	2000 g 64 AT	10 000 g 320 AT
1650	10	88	176	88	44	17.6	1.75	1.38	0.27
833	20	11	22	11	5.5	2.2	0.34	0.17	0.03
589	28	4	8	4	2	0.80	0.215	0.062	0.021
295	48	0.50	1.0	0.50	0.25	0.10	0.016	0.008	0.002
208	65	0.17	0.34	0.17	0.18	0.03	0.005	0.002	0.001
147	100	0.061	0.12	0.06	0.03	0.01	0.002	0.001	0.001
104	150	0.021	0.04	0.02	0.01	0.004	0.001	0.001	0.001
74	200	0.0078	0.02	0.01	0.00	0.002	0.001	0.001	0.001
45	325	0.0017	0.004	0.002	0.001	0.001	0.001	0.001	0.001
8	400	0.00100	0.002	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
20		1.56E-4	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
5		2.41E-6	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
2		1.56E-7	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001

Note: E-6 = 10^{-6}
*AT = Assay Ton

1. mixing of the ore with the flux components plus a noble metal collector in a fireclay fusion pot (crucible);
2. fusion of the mixture;
3. pouring the melt into iron molds;
4. separating the slag from the lead or other collector metal;
5. scorification when required;
6. cupelling the lead-collector button;
7. recovering a precious metals bead, weighing;
8. parting of the bead with nitric acid; and
9. gravimetric determination of Au and Ag separately.

The purpose of the crucible fusion and the flux components is to eliminate the gangue minerals and to concentrate the noble metals into lead, drastically reducing any interfering elements. The flux must fuse to form a slag at a relatively low, easily reached temperature. The flux forms the slag by dissolution of the sample components *not* by melting. The flux is usually composed of all or parts of:

Chemical constituent	Chemical formula
Litharge	PbO
Flour	(household flour)
Borax	$Na_2B_4O_3 . 10H_2O$
Sodium carbonate	Na_2CO_3
Silica	SiO_2
Potassium nitrate	KNO_3

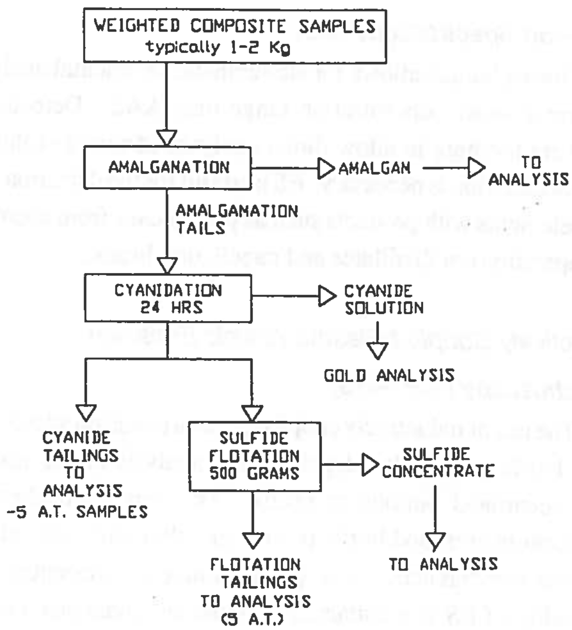

Figure 2. Composite gold extraction flowsheet (after McLean, 1982).

For most fire assays, a standard flux recipe is used. Under ideal conditions the flux is designed by the assayer after he knows the chemical and mineralogical composition of the sample. This can involve semiquantitative emission spectroscopy or inductively coupled plasma atomic emission spectroscopy and basic mineralogical analysis. An experienced assayer may simply use a small subsample, a low power microscope and an acid to perform his analysis. In the end he should determine if the sample to be assayed is neutral, reducing or oxidizing before he can design a successful flux for the assay of the sample. All sulphides and carbon are natural reducing agents. Many oxides are oxidizing agents.

Once the flux is selected it is placed in the fusion crucible with the pulverized sample and the charge is thoroughly mixed (a cover of borax or flux mixture is added to prevent dusting). The crucible is then placed in a fusion furnace at 1000°C to 1100°C until the fusion is complete (usually 30 to 60 minutes). After fusion, the crucible is removed from the furnace and the molten charge is poured into iron molds. During the fusion process a portion of the PbO is reduced to metallic lead which becomes the "collector phase." The high density molten lead and precious metals sink to the bottom of the clay crucible and to the bottom of the iron mold.

When the fusion charge has cooled and solidified, the lead button is broken away from the slag. Normally the slag is discarded unless it is reassayed to check for suspended or "shotted" precious metals. The lead button, weighing 20 g to 30 g, is next hammered into a cube to remove any adhering slag and to assist in handling with tongs during subsequent cupellation.

Throughout the crucible fusion portion of a fire assay, an experienced assayer watches carefully for unusual behavior, such as:

- boiling over of the molten slag;
- high viscosity on pouring;
- hang-up of lead on the crucible after pouring;
- shotting of the lead in the slag;
- matte formation on the lead;
- speiss formation;
- dirty lead;
- hard, brittle or non-ductile lead on hammering the cube;
- unusual texture to the lead;
- too much or too little lead in the button; and
- any unusual occurrence from pouring to hammering the lead cube.

Depending upon the severity of the problem the assayer may scorify (oxidizing fusion with an excess of granulated lead) the lead button or simply perform another fusion. A second fusion, with the knowledge learned from the first improper fusion, is the preferable route.

The next step in the classical fire assay is cupellation. At this stage a direct addition or "inquartation" of silver is sometimes made to facilitate possible subsequent acid-parting of the silver from the gold-silver bead (the inquartation is sometimes made in the fusion stage in high productivity-commercial assay labs). The purpose of cupellation is to extract the noble metals from the lead. This is accomplished by oxidizing the lead in a porous cupel made of MgO, bone ash or cement in a muffle furnace at 950°C to 1000°C. The oxidized lead (liquid litharge) is mainly absorbed into the cupel with approximately 2% of the lead being volatilized. As with the fusion stage, observations made during the cupellation procedure and of the resultant bead inform the assayer whether the cupellation has been successful. An example of difficulty is the occurrence of tellurides in a sample; this can lead to non-spherical multi-beads. These difficulties can be eliminated by repeat assays of preoxidized or roasted telluride ores. Cracks in the cupel and certain scoria indicate unsuccessful cupellation.

In the classical fire assay, the gold and silver contents of the Doré bead obtained from cupellation are determined gravimetrically. After carefully weighing (to ~.002 mg) the gold-silver Doré bead, the silver (and any base metal) contents are dissolved away in hot nitric acid (parting) and the remaining gold is reweighed. The silver content is obtained by the difference in weight. If additional silver is added as "inquartation," both the weight and purity of this silver must be assured in order to determine accurately the silver content of the sample. As the name implies, the optimum ratio of silver to gold during inquartation is approximately 4:1.

If present in a sample, platinum group elements (PGE) are also collected in the lead button during a crucible fusion. In some laboratories, nickel sulphide rather than lead is preferred as a fusion-collector for the platinum group elements, Dixon (1975). Upon cupellation, platinum, palladium and rhodium alloy with the gold-silver Doré bead and cause characteristic changes in its appearance. Nitric acid parting will dissolve most of the palladium, part of the platinum and none of the rhodium. Iridium does not alloy with the Doré while osmium and ruthenium tend to form volatile oxides which can be partially volatilized and lost during cupellation.

While procedures are available to complete analyses of gold-silver and the PGE using the classical fire assay and wet chemical analyses, these are now often superseded by instrumental techniques. For Au-Ag-Pt-Pd-Rh assays, standard crucible fusion and cupellation are often followed by digestion of the bead and use of atomic absorption or emission spectroscopy to finish the analyses. Sometimes the Doré bead is analyzed directly using neutron activation analysis. To include the other PGE's, the lead collection button is often digested and analyzed instrumentally.

Other Noble Metal Assay Methods

Several types of spectroscopy are utilized directly on solid samples with or without physical preconcentration of the precious metals. Pyrometallurgical preconcentration using the fire assay procedures of crucible fusion plus lead collection alone, or in combination with cupellation, are now commonly used in combination with instrumental analyses.

Atomic Absorbtion Spectroscopy (AAS)

This is the most common instrumental technique used for completing the assay of noble metals. It is often used after preconcentration by fusion and cupellation. The use of the solvent extractant methyl isobutyl ketone (MIBK) further to concentrate the noble metal has improved its sensitivity and reduced interferences. Sometimes dissolution and MIBK extraction are used directly on ore samples. Modern AAS employs flameless spectrographs, various buffers or releasing agents and chemical extractions that have greatly reduced the interferences that have plagued AAS in the past. It is most important as the finishing step in noble metals assaying.

Emission Spectroscopy (ES)

This technique allows for simultaneous elemental analysis over a wider concentration range than AAS. Detection limits are too high to allow direct analyses of ores and thus, preconcentration is necessary. ES is useful for the detection of trace elements with products such as precipitates from chemical separations or distillates and cupellation beads.

Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES)

The use of inductively coupled plasma emission spectroscopy has largely replaced gravimetric analysis of fire assay preconcentrated buttons or beads. The very high plasma temperatures reached in the plasma greatly reduce the inter-element interferences. The greater range of concentration capability of ES is maintained as is the simultaneous multi-element capability. Analysis of cupellation beads by ICP-AES allows assays in the ppb range. The high capital cost of ICP instruments (>$100,000) discourages widespread application of this technique.

Neutron Activation Spectroscopy (NAS)

The analytical chemistry of noble metals lends itself to NAS because of sensitivities that are at least two orders of magnitude more sensitive than other methods. However, extremely complex interferences do not allow such high sensitivities to be readily realized. Without preconcentration, the maximum sample size is approximately 2 g to 3 g and, thus, if there is inhomogeneity in the noble metal distribution, the sample size precludes accuracy. Analysis of cupellation beads allows noble metal assays in the ppb range.

X-ray Fluorescence Spectroscopy (XRF)

Separation, recovery, and/or preconcentration techniques are necessary because this method requires samples with at least 1 mg of the noble metal in the sample for accurate determination. This method is often used with fire assay preconcentration into a lead button followed by cupellation into a silver or gold bead. The bead is annealed, flattened and reannealed and analyzed as a solid. Portable XRF analyzers are sometimes used for analyzing gold distribution in slope faces or on drill cores or in drill holes, but this technique is semi-quantitative at best.

Myths

Complex Stable Compounds

"Certain ores which contain precious metal values are

not amenable to recovery or assay by standard methods. In our particular ore the standard lead collection fails to gather values. We believe the metals occur as complex and very stable compounds. These compounds dissolve and go into solution as the compounds. They likewise are not decomposed by the standard lead fusion assay but remain in the slag rather than gather in the lead."

This quote demonstrates a typical belief that "unassayable" gold is the result of complex stable compounds. This is, of course, incorrect, as the stability of compounds is not a consideration if the flux is correctly designed. If a gangue mineral does not dissolve, it becomes immediately apparent in the slag during pouring or removal of the solidified slag from the lead button. Complex stable compounds are easily dissolved when a suitable flux is incorporated in the fusion allowing the lead to collect any noble metals present.

Noble Metal Interferences

There is a widely held misconception among incompetent "assayers" that the presence of various noble metals causes interferences that negate the fire assay procedure. The following data are from the popular mining magazine *California Mining Journal* and was authored by Alvin C. Johnson, Jr., Ph.D. The article is titled, "Platinum Group Interference Phenomena in Fire Assaying Methods" and the data presented are shown in Table 3.

These data are from analyses of splits from an ore sample that is described as "precious element bearing and is from a refractory desert alluvial deposit that is located in southwestern Arizona. The deposit is quite extensive and does not respond well to standard fire assay methods or atomic absorption methods."

Johnson hypothesizes that "the precious elements are not initially reduced to low valence or metals prior to or during analysis and that they do not react chemically or metallurgically in a normally predictable fashion." Experience suggests to Dr. Johnson the existence of numerous refractory precious element compounds which have not, as yet, been identified.

Our analysis of the above data (Table 3) would indicate that at least three of the reported "assays" are not correct values. To report gold results that vary from trace, 0.001, 0.01 to 0.79 oz/st is unacceptable and indicates human incompetence and not errors in the method. The presence of platinum group metals does not affect the results of trace assay methods by competent assayers. Because the D method of Table 3 utilized a crucible fusion with the precious metals collected in a lead button, the probable errors would involve manipulation of the emission spectroscopic data.

TABLE 3. Suspect assays

	Assay method			
Element	A	B	C	D
	Assay oz/ton			
Ag	0.8	0.01	0.1	0.29
Au	trace	0.001	0.01	0.79
Pt				1.31
Pd				0.32
Ir				1.81
Os				1.53
Rh				1.31
Ru				2.07

A = fire assay, reductive flux
B = fire assay, "standard" flux
C = AAS after aqua regia leach
D = ES of fire assay lead button

Micrometer Gold

Micrometer-sized gold is often blamed for the inability of the classical fire assay to result in the correct assay value. This makes no sense since micrometer-sized gold particles are ideal from both a sampling and assaying perspective (Fig. 1). Truly micrometer-sized gold should result in the most reproducible assays from all assay techniques.

Sometimes the micrometer-sized nature of the gold is stated to "cause volatilization of the gold during fire assay." This is also nonsense as the fire assay fusion does not exceed 1100°C and the boiling point of gold is 2600°C.

Myths also abound that the micrometer-sized gold particles are not collected by the lead but stay suspended in the molten slag. Again, this is untrue. With micrometer-sized particles the collision statistics with the lead collector are favored and there should be even less chance of gold loss to the slag.

Volatilization of Gold Compounds

A common mistakenly-held theory is that unassayable gold occurs as a halide compound, the favorite of which is chloride. These incorrect theories postulate that the halide is volatile as a gold chloride. While there are two stable gold chloride compounds ($AuCl$ and $AuCl_3$) that can be made to volatilize and transport, this is only true in an atmosphere of chlorine. In a fire assay fusion, $AuCl$ and $AuCl_3$ would decompose to gold and chlorine gas at 170°C and 254°C, respectively. The gold would then collect in the lead as from any other gold source.

Similar incorrect theories abound for organo-complexes of gold and other low melting point compounds.

Corrected Assays

A common procedure used by incompetent assayers is the so called "corrected" assay technique. The "corrected assay" is a legitimate technique used to account for minor losses to the slag and cupel that inevitably occur. A legitimate corrected assay technique is sometimes used for an initial check of potential losses particularly when assaying high grade ores and bullion.

A legitimate corrected assay consists of a standard fire assay of the ore or bullion with fire reassays of the first fusion slag (refusion) and an assay of the cupel from the first assay. The total assay is then the total of the gold or noble metals from the three fire assays (first fusion, slag refusion and cupel fusion). For high grade ores or bullions this total of the three assays can be 2% to 3% higher than the first fire assay (Table 4, from Hosking [1982]).

The fraudulent assayer performs the "corrected" assay in the same way initially but continues the process by reassaying the second slag and second cupels. This may continue for as many a four times, thus having assays for three or four assays of slags and cupels for one sample. This type of "corrected" assay is easily recognized to be fraudulent if the amount of noble metal in any assay after the first is more than 5% of the noble metal recovered in the first assay. In some cases these "corrected" assays are claimed to recover 50% to 75% of the noble metal in the subsequent reassays of the slags and assays of the cupels. This is always fraudulent or incompetent and these assays should be ignored.

Other Fraudulent Assay Technologies

There are various other techniques used by fraudulent or incompetent "assayers" in perpetuating their businesses. While the names of the techniques they use have legitimacy, the techniques are either misapplied, overly complicated, useless, antiquated or simply not applicable to modern noble metal assaying.

Among the methods or techniques used to "assay" for "unassayable gold" that should alert the reader to possible fraudulent, or incompetent assays are:

- wet chemical analyses, while certainly legitimate, is not used for noble metal analysis of ore samples as a standard method;
- electrolysis methods;
- unusual time — temperature combinations during fire assay — they are unnecessary;
- basic or acidic pretreatment of ores prior to fire assay — especially if both the lixiviant and undissolved solids need to be assayed; and
- methods that prevent "volatilization" of noble metals.

TABLE 4. Example of gold losses in fire assay (after Hosking [1982])

Method of loss	% Gold loss
to slag	0.2
to crucible	0.2
to hammering button	0.1
to cupel	0.7
to parting solution	0.03-0.8
total loss	1.15-2.0
gain by silver retention	1.25-1.65

Evaluation of Laboratories

The best method to evaluate a laboratory is to consult with well recognized mining firms in the locale producing noble metals. These firms will have their own assays checked by outside commercial laboratories. These laboratories will be legitimate.

Secondly, submission of samples of the following nature to evaluate the laboratory is good practice:

- duplicate samples from another laboratory;
- blank samples; and
- known standard samples (USBM and CANMET).

Significant deviation from zero on the blank or from the standard value leads one to conclude the laboratory may not be of high calibre. Large deviation suggests incompetent or fraudulent assaying.

If the duplicate assays are different, then a third laboratory must be selected because either of the first two laboratories may be in error.

Conclusions

The pyrometallurgical techniques of fire assaying, whether used for the gravimetric analyses of total gold and silver values or for preconcentration prior to instrumental analyses, remain the most reliable and most respected procedure for analysis of precious metal values. The procedure is relatively expensive, hence, they are now often augmented by instrumental analyses for many routine high production-number assays.

Of all the possible errors in sample analyses, lack of representivity of the sample to be assayed often constitutes the greatest source of error. Nugget effects of coarse precious metal values necessitate strict attention to the adherence to using meaningful sample size or preconcentration techniques. Experienced competent assayers will find precious metals if present. The fire assay technique, while requiring competence and skill, is a relatively forgiving technique. The most respected expert in the field of fire assaying in this century, Professor F.E. Beamish, once stated "that during 40 years of research in the field, he had not experienced a single example of failure of this classical assay to find a paying ore." The authors of this paper have a collective experience with fire assay on the order of sixty years and unequivocally agree with Beamish! A major conclusion is that "there is no such thing as unassayable gold and silver."

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Mining and Recovery of Placer Minerals

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and
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IHC Holland

1. Introduction

The organization which I represent, IHC Holland, is a group of manufacturers specialized in the design and construction of all types of dredging equipment, including mineral dredgers and ore separation plants. The group has a record dating back to 1880, and has built

152 mineral dredgers
374 bucket dredgers
280 trailing suction hopper dredgers
680 cutter suction dredgers
47 barge unloading dredgers
680 barges for transporting dredged soils
79 rock breakers
172 tugs
639 ferries and cargo ships
22 self-elevating platforms
23 jig treatment plants
and numerous jig units.

With this record IHC fully qualifies as a major supplier to the mining and dredging industry.

2. Placer Mining

A multitude of mining methods are available to win the valuable minerals from alluvial deposits. When we talk about placer or alluvial deposits, we define these as 'potentially commercial deposit of detrital natural material containing valuable minerals in the form of discrete grains.' It is a successful coincidence of exposed mineralized catchment, vigorous drainage and the right conditions for 'selective accumulation' as a geologist aptly described.

A placer can be formed by weathering and/or transportation of mineral-bearing rock. The transport can be done by glaciers, wind, wave action, and rivers over long or short distances. The mineral bearing layers may be covered later by deposited layers of barren material. The degree of segregation can vary widely, as can the grain size of the valuable minerals; these in turn can be very different (cassiterite, gold, diamonds, heavy minerals such as rutile, magnetite, ilmenite etc.) and the size of the deposit may vary from thousands to many million of tons.

Thus the variety of placers has no limit, and (subject to the laws of economics) mankind has devised many different ways to extract the valuable minerals from their environment. Placer mining is one of the earliest and simplest forms of mining. All methods involved movement of relatively large quantities of material, the most used methods being dry earth moving, gravel pumping and dredging. Each has its own characteristics as to required technology, cost, and attainable throughputs. Often the characteristics of the alluvial deposit more or less determine the technical possibilities for handling the material.

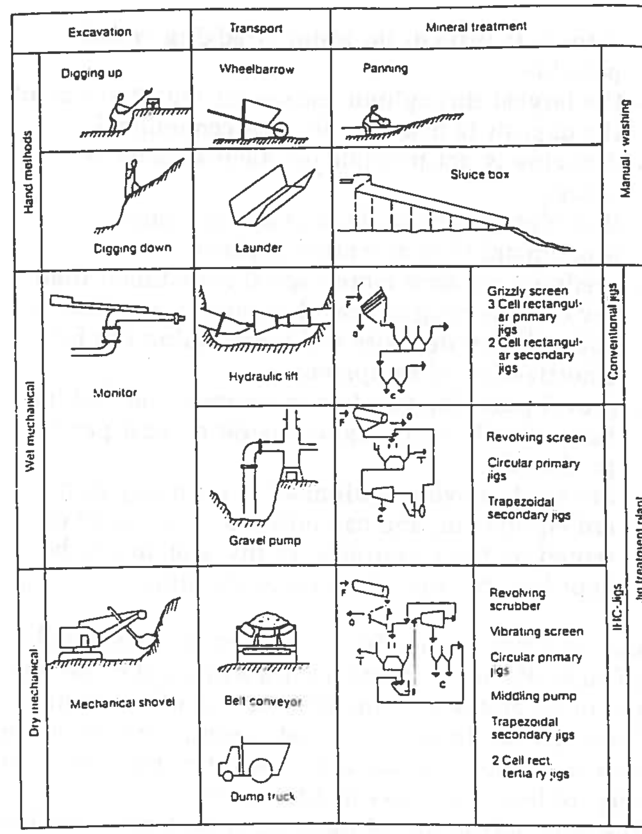


Figure 1. Open cast mining systems.

Figure 1 shows the different open cast mining systems in a simplified form. Most of the methods will be familiar to a practicing miner.

All but the small pick and shovel panning operations warrant a study at some depth. It is beyond the scope of this paper to cover all aspects of such a study; we only mention some of the obvious.

Mineral Specific Gravity	Mineral Name	Chemical Composition	% Valuable Constituent	Hardness Mohs Scale
16-19	Platinum	Pt(+Fe,Ir,Rh, etc)	Pt: 60-90	4.0-4.5
15-19	Gold	Au(+Ag,Cu,Fe)	Au: 50-95	2.5-3.0
10.5	Silver	Ag	Ag: 100	2.8
8.8	Copper	Cu	Cu: 100	2.8
7.5	Galena	PbS	Pb: 86.6	3.0
7.0-7.5	Wolframite	(Fe,Mn)WO_4	W : 60.6	4.5-5.5
6.8-7.1	Cassiterite	SnO_2	Sn: 78.6	6.0-7.0
5.9-6.1	Scheelite	$CaWO_4$	W : 63.9	4.5-5.0
5.5-7.5	Columbite/Tantalite	(Fe,Mn)O(Nb,Ta)$_2O_6$	Nb,Ta: variable	6.0-6.5
5.2	Magnetite	Fe_3O_4	Fe: 72.4	5.5-6.5
4.9-5.3	Hematite	Fe_2O_3	Fe: 70	5.5-6.5
4.9-5.3	Monazite	(Ce,La,etc)PO_4+ThO_2	R.E.+ThO_2:70	4.5
4.5	Xenotime	Y PO_4	Y : 43.3	4.5
4.5	Barite	Ba SO_4	BaO:65.7	2.5-3.5
4.4-4.8	Zircon	Zr Si O_4	Zr: 50	7.5
4.3-5.0	Ilmenite	$FeTiO_3$	Ti: 31.6	5.0-6.0
4.25	Rutile	TiO_2	Ti: 60	6.0-6.5
4.1-4.7	Chromite	(MgFe)O(Cr,A,Fe)$_2O_3$	Cr: 46.2	5.5-6.5
3.9-4.1	Sphalerite	ZnS	Zn: 67.1	3.5-4.0
3.5	Diamond	C	C : 100	10.0
3.4-3.6	Topaz	(ALF)$_2SiO_4$	Gem varieties	8.0
2.8-2.9	Dolomite	CaMg$(CO_3)_2$		3.5-4.0
2.7	Calcite-Aragonite	$CaCO_3$		3.0-4.0
2.65	Quartz	SiO_2		6.0

Figure 2. Some physical and chemical data of placer minerals.

- if there is little or no water, dredging is not possible.
- the largest throughput consistent with the size of the deposit is usually the most economical.
- dredging is not possible on steep surface inclines,
- if dredging is possible, it is always more economical than any other method.
- dredging requires more capital investment than dry earthmoving or gravel pumping, and thus requires larger deposits and longer mine life for amortization of equipment.
- gravel pumping is a low investment method but has a significantly higher operating cost per ton produced.
- dry earth moving equipment is normally standard equipment, and can either be purchased or rented on term contracts, so investment can be kept low, but operating costs are high.

These, and many more factors should be considered in any feasibility study together with a forecast of the sale of the mine product during the lifetime of the mine.

Some pre-dredging open cast mining practices are shown in Figure 1; a monitor - gravel pump mine and the use of belt conveyors in a tin mine.

The great advantage of these open cast mines is that the bedrock is visible, enabling efficient clean-up. Therefore these mines can have a R/E (Recovery/Evaluation) factor of 1.3, while dredging averages an R/E of 0.9 only.

However cost wise, the cost/unit throughput parameters of dredging, gravel pumping and dry earth mining roughly compares as 1 : 2 : 4.

3. Mineral Dressing

No matter which mining method is selected the next step in the operation is always the separation of the valuable mineral from the matrix material. The in-situ mineral content of the deposit may vary from less than 1 ppm to several thousand ppm, and it is therefore necessary to upgrade this valuable content as close to the deposit as possible to minimize transportation cost of material to the refineries where the mineral is produced in the purity required by the market.

Some, if not most deposits, contain a mixture of valuable minerals, one of which is the main target of the exploitation. The accessory minerals however, may be worthwhile to recover and sell to separate markets. The separation of the individual minerals often requires sophisticated equipment and close process control, neither of which are compatible with the relatively rough process of earth moving, whatever the method.

Therefore the on-site separation is primarily an upgrading process, using non-sensitive equipment and aiming at discarding as much of the matrix material as is possible.

Fortunately the valuable minerals have a significantly higher specific gravity than the matrix material, which is mostly quartz. (See Figure 2 for list of principal placer minerals). All methods use the 'Gravity separation' principle and require some preparation of the material such as screening, addition of water and control of feed flow.

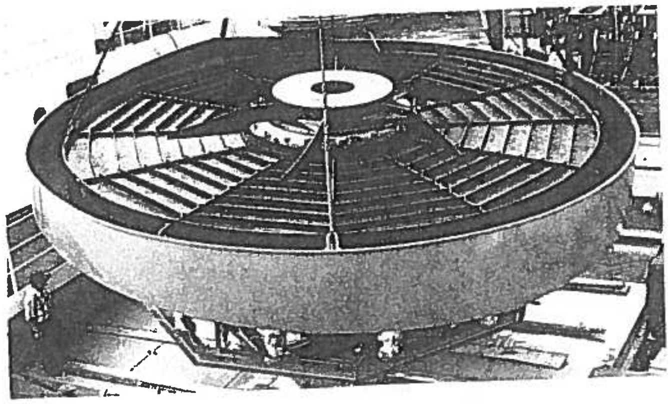


Figure 3. IHC's largest jig, the 12 cell, 42 cubic meters, type 25.

The principal types of separation equipment can be characterized as follows:

- sluices: simple and low cost, they require a high amount of water, periodic cleaning, have varying efficiency, are relatively high polluting, and are not sensitive to feed conditions.
- thin film separators such as trays, cones, spirals: very sensitive to feed conditions, require fine screening, close slurry control, low capacity per unit, average water consumption, suitable for low specific gravity minerals. The variety in these types of separators is staggering. Just to mention a few proprietory names: Van Bergen strake, Lovegreen jet separator, Kower separator, Kyna whirlpool process, Bartless-Mozley concentrator, HY-G concentrator, and Knelson concentrator etc. besides the initially indicated devices

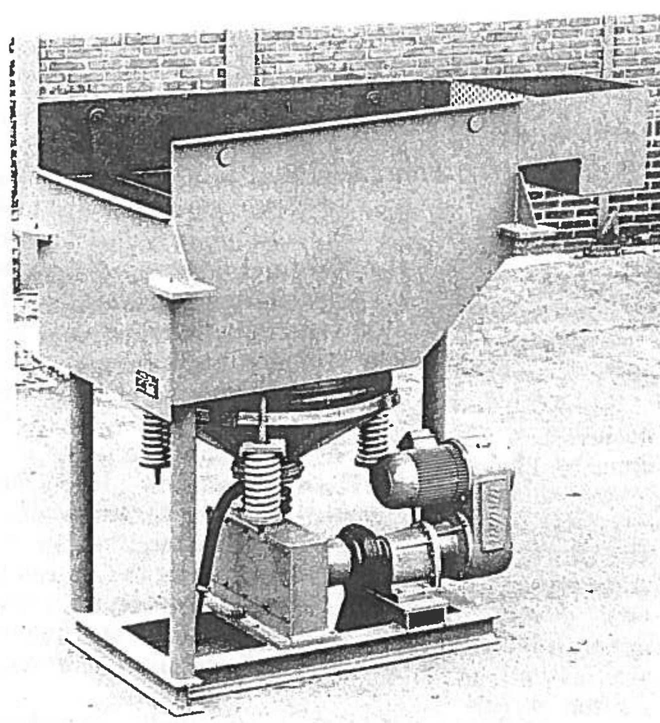


Figure 4. The IHC Mini-module Jig.

with brand names as Wright tray, Reichert Cone and Humphrey Spiral.
- jigs: pulsating thick bed separators, relatively insensitive to feed grain size and slurry density, insensitive to variations in feedrate, relatively high capacity per unit, simple staging, high efficiency, little maintenance, high upgrading possible.

Of specific interest to this audience will be the development of the IHC jig, a large capacity ore concentration machine which has found wide acceptance in the alluvial mining industry, specifically in tin and gold mining.

Figures 3 and 4 show two of the IHC jig types (largest and smallest) available, while Figure 5 shows our jig on a mining dredger. It is also interesting to note that we have a full size module type jig for testing purposes, located at the University of Technology in Delft.

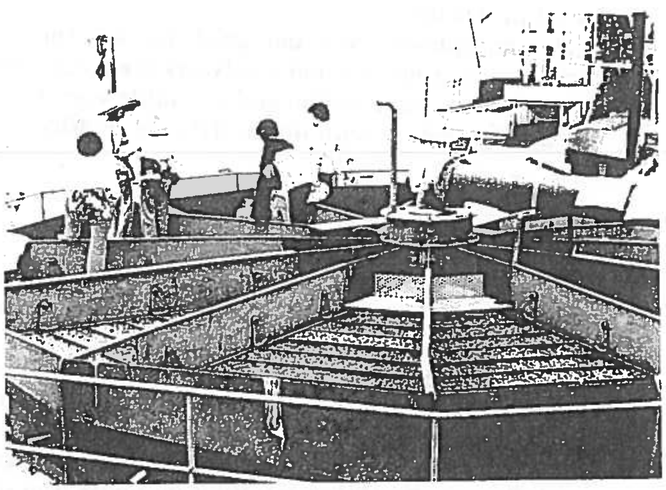


Figure 5. 12 module IHC Jig mounted on a dredge.

The deposit's characteristics determine which equipment is most suitable for a particular application. A jig plant, despite its higher cost and complexity compared with a sluice installation seems a good solution for many Alaskan gold operations because of its water economy and its high recovery efficiency. The possibility of designing and building a self contained plant of reasonable capacity for an acceptable investment, which can be skidmounted to follow the earth moving operation makes it an attractive alternative.

We will now describe such a plant, which is planned to be operational in the 1984 season. It will be used at the GDM operation, which we understand is typical for a large number of Alaskan operations.

4. Description

IHC will deliver to GDM a skid-mounted mobile treatment plant for alluvial gold mining with a separate diesel-driven power unit.

The plant is composed of two main sections, each 35 ft. long, 11 ft. 10 in. wide. The bottom section is 11 ft. high and the top section 10 ft. high. Both sections weigh less than 32 metric tons. In addition, there is a small stacker belt and a tailings launder. A basic feature of the plant is its low water consumption, thus reducing environmental pollution.

The treatment plant is designed to handle a feed of 200 cu. yds/hr (Figure 6). This feed is delivered by an owner-supplied track feeder and belt conveyor to a hopper, with a Vibro-barsizer to scalp +12" boulders. The grizzly is set at approximately a 20 ft. height, so as to ensure a maximum gravity feed inside the plant, minimizing pumping requirements. The hopper feed is delivered to a revolving scrubber/screen trommel to wash the feed and screen off the oversize.

The undersize is gravity fed through a screen collecting box to a three-stage jigging plant, concentrating and reducing the gold-bearing solids. Oversize material from the trommel screen is discharged to a stacker belt conveyor.

The primary jig tailings are gravity discharged into a tailing tank. A tailing pump draws from this tank and discharges through a dewatering cyclone, which is fitted at the tail end of the upper frame. The cyclone overflow is directed into the headertank, thus reducing the quantity of new water needed. Cyclone underflow is discharged into the tailing launder. The primary jig product is gravity discharged to a middling tank. From there the product is pumped to the secondary jig.

The secondary jig tailings are recirculated to the primary jig feed. The secondary spigot product gravitates to the tertiary jig. Tertiary jig tailings are recirculated to the middling tank The final concentrate,

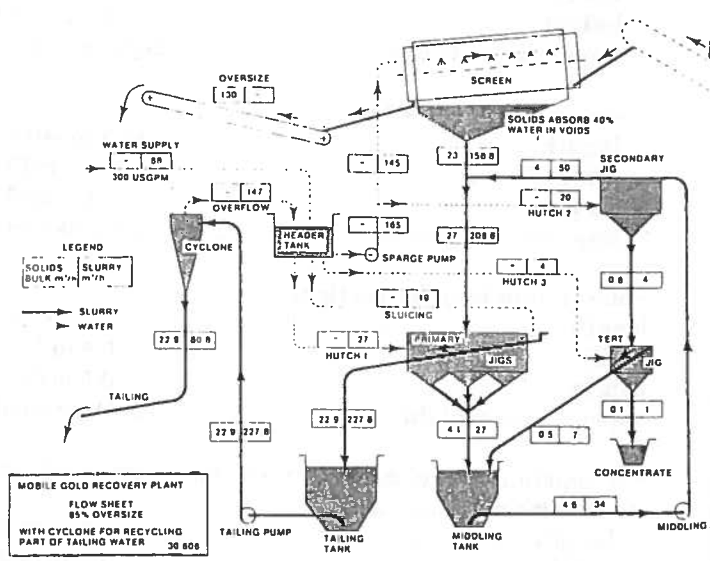


Figure 6. Flowsheet of the GDM mobile gold recovery plant.

recirculated to the middling tank The final concentrate, discharged by the tertiary jig spigot, can be collected in owner-supplied buckets.

Jig hutch water supply to primary and tertiary jig is from a header tank, which is kept filled by cyclone overflow and make up water. Hutchwater supply to the secondary jig is by means of a branch on the sparge pump discharge line.

Feed conditions may vary over a certain range. The plant is designed to handle a full feed rate of 200 cu yds/hr, containing an oversize (+3/8") percentage between 70 and 90%. For this reason the maximum stacker conveyor belt capacity is 180 cu yds/hr and the maximum jig capacity is 60 cu yds/hr plus recirculation products.

4.2 Principal Particulars

a) Treatment plant design criteria:

slope of revolving screen .... 1 in 12
slope of distribution and collecting chutes .... 1 in 6
slope of tailing chutes .... 1 in 12

b) Road transport dimensions

NOTE: Weights are the net construction weights, excluding jig ragging.

The plant will be sectioned as follows:

- one bottom section:
  length .... 10.4 m (34.2 ft)
  width .... 3.6 m (11.8 ft)
  height .... 3.35 m (11 ft)
  approximate weight .... 32 tons metric

- one top section:
  length .... 10.4 m (34.2 ft)
  width .... 3.6 m (11.8 ft)
  height .... 3.5 m (10 ft)
  approximate weight .... 26.5 tons metric

- one stacker belt section:
  length .... 12.3 m (40.5 ft)
  width .... 1.5 m (5 ft)
  height .... 1.5 m (5 ft)
  approximate weight .... 2.5 tons metric

- one tailing launder section:
  length .... 5.5 m (18 ft)
  width .... 0.4 m (1.4 ft)
  height .... 0.5 m (1.7 ft)
  approximate weight .... 0.9 tons metric

- a separate diesel-generator set to be mounted on a truck with dimensions:
  length .... 3 m (9.8 ft)
  width .... 1.5 m (4.9 ft)
  height .... 2.3 m (7.5 ft)
  approximate weight .... 3500 kg (7700 lbs)

b) Approximate weight of the plant in operating condition:

Nett construction weight of plant .... 52.9 tons metric (138,800 lbs.)
Diesel generator set .... 3.5 tons metric (7,700 lbs.)
Small mounting materials such as bolts etc. .... 0.1 tons metric (200 lbs.)
ragging .... 4.0 tons metric (8,700 lbs.)
water and slurry .... 18.0 tons metric (40,000 lbs.)

In total 88.5 tons metric (195,400 lbs.)

5. Gold Mining Dredging

The first gold dredger in the Yukon was started on the Lewes River in 1899 and in 1914 there were 42 dredgers operating. Therefore it may be of interest to give some information on a medium sized gold dredging operation, which uses modern dredgers with an integrated gold recovery plant in Northern China, where the climate conditions are similar to those found in Alaska.

The equipment was designed by us, the components and prefabricated steelwork were shipped to the minesite. Construction and assembly were carried out by the owners with our assistance in 1980.

5.1 Design Parameters

The mining dredger is an inland 300 litre bucket dredger with treatment plant for the recovery of gold from deposits in the sub-arctic region.

The solids dredged by the bucket line are delivered into a drop chute leading into the primary treatment plant through a revolving screen. Oversize disposal is from the screen via a stone chute. Tailing disposal is partly by gravity through tailing launders, the other part is discharged through a tailing pump via a cyclone after the chute. With this system the oversize material will be covered by fines which will enable re-soiling.

The treatment plant is comprised of a three stage jigging system. The tertiary jig concentrate is further upgraded by a hydraulic classifying process in the concentrate tank.

The dredger and its equipment are designed for continuous operation 24 hours a day.

The dredger is fitted with a diesel driven power plant, the power system being 380 Volts A.C., 50 c/s (3-phase). The top tumbler is electric motor driven thyristor controlled variable speed. The ladder hoisting winch, the headline winch and the side winches are hydraulic motor driven, with the hydraulic oil being supplied by electric motor driven power packs.

The dredger is designed for the following conditions:

- a rated dredging capacity of 243,000 cu. m/month at 600 dredging hours per month at 75× average bucket fill and 30 buckets per minute

- a maximum dredging depth of 11 m below water level
- typical soil analysis:

particle size mm	weight percentage
+100	4.78
100 - 50	14.74
50 - 25	14.80
25 - 16	8.09
16 - 10	6.9
10 - 4	12.69
4 - 2.5	5.6
2.5 - 1	5.96
1 - - 0.1	19.24
-0.1	7.2

- typical grain analysis of the gold:

particle size mm	weight percentage
+2.5	0.7
2.5 - 1	7.8
1 - 0.5	19.3
0.5 - 0.25	54.41
0.25 - 0.15	16.33
0.15 - 0.1	0.75
-0.1	0.68

- gold content average 0.316 grams/cu.m
- maximum face 2 m above water level and a average face height of 1.50 m.
- depth of the deposit below water level is max. 10.2 m, min. 3 m, average 6.16 m.

The alluvial from top to bottom consist of: humus, clay, upper sand and gravel, lower sand and gravel, and gravel.

The gold is mostly concentrated in the gravel. The bedrock is decomposed granite.

The top soil freezes up to a depth of 2.8 m below the surface while permafrost lenses occur at 4-6 m depth for 60%.

These permafrost lenses are defrosted two years prior to dredger operation. Surface is marsh land with grass with scattered bushes and small trees.

Silt content is as follows:

- upper sand and gravel .... 3 - 10%
- lower sand and gravel.................... 9 - 15%
- gravel max.20%

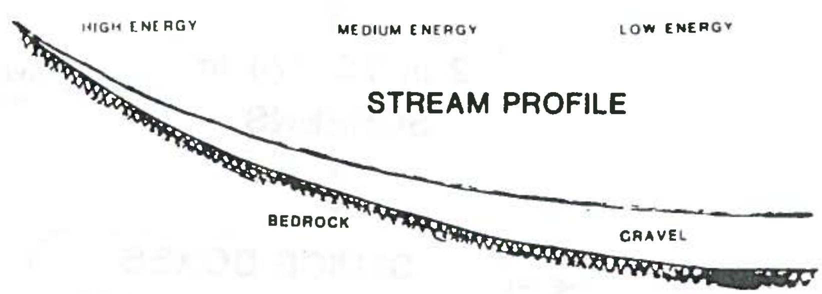
Specific gravity of the soil in situ is 2 tons/cu. m.
Swell factor of the soil is 1.16
Accessory heavy minerals of the gold are ilmenite and garnet.
Stones of over 200 mm in size are found in clay 1.33%, in upper sand and gravel 1.17% and lower sand and gravel 2.28% and in gravel 3.79%.
Minimum water flow is 0.312 cu.m/sec. There is a big river 10 km from the mine site.

Gold Recovery with Sluice Boxes

Will Godbey, Graduate Student

The recovery of gold from placer deposits is one of the oldest mining technologies known to man. The use of sluice boxes to recover gold is as old as man's known history and is still in use today. The idea for the sluice box came from nature itself. The first miners probably used the method of ground sluicing to recover the gold. This method does not recover the fine gold, but in areas where nature has created coarse gold pay streaks this method is satisfactory.

Figure 1 shows a profile of a typical placer stream. Since bedrock in a stream behaves much the same as in a sluice box, some direct analogies can be drawn. The high energy or upper section of a stream has a high gradient and contains the largest gravel sizes. It is here that most of the coarse gold will stay while the fine gold is carried away.



GRAVEL SIZE	BOULDERS & COARSE GRAVEL	MEDIUM & FINE GRAVEL	FINE GRAVEL
DEPOSIT YARDAGE	LOWEST YARDAGE	MEDIUM YARDAGE	LARGEST YARDAGE
CHARACTER OF GOLD IN DEPOSIT	COARSE GOLD ON AND NEAR BEDROCK	MEDIUM & FINE GOLD ON AND NEAR BEDROCK FINE GOLD ABOVE BEDROCK	FINE GOLD ONLY MOST OF GOLD ABOVE BEDROCK
GRADE	STEEP	MODERATE	FLAT
SLUICE ANALOGY	UNCLASSIFIED FEED RIFFLED SLUICE	CLASSIFIED FEED RIFFLES AND/OR EXPANDED METAL	CLASSIFIED FEED CARPET AND MERCURY

Figure 1. Stream Profile.

The same will happen in a sluice box that has an unclassified feed. In the stream, the medium and fine gold do not begin to drop until the coarse gravel has been left behind, i.e. the gradient decreases. It is in the medium energy area that the problems with sluice boxes begin. A sluice box having a classified feed will be able to catch medium and some fine gold, but a lot of fine gold will get away. The fine gold gets away because the water velocity is too high to allow it to settle.

In the low energy environment of the stream the fine gold is able to settle out but unfortunately the gravel bed is not loose enough to allow it to drop to bedrock. The same thing happens in a sluice box. For fine gold to settle, the water velocity must be reduced. Unfortunately this allows the sluice bed to pack and thus the gold is unable to penetrate.

With these basics understood, a flow sheet can be developed that will show a sluice box doing the same things that nature does in her streams, only better. Figure 2 shows this flow sheet.

First, rocks larger than the largest nugget you expect to find must be screened out. Then the gravels are put through a nugget trap to get all the large gold out of the gravel. The gravel is screened and put through sluices to recover medium and some fine gold. The sluice tails are again sized to get rid of all but the fine gravels and gold. This feed is then processed through some type of fine gold recovery unit. This paper will not discuss the use of devices other than sluice boxes and little will be said at this time about fine to very fine gold recovery units.

Most operations use some type of grizzly to remove the large boulders. Few operations can run all their ground straight through a box but those that can and do are losing a lot of gold. Figure 3 shows the Moganson (German Co.) Sizer. In the area I've been working there is so much glacial material with flat boulders that it tends to hang up in the grizzly. Most people are now either picking out the boulders one at a time or just dumping them all in a sluice box and putting up with it, causing tremendous wear. The Moganson sizer has a three dimensional screen. On this grizzly the bars diverge not just laterally, but vertically also. The result is that you can get a greater spacing from the top of your bars to the bottom of your bars. Not only that, but these bars vibrate, reducing hang ups. In the area I'm working, the clay on the boulders allows them to wedge in so solidly that you can't get the boulders out, no matter what you do. Some miners have been using their backhoes, to remove boulders, ripping their grizzly apart.

Once the gravel passes through the grizzly it usually drops into a dump box. It is here that the gravel should be disintegrated and the nuggets extracted before the undercurrent screens are reached. Many operations use the undercurrent screens first and then the oversize

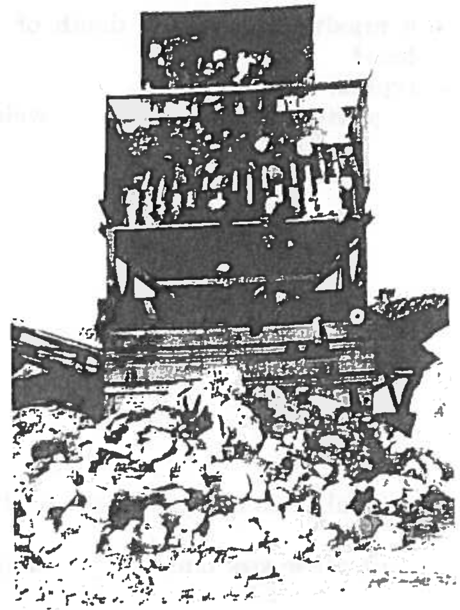


Figure 3. Moganson Sizer.

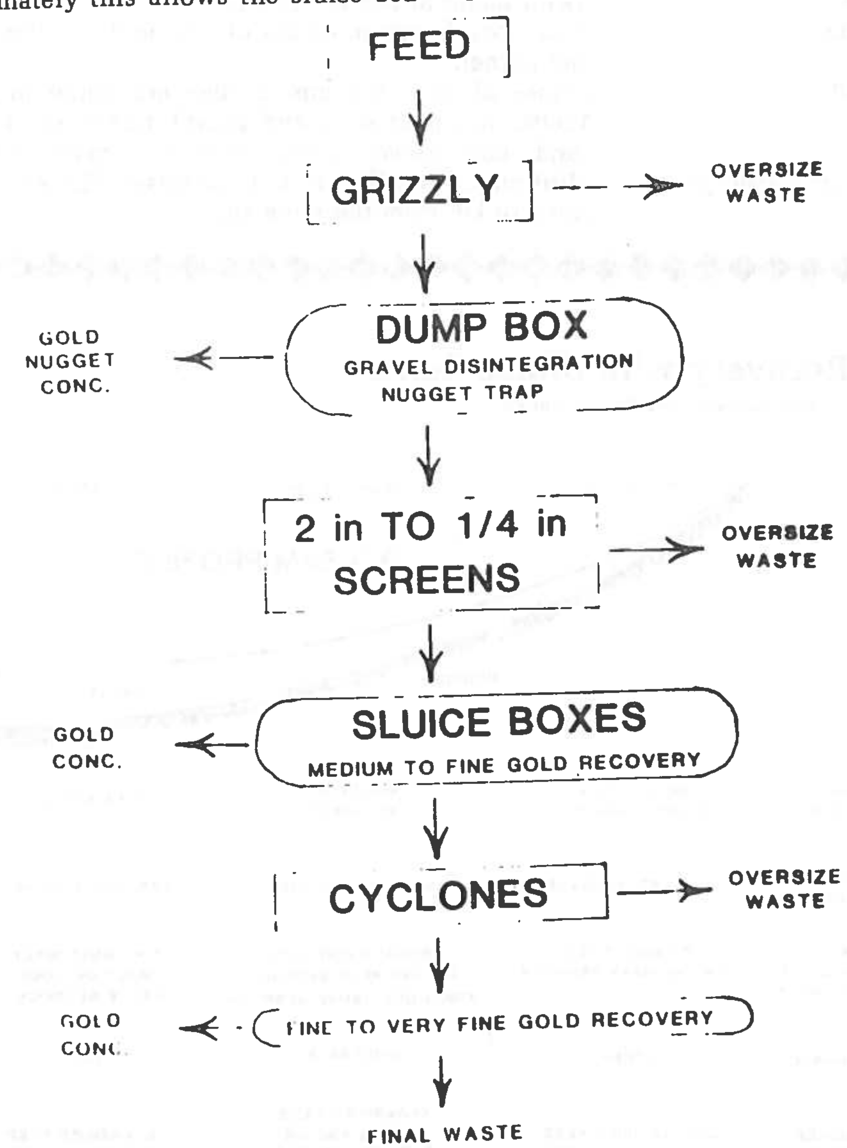


Figure 2. Flowsheet for gold bearing placer gravel processing.

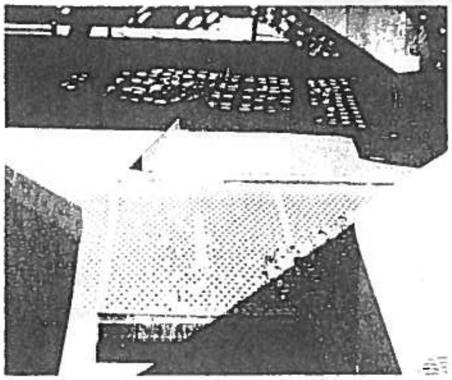
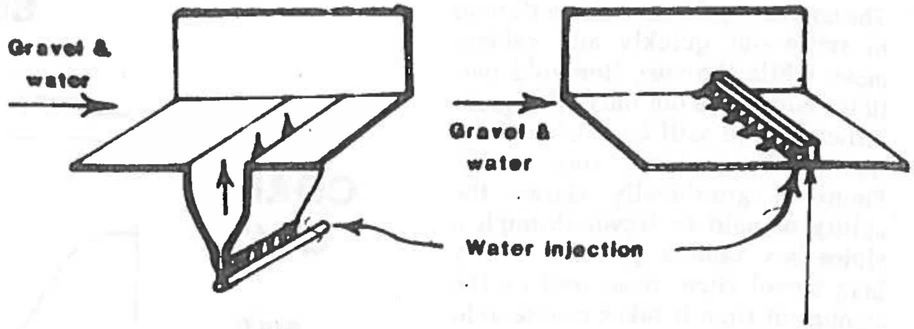


Figure 4. Sluice punch plate.

goes to a large sluice box which is used as a nugget trap (Figure 4). This method is not as efficient and results in greater losses of gold. If the nuggets are caught before the undercurrent is reached, the fine material in suspension makes it easier to concentrate the nuggets. Figure 5 shows an example of two types of nugget traps. When fine material is present in the sluiced pulp, the density of the resulting slurry, or pulp, is greater than that of water alone. Using a specific gravity for gold of 17 and a specific gravity of 3 for the gravel, then the ratio of effective specific gravities is 15 for an unclassified feed and 8 for gravel which has had the fines removed. This means that coarse gold can be separated more efficiently in a sluice box or nugget trap which has both coarse and fine gravel running through it, compared to one that processes only coarse gravel. Another problem exists when putting the undercurrent screen before the coarse gold trap. In order to move this material through the sluice box a great deal of water at a faster flow is required. This means that most of the water goes over the undercurrent screen and not through it. This results in most of the fine gold also going over the undercurrent screen and almost all is lost. Only when all of the water goes through the undercurrent can you be sure the fine gold has also gone through.

Although gold is very dense, this does not mean it will not move in a sluice box. Looking back to Figure 1 (stream profile) we can see that in nature the gold particles which are finest and also flattest, will travel further down a stream valley. The same thing happens in a sluice box.



1 Use before classification

Fine material increases density of water (pulp) and makes it easier for nuggets to settle

2 Use after classification

Gold nuggets much cleaner
Less volume to handle

Ratio of effective specific gravities

$$\frac{\text{SpG Au} - \text{SpG pulp}}{\text{SpG Gravel} - \text{SpG pulp}}$$

EXAMPLE

No classification

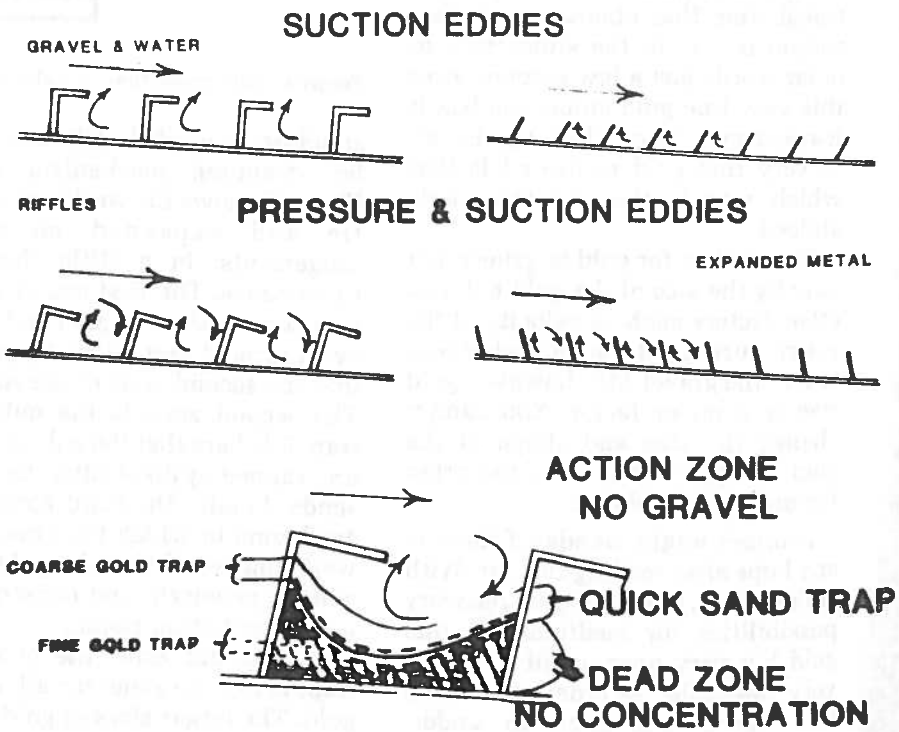
$$\frac{17 - 2}{3 - 2} = 15$$

Classification

$$\frac{17 - 1}{3 - 1} = 8$$

Figure 5. Nugget trapes (above).

Figure 7. Riffle action (below)



The coarsest gold particles will tend to settle out quickly and seldom move while the very fine gold particles will settle out only with great difficulty and will be flushed from their resting places very easily. Figure 6 graphically shows the ability of gold to travel through a sluice box. Coarse gold has a very long travel time, measured as the amount of time it takes coarse gold to travel one foot down a sluice box.

Medium gold has a shorter travel time and some of this gold will be lost out of the box in a matter of minutes. The flatter gold particles have the shortest travel time. If you can recover most of your flat medium gold in the head of your box, then you have increased the travel time of this gold enough to keep it in your box till clean-up.

Fine gold has a short travel time and is very difficult to trap at any point in the sluice box much less at the head where you want it. A small amount of fine gold will have a long enough travel time that there will be some concentration at the head but most will make its way down the box. Once this gold starts leaving the box, the amount leaving is almost equal to the amount being fed into the box. Once this condition is reached, the amount of fine gold recovered will not change much no matter how many yards are sluiced.

Very fine gold has such a short travel time that almost no concentration occurs in the sluice box. In other words just a few seconds after this very fine gold enters the box it leaves again. Upon clean-up the only very fine gold recovered is that which was in the last few yards sluiced.

Travel time for gold is gauged not only by the size of the gold but also other factors such as velocity of the water, surging of the feed, clay content of the gravel, etc., however, gold size is a major factor. You cannot change the size and shape of the gold but you can control the other factors to some extent.

A miner might wonder if there is any hope after reading this far. With sluice boxes, there are good recovery possibilities for medium and fine gold but very poor possibilities for very fine gold. In order to achieve these good recoveries an under-

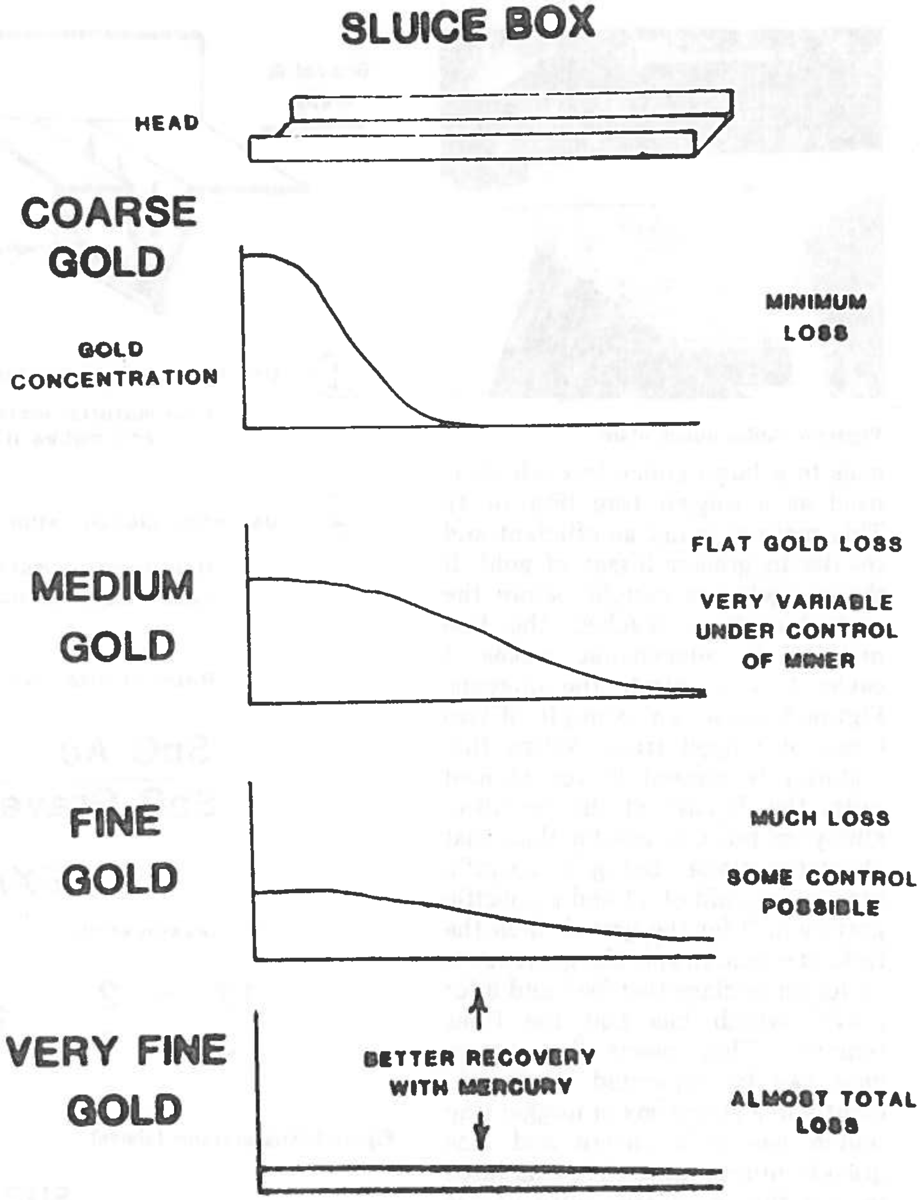


Figure 6. Gold travel time in a sluice box.

standing is needed of how a sluice box trapping mechanism works. Figure 7 shows the workings of a riffle and expanded metal arrangements. In a riffle there are three zones. The first one is the action zone where the gold and gravel swirl around and either drop down into the second zone or are rejected. The second zone is the quicksand trap. It is here that the gold particles are trapped by displacing the lighter sands. Finally the third zone is the dead zone in which the gravel is so well compacted that the gold is not able to penetrate and consequently no concentration occurs.

The second zone, the quicksand trap, is not the same for all sizes of gold. The larger sizes of gold need a thicker quicksand trap and the smaller sizes a much thinner trap located much lower in the riffle. This quicksand trap does not stay in the same place but moves up and down according to material feed rate and water velocity. A very high feed rate or low water velocity will cause the quicksand trap to rise, perhaps even above the riffle surface, which is the case of sanding up of the riffles. On the other hand, a low feed rate or high water velocity will lower the quicksand trap, possibly even to the bottom of the box. It is at this point that all the fine gold present in the quicksand trap will be flushed out. If the water velocity becomes extremely high some of the coarse gold itself may even be re-

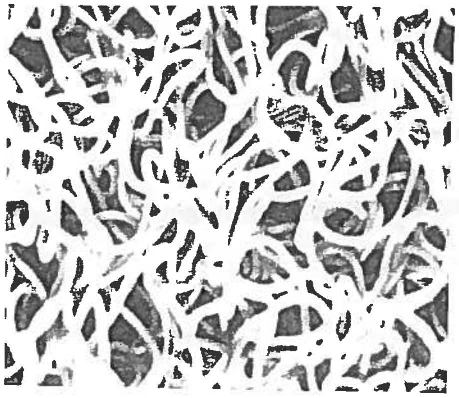


Figure 8. NOMAD® carpet.

jected. Fluctuation in the height of the quick sand trap is very detrimental to fine gold recovery. Inducing contact between the fine gold and the quicksand trap is the second major problem.

Figure 7 shows the type of action one gets with riffles and expanded metal. With riffles put in flush or expanded metal put in the usual way, the quicksand traps are created by only suction forces. However, if the riffles are tilted or expanded metal is put in backwards, one gets not only suction forces but pressure forces as well. With both forces present, the quicksand trap becomes thicker because more energy is available to create it. This can cause a problem with riffles in that it is easier to drop the quicksand trap down due to booming and thus lose fine gold. However, this extra boiling action gives the fine gold greater access to the trapping medium. The case with the expanded metal is just the opposite. In the standard orientation with only suction eddies present it is easier to lose the gold and bottom out the quicksand trap than it is when the expanded metal is reversed, causing the eddies to be more violent. This is because the pressure eddies drive the material down and keep it down. This results in the expanded metal bed always being full and looking like it is not working but in reality it is working very well and the quicksand trap is protected from quick movements up and down. This is the case only when two or more layers of expanded metal are used. Fine gold also has very good access to the quicksand trap because of the pressure eddies forcing fine material through the expanded metal.

Carpet works for the same reason. Quicksand traps are created in it just the same as with riffles. The reason fine gold is trapped is because the pore spaces in carpet are too small to allow large gravel in thus only fine gold quicksand traps form (Figure 8). These traps allow easier access for fine gold and most importantly these traps fluctuate very little and the gold is not easily flushed out of the carpet. Carpet essentially screens the material running over it and creates many small traps which can hold the fine gold dropping close enough to access these traps.

Figures 9 and 10 show NOMAD® carpet and expanded metal together. The expanded metal creates the large quicksand traps for the coarser gold while also screening out the larger gravel so only the fine gravel can get access to the carpet. This not only results in longer life for the carpet but better access for the fine gold to the carpet traps.

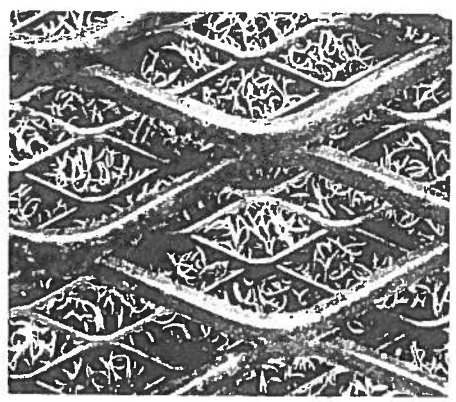


Figure 9. NOMAD® carpet overlain by two layers of expanded metal.

Figure 11 shows a system of screening which gives the greatest chance for gold to work its way through the punch plate and still leave enough water on top to transport the oversize material. This punch plate has a layer of Linaclad rubber on top to keep the rocks bouncing so the punch plate will stay clean. This system was designed and used with great success by Jim Wallis of Atlin, British Columbia. His screen was around 70 feet long and the entire area under it was covered with heavy duty NOMAD® carpet with a layer of expanded metal over it. The resulting action from the punch plate and the flow of

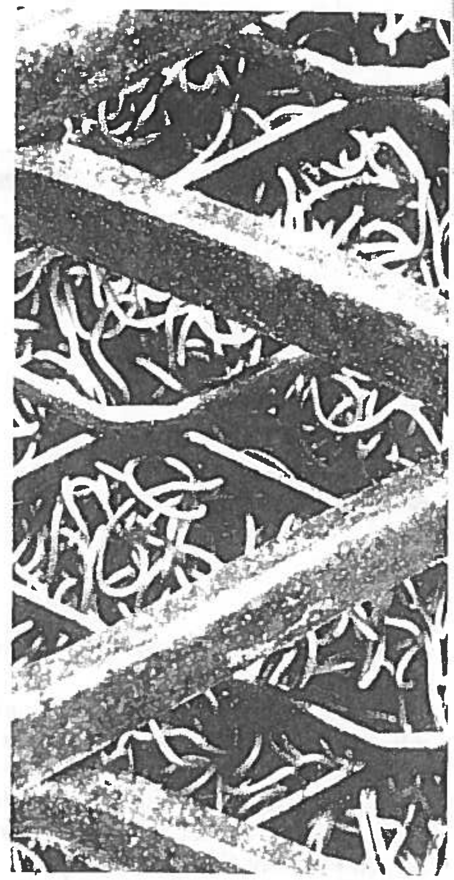


Figure 10. Close-up view of Figure 9 (NOMAD® carpet overlain by two layers of expanded metal).

material over and under the wooden blocks gives the fine gold tremendous access to the carpet quicksand traps. This action also keeps the quicksand traps in the carpet alive. All the material that works through the punch plate eventually goes to side sluices (undercurrent boxes). Jim told me that the total width of these boxes was 40 feet compared to 4 feet for the punch plate box. The action in the punch plate section was so good that approximately 80% of the gold was caught here and only 20% in the side boxes.

This undercurrent is vastly superior to any other I know. It not only helps the gravel to disintegrate but gives fine gold many chances to pass the punch plate. It also makes use of the area under the screen for catching gold instead of wasting it like most boxes do. This area under punch plates or under trommel screens (Figure 12) gives fine gold the greatest possible access to quicksand traps if heavy duty carpets are used. NOMAD® carpet seems practically indestructible and creates the

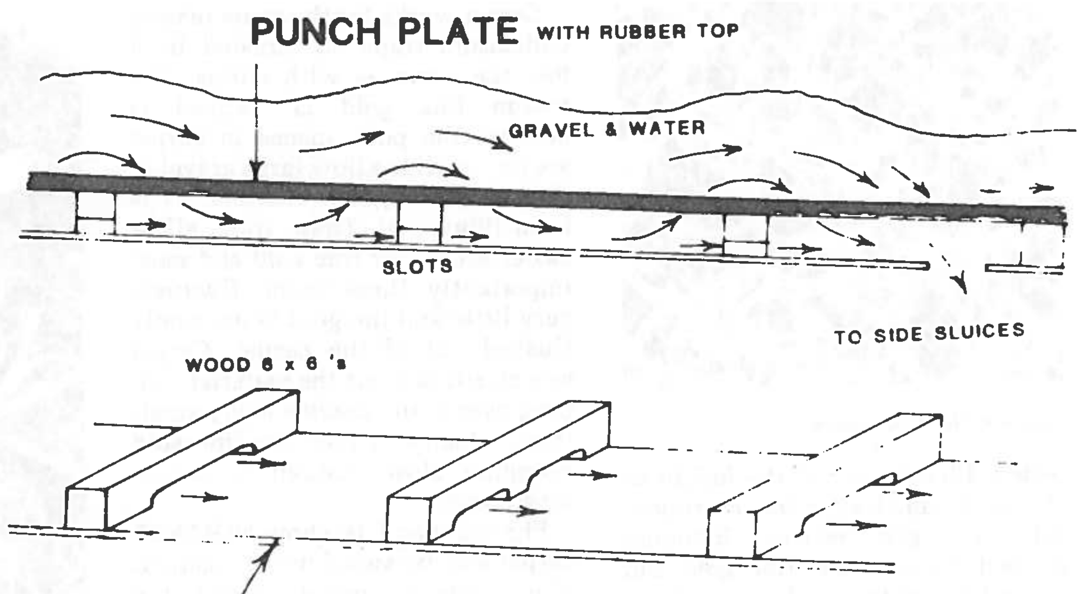


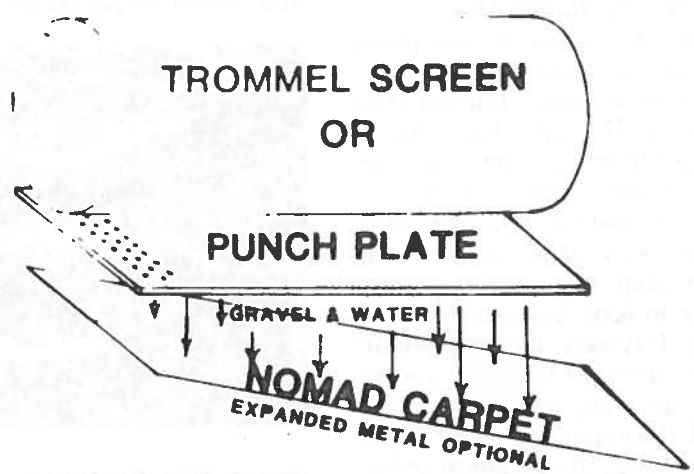
Figure 11. Fine gold recovery design for a sluice box.

greatest number of small quicksand traps.

Carpets which have only vertical pile are much more susceptible to scouring and thus to a loss of fine gold. Scouring causes the action zone to drop to the bottom of the carpet, ejecting the captured gold.

Vibration of sluice boxes helps increase recovery by allowing one to slow the velocity of the water and gravel without losing quicksand traps. This lower velocity allows the finer gold to settle. One way to get vibration is by suspending boxes with chains. Once the gravel is screened to -2 inch, it is more economical to construct your sluice boxes out of wood since wood is lighter, cheaper, and does not freeze as easily as steel. For this reason, wood boxes can extend your season up to 2 weeks.

Finally, a word about the importance of sluice boxes. Half of the world's supply of tin comes from placer mines where hydraulic monitors are used to break and move gravel. Large pumps move the -6 inch material to wooden sluice boxes. The other half of the world's tin supply comes from 3/4 of all the mining dredges in the world. This numbers into the hundreds, with most of the dredges having over half a million cubic yards per year through-put capacity.



CARPET SCREENS OUT OVERSIZE

BOUNCING ROCKS KEEP QUICK SAND TRAPS IN CARPET FLUID

ALL GRAVEL HAS ACCESS TO TRAPPING MEDIUM

SURFACE AREA / CONC. VOLUME VERY HIGH. BEST FOR FINE GOLD

DEAD ZONES VERY SMALL

VERY EASY TO CLEAN SO DAILY CLEANING CAN BE DONE

Figure 12. Advantages of placing carpets beneath trommel screens or punch plates.

SOME ASPECTS OF GOLD RECOVERY WITH IHC JIGS

by

Del Ackels

Del Ackels has mined on Gold Dust Creek in the Circle mining district since 1965. Simultaneously, he has worked for the benefit of the Alaskan mining community. Del has served on such committees as the Alaska Miners Association, Placer Mining Committee and also on the Board of Directors. In addition, he has served as a board member of the Federal Citizens Advisory Commission and is the past-president of the Circle mining district. Del has an engineering background and within the past few years has applied his knowledge to the selection of equipment that might serve the placer miner by recovery of the fine gold, not achievable by standard sluicing methods while, at the same time, utilizing a minimal amount of water.

ABSTRACT

After a brief introduction into the bewildering multitude of gold recovery systems, and then limiting ourselves to the gravity devices only, the "thick bed" concentrators - Sluice and Jig are discussed.

Due to their tolerance to feeding abuse these gravity separators are the survivors for "in-mine" or "on-dredge" use. To recover the fines, jigs should be preferred. The difference and superiority of the IHC-jig concept is highlighted.

An attempt to project the recoveries of the successive size fractions of gold is made in addition to some remarks on feed preparation.

INTRODUCTION

Gold Dust Creek is located near 100 Mile Steese Highway in the Circle Mining District. Like many other placer mining operations, we have had a number of problems with fine gold recovery. For years we had attempted to resolve these problems through various modifications to our sluice boxes. However, a couple of years ago, after examining our tailings and actually performing some screening and other tests we found that significant losses of fine gold were occurring, especially in certain size ranges. Sixty mesh was the most notable range where large losses were suffered. This was especially true when we recycled the water. In addition, the losses appeared to be directly related to particle shape, size and cementing problems within our sluicing system.

To resolve some of these problems we began looking at different types of recovery systems and their associated technology. This eventually led us to jigs. Although jigs are basically very simple, at that time they were very mystifying, magical things. As a result, in order to develop the knowledge base that we needed from the various manufacturers, it was necessary for us to look outside the standard placer gold recovery methods and carefully examine other types of alluvial concentrating systems. We soon found that placer operations seeking economic minerals other than gold not only have problems similar to ours but these are amplified because they are attempting to recover minerals that are one-third the weight of gold, ranging around specific gravity four to seven.

During our study we found that there were certain manufacturers, especially in some of the world-class operations involved in offshore dredging, that had been successful in developing technology which guaranteed relatively high recovery rates on cassiterite with a specific gravity of around seven.

From that point we travelled to Phuket, Thailand, to look at what was available on some of the tin (cassiterite) dredges. Although we did see some operations that were utilizing sluice box systems very similar to those we use, many of the larger dredges had converted to jigs. Upon the introduction of jigs into the South East Asia placer industry, tin recovery improved, often as much as three times that obtained by the conventional sluicing systems. Tests where jigs were placed at the end of these sluicing operations revealed that the jig was capturing as much as 70% of the primary product.

On the particular dredge we visited they were using circular jigs. The circular jig concept is one of the primary reasons we took the trip. There are certain advantages to the circular jig, not only in the minimal amount of room they require, but also the rather impressive amount of material they are capable of handling. In addition, these jigs are hydraulically driven, which has an advantage over mechanical drives.

Out of all our studies it was recognized that the major objective was to locate the appropriate primary recovery system. Whether a jig or any other type of system, the primary recoveries are the most important. As will be pointed out later, if one does not achieve a high recovery at the primary what is done down the line doesn't make much difference.

JIG OPERATION

By way of demonstrating how a jig works, there are basically two types, the standard rectangular jig and the trapezoidal jig. The rectangular jigs are mechanically driven and contain one or more square or rectangular cells aligned in a series as shown in the top example of Figure 1-A. The trapezoidal jig, on the other hand, is and independent, trapezoidally shaped recovery system as shown in Figure 1-B.

In comparing the size and ease of feed distribution within complete jig plants constructed from the two types it is

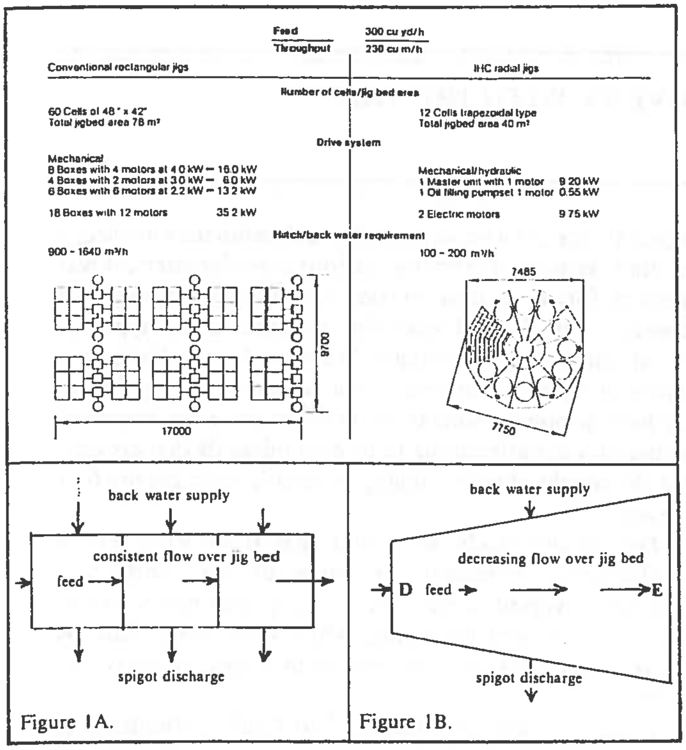


FIGURE 1: Comparison of IHC radial jig and conventional rectangular jigs.

recognized that the rectangular jig system would require significantly more space than a plant of equal processing surface utilizing trapezoidal jigs. This becomes apparent when one considers that the trapizoidal shape is ideally arranged into a circular or radial jig plant consisting commonly of twelve trapezoidal jigs. It should be kept in mind that fewer trapezoidal jigs can be combined which form less than the entire circle. This, of course, is dictated by plant design and volume requirements. Another advantage to the radial form is the single central feed intake which eliminates the necessity of distributing the feed to multiple rectangular jig lines.

In general, feed is introduced at the upper end of a jig and works its way down the jig table and out the lower end. The basics of the system, as shown in Figure 2-A and 2-B, include a cell which contains water. Above the water cell is a punch plate or screen which has the appropriate sized perforations or openings. Above this is placed bedding material often referred to as "ragging material", which supplies the bed load necessary in capturing the primary product. Ragging material can consist of a number of materials including larger cassiterite nuggets, steel shot, chain, taconite pellets or any substance around a specific gravity of 5-8. The bottom part of this cell is driven up and down either by a mechanical or hydraulic method.

Figure 2-A illustrates that when the bottom portion of the jig (jig diaphragm) is driven up, it causes the water column to be forced up through the screen and ragging material. Simultaneously, the upward movement of water forces the feed to rise into the upper portions of the water flow, commonly referred to as the "cross flow". This carries

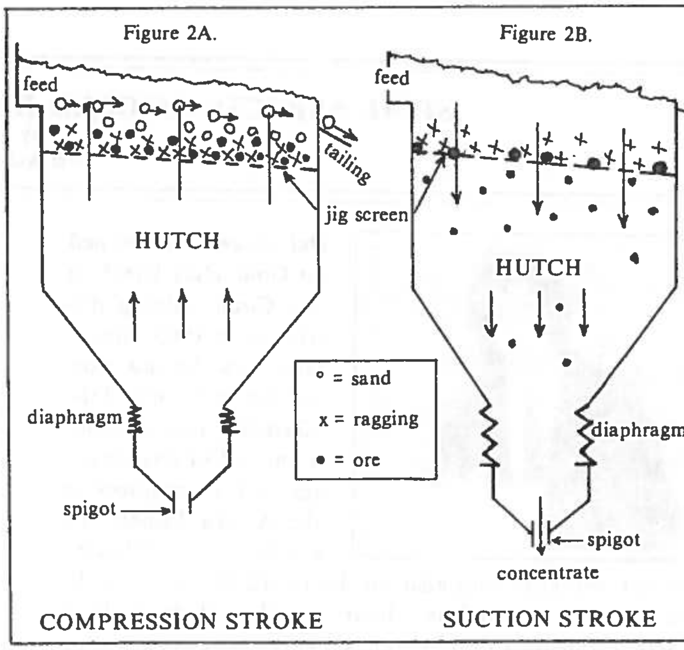


FIGURE 2: Basic outline of a jig.

the lighter material to tail while during the subsequent down stroke, as shown in Figure 2-B, the heavier concentrate is pulled down and locked into the ragging material or, as in the case with the finer material which passes through the punch plate or screen, it drops to the bottom of the cell and is drawn off. As this process proceeds a certain amount of water is lost due to the transportation of the low specific gravity material to tail. As a consequence, it is necessary to add water to the hutch. There is a good reason for maintaining the proper water level. As pointed out earlier, when the jig strokes up it forces water out of the hutch along with the tailings. If water were not added, the subsequent down stroke might cause a vacuum or void and this in turn would cause the bedding to become very hard or very dense and retard the concentrate from getting pulled back to the bed. The continual addition of water to the hutch causes a flow up through the punch plate which alleviates this problem. But, within a rectangular jigging system, a compounded problem develops when the hutch water from the first jig is passed onto the second. The water velocity is accelerated. In contrast, to avoid this problem the feed is introduced into the center of the radial jig and as a result can be split into several directions. In addition, the shape of the trapezoidal jig which fans out from the smaller intake end to the wider discharge end eliminates some of the hutch problems by maintaining a linear decrease in flow velocity across the jig bed as indicated from point "D" to "E" in Figure 1-B.

Figure 3 is a view of a complete radial jig plant. As shown, the feed enters from the center. Each cell is a hydraulically driven independent recovery system.

To bring the concentration method into clearer focus we must examine the jigging action and some of the phenomena associated with it.

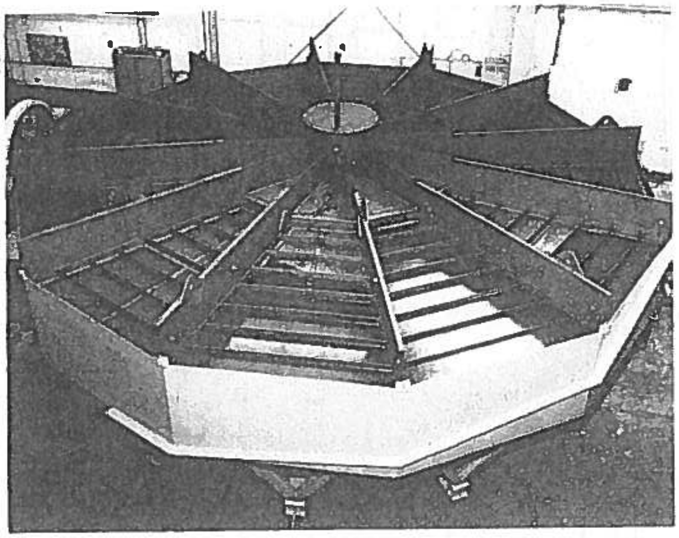


FIGURE 3: 12 module type jig.

Jigging Action

In a jig the mineral particles in the fluid medium are subjected to pulsating water current. Besides gravity settling and lateral transport, these alternating water currents interrupt and reverse the vertical movement of mineral grains depending on their weight and shape, thereby resulting in a differential settling of heavy minerals in preference to lighter grains. Some of the phenomena occurring in the jigging action are:

a) differential acceleration at beginning of fall
b) hindered settling classification
c) consolidation trickling

In way of demonstration, consider the behavior of four specially selected grains: a fine quartz sand grain (specific gravity 2.65), an equally fine heavy mineral grain such as cassiterite (specific gravity 7), a coarse quartz sand grain and an equally coarse grain of cassiterite.

With these four grains we can give a schematic discussion to visualize jigging and its associated phenomena.

A) Differential acceleration at beginning of fall - this is the initial movement of grains at the start due to the acceleration by gravity. Where the falling time is very short, i.e. where the speeds occurring are low and the resistance therefore can be ignored, the initial speeds will depend only on the specific gravities of the grains. For a stagnant medium this can be represented as shown in Figure 4.

B) Hindered settling classification - if we examine the grains after a longer time, the speeds will have increased to such an extent that a balance is struck between the resistance which the grains meet in respect of their speed and the downward force due to gravity so that the grains have attained what is called their terminal velocity.

Because numerous grains settle simultaneously and consequently influence each other, the process is referred to as "hindered settling", whereby the terminal velocity of the lighter grains is more adversely affected than that of the heavy grains, thereby enhancing the hindered settling ratio

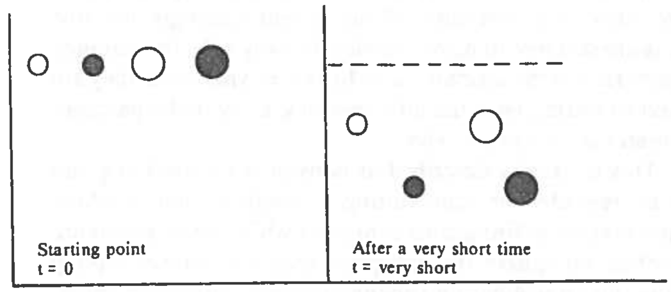


FIGURE 4: Differential initial acceleration between low and high specific gravity minerals.

compared to the free settling ratio.

As these terminal velocities are dependent on weight and also on size, the behavior of our selected four grains in hindered settling condition can be represented as shown in Figure 5.

If we consider the settling process of these four grains in an upward flow with speed V1, we see that the fine light grain is propelled upward and carried away in the cross flow because its settling speed is lower than V1, while the other three grains continue to settle against the upward flow.

If the upward speed of V2 is chosen, only the coarse grains will settle, and the fine ore will also be carried away with the cross flow.

Finally, at an upward speed of V3, only the coarse ore grain will settle. From the foregoing it can clearly be seen that, in case of hindered settling, it is not possible to choose a magnitude for the upward flow that allows the fine heavy mineral to be separated from the coarse sand (classification process).

At the speed of V3, the coarse heavy mineral is fully separated from the sand. Consequently, hindered settling is suitable for separation of coarse minerals, but not for fine grains.

C) Consolidation trickling - in view of the fact that different grains of either the same or different specific gravity do not travel the same distance during one of the settling periods, they will come to rest at different instants as shown in Figure 4 and Figure 5.

A coarse grain may remain in suspension when considering a settling time of 0.5 seconds, perhaps for only 0.05 seconds, while a small grain may remain suspended for as long as 0.3 seconds. Clearly a period of time exists during

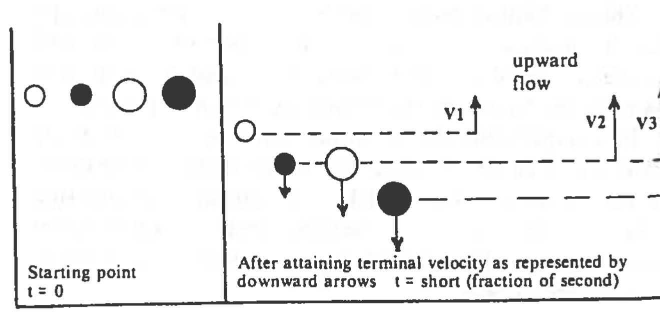


FIGURE 5: Hindered settling.

which the fine grains are still moving while the coarse grains are immobile. The coarse grains have settled, bridging against each other and incapable of movement although the fine grains are still free to move. Aside from any velocity that may be imparted to these small grains by the moving fluid, they are bound to settle under the influence of gravity in the passages between the coarse particles.

This is usually described as consolidation trickling and can be regarded as representing a condition under which sedimentation of fine grains continues while coarse grains are immobile. Of course, the fine grains may not settle as rapidly during this consolidation trickling phase as during the initial acceleration or suspension, but if consolidation trickling can be made to last long enough the effect, especially in the recovery of the fine heavy minerals, can be remarkable. This is easily visualized by examining Figure 6.

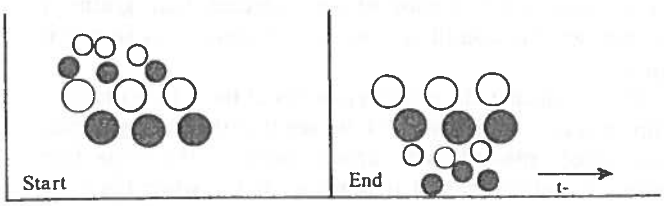


FIGURE 6: Consolidated trickling.

To summarize we can visualize an ideal jigging process by examining the combined effects of differential initial acceleration, hindered settling and consolidated trickling on our four selected grains as shown in Figure 7.

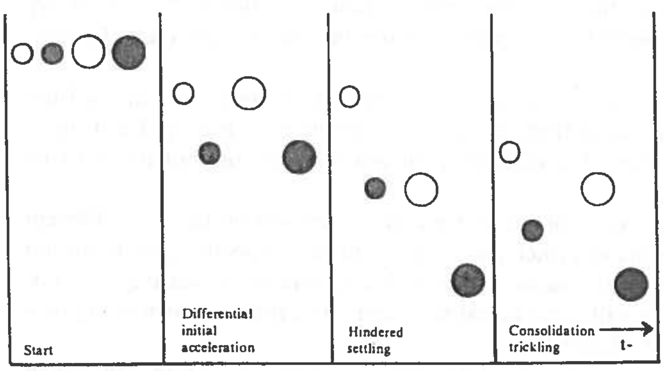


FIGURE 7: Ideal jigging process.

Comparison of Mechanical Drives of Conventional Rectangular Jigs with Hydraulic Drives of Trapezoidal Jigs

The mechanical drive of the conventional rectangular jig generally provides a harmonic pulse cycle. The time and distance is divided equally between the up and down strokes. This provides harmonic wave form as shown in Figure 8-A.

In comparison, the hydraulic drive of the radial jig provides a much cleaner and more controllable stroke cycle. The up stroke is a very quick short upward compression stroke while, in comparison, the downstroke is a much slower suction stroke. This provides a saw-tooth wave form as shown in Figure 9-A.

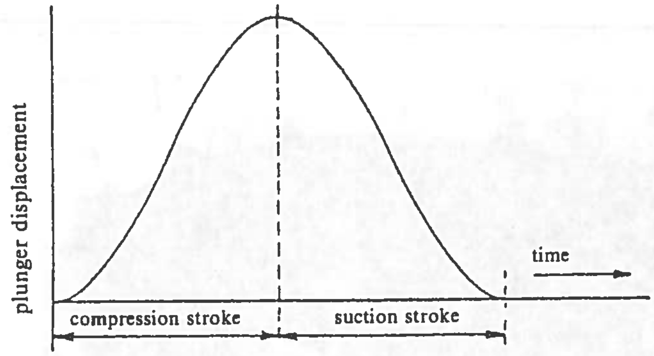


FIGURE 8A: Harmonic wave form of mechanical drive typical of conventional jigs.

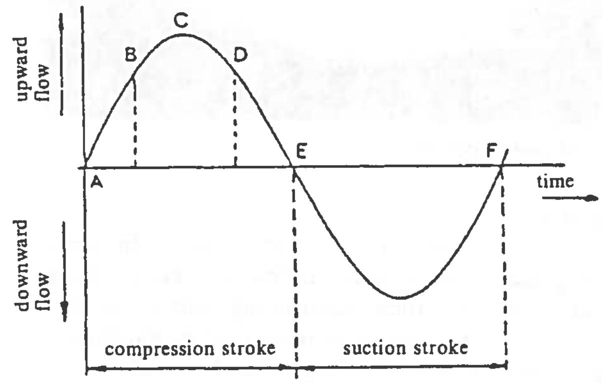


FIGURE 8B: Velocity/time diagram.

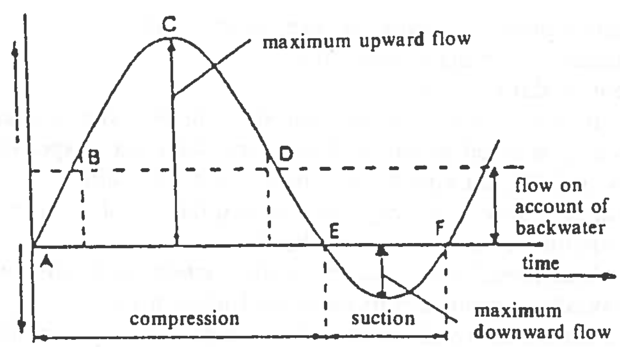


FIGURE 8C: Velocity/time diagram with back (hutch) water.

Conventional Jig with Harmonic Wave Form

In a conventional rectangular jig the pulsating water currents are caused by a pulsator or plunger actuated by a mechanical eccentric drive mechanism.

The resultant pulsator movement is a harmonic wave form and can be represented as shown in the displacement/time diagram in Figure 8-A.

The vertical flow speed through the bed is proportional to the speed of the plunger. When the speed of the plunger is greatest, the corresponding speed of flow through the bed is also greatest.

The speeed-of-flow diagram (Figure 8-B) is derived from the displacement diagram as shown in Figure 8-A. The upward speed of flow increases after point A. In the beginning, due to hutch water, there will be a weak flow through the pores between the grains of the light bed, which will then still be firmly planted. When the speed of flow becomes greater, the grains will be loosened and the bed forced up or "open". Assuming that this situation has been attained at B, the grains are now in the phase of hindered settling in an upward flow, and since the speed of flow from B to C is still increasing, the fine sand and ore grains are pushed upward by the flow. The possibility that the finer ore grains will be lost when they are forced into and carried along the top flow to the tailings is enhanced. In the vicinity of D, first coarse grains and later on the remaining fine grains will fall back. The combination of initial acceleration and hindered settling now occurs and it is mainly the coarser ore grains that will lie underneath.

During the period E-F the flow through the sand bed is directed downward.

At the point of transition between the compression and the suction stroke (point E in Figure 8-B) the bed will suddenly become compact. The above mentioned "consolidation trickling" can then only take place to a limited extent. It is true that the fine ore and sand grains might still trickle, but during the compression stroke a fair number of these grains have already been lost.

Experience shows that the coarse ore grains can penetrate only with difficulty through the bed, and since the entire bed undergoes a horizontal shift towards the tailing end, there is a possibility that the heavy ore grains will also get lost in the tailing. A further disadvantage is that the bed as a whole is rather compact, and this impedes the horizontal transport of the feed over the jig.

These problems can be partially overcome by the use of hutch water as indicated earlier. This involves the addition of a constant quantity of water to each cell, thereby creating a constant upward flow through the bed. The effects of this constant upward flow on the varying flow caused by the plunger movement, as shown in Figure 8-B, can be superposed as shown graphically in Figure 8-C.

The suction is reduced by the addition of hutch water and also lasts a shorter time; by adding a large quantity of hutch water the suction may even be entirely eliminated. The coarse ore now penetrates more easily through the bed, since the latter has been loosened by the backwater, and the horizontal transport of the feed over the jig also progresses more satisfactorily.

In the case of jigs with eccentric drive, the addition of backwater will give good results with coarse ore. However, the losses of fine ore will increase, partly because of the long duration of the compression stroke, and partly because the added backwater increases the speed of the top flow.

IHC Jig with Hydraulic Drive that Produces a Saw-Tooth Movement

IHC Holland and the Mineral Technological Institute (MIT), IHC Holland's own laboratory, have been seeking to develop a modified plunger movement that would cause not only the coarse ore, but also the fine ore to be removed.

They have proceeded from the assumption that, for the jig to work more efficiently, it is desirable to utilize the phase of initial acceleration and the phase of consolidated trickling, while the phase of hindered settling should be suppressed as far as possible. In order to prevent loss of fine ore during the upward stroke, the aim has been to produce a strong upward stroke of short duration.

The ideal movement would therefore be saw-tooth shaped as represented in Figure 9-A, and the related flow/speed diagram shown in Figure 9-B.

As shown in Figure 9-B, during the short period B-C, the strong upward flow brings the light bed into motion as one unit.

By reason of inertia, the grains remain pressed together so that they do not move in relation to one another but are simply lifted.

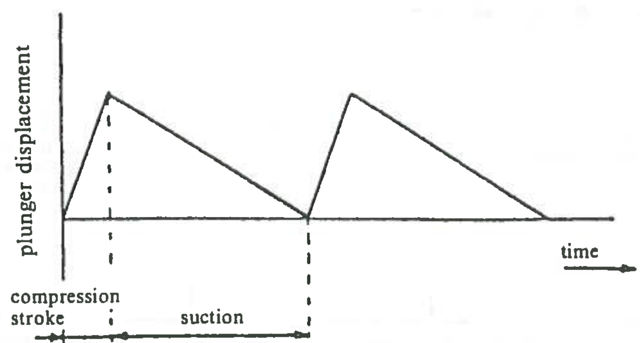


FIGURE 9A: IHC saw-tooth wave form.

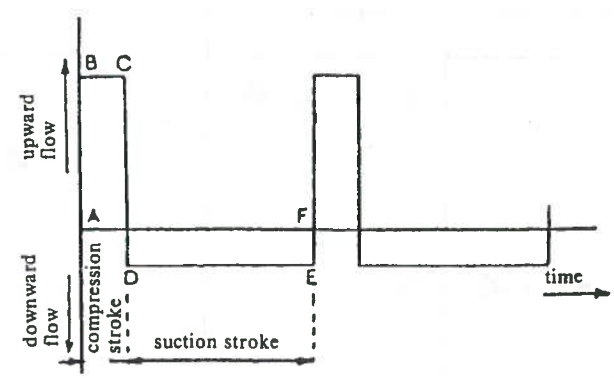


FIGURE 9B: Velocity/time diagram for saw-tooth wave form.

At C the upward flow suddenly stops and initial acceleration in combination with a certain hindered settling occurs. It should be noted that, since the flow breaks off abruptly, this process can only last a short time. The favorable effect of the initial acceleration, however, is far greater here than in the case of the harmonic-shaped plunger movement.

Because the suction is very weak, the bed does not become compact during the long period D-E. Therefore the ore grains have ample opportunity to trickle and the horizontal transport over the bed is not hampered. The reasons why back (hutch) water has to be added where the plunger has eccentric drive no longer apply in the case of the IHC plunger movement, since the suction over a long period is very weak.

On account of inertia phenomena, the diagrams in Figures 9-A and 9-B cannot be completely realized, but they can be approached.

The numerous experiments revealed that the separation of fine ore can indeed be considerably improved by the application of the saw-tooth shaped plunger movement.

The diagrams in Figures 10 and 11 show the movement of the various grains during one complete jig cycle for a jig with the normal eccentric drive and added backwater as compared with a jig with the IHC saw-tooth drive characteristic.

Combining the radial jig with the IHC saw-tooth movement, where the cross flow velocity over the jig bed is

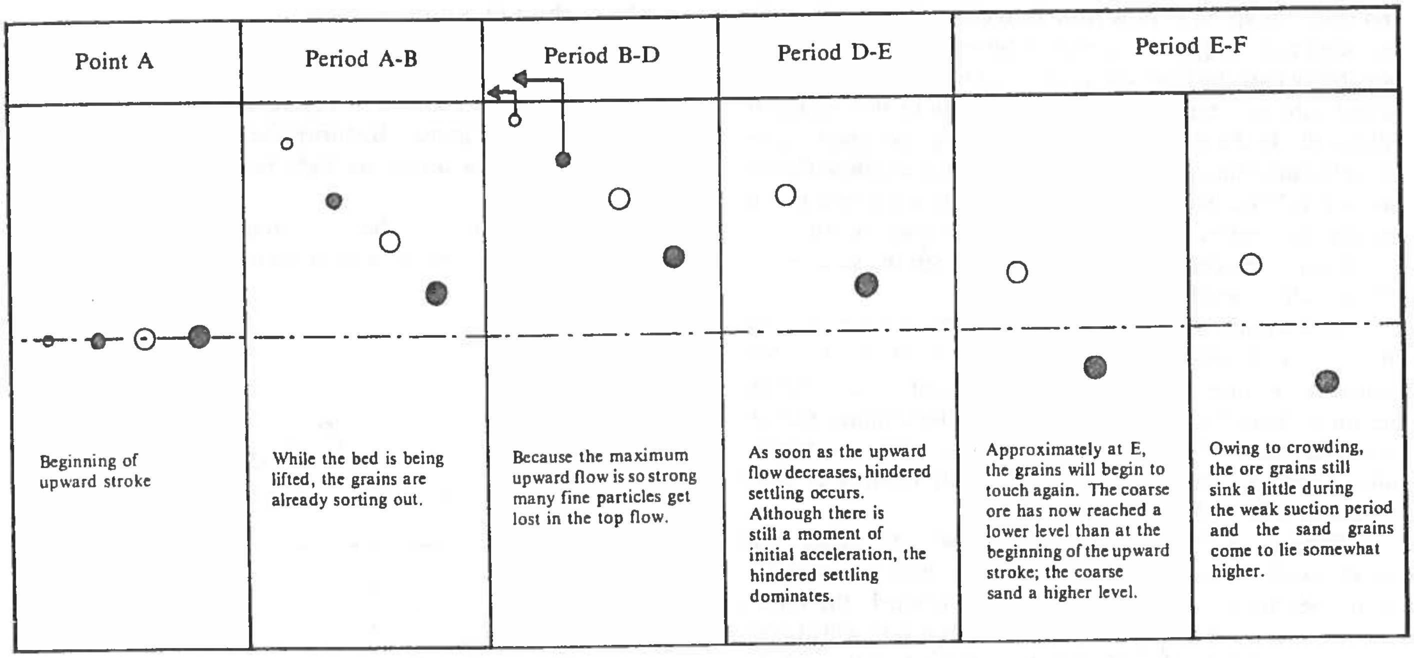


FIGURE 10: Pictorial representation of harmonic wave form jiggIng cycle.

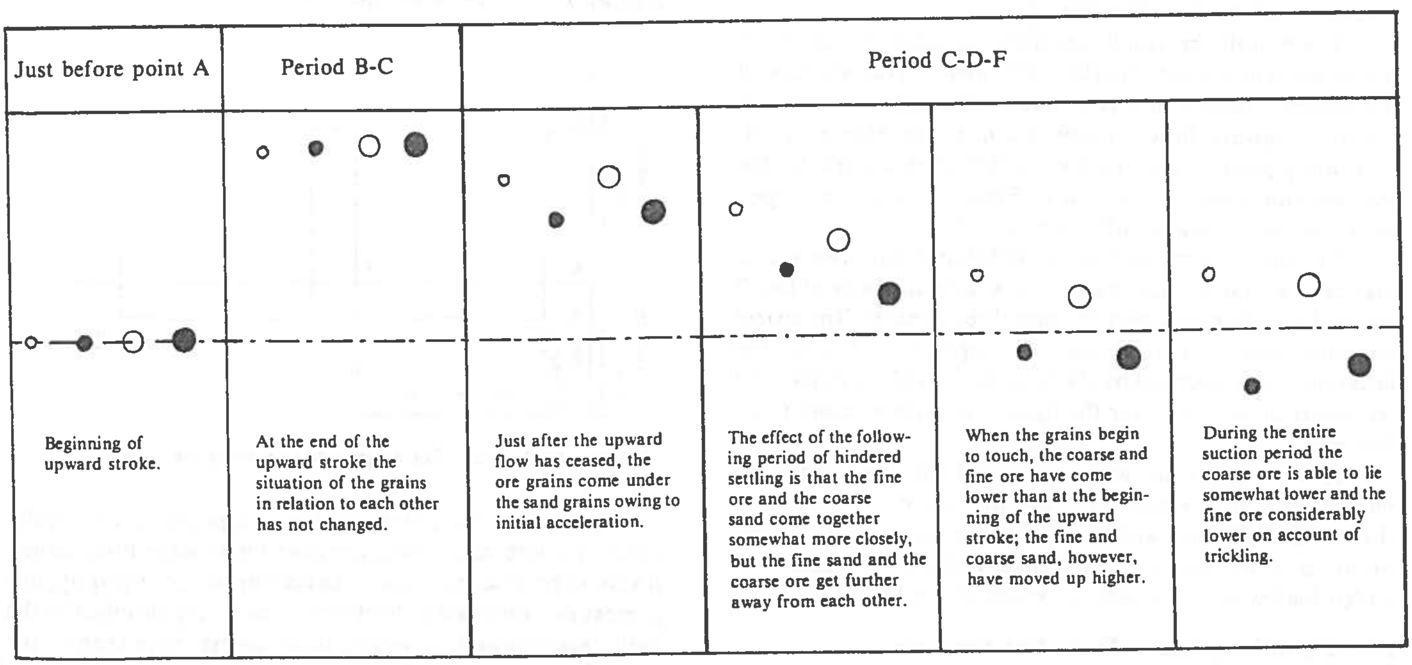


FIGURE 11: Pictorial representation of IHC saw-tooth characteristic jigging cycle.

decelerated, offers the possibility of optimal recovery of the valuable minerals over the entire size range and the ability to cope with large throughputs in one jig unit.

RADIAL JIG PERFORMANCE RESULTS

A known amount of material was sterilized and then known amounts of cassiterite reintroduced. Size ranges were between 50 mesh and 300 minus. The total recovery was compared with the known amount introduced into the test material. Although the percentage of losses was progressively larger in the finer mesh sizes, the overall recovery was in excess of 96 percent. The same test run on mechanical rectangular jig systems resulted in an 86 percent recovery. Either unit would be far superior to even the best sluicing system which would probably have losses in this size range of approximately 60 percent. However, remember that these tests were performed on cassiterite with a specific gravity of seven, not gold - our primary mineral, with a specific gravity of 19. Figure 12 compares the efficiency of jigs with conventional sluice boxes.

PLANT DESIGN

After examining and studying the various systems we took the jig concept and adapted it to our operational needs. First, we decided that the hydraulically driven jigs would serve much more efficiently as the primary collection units and that mechanical jigs would serve after the ore had been locked into the system, as secondary and tertiary collection units. We also recognized that in Alaska there are certain restrictions regarding the transport of a 30-40 foot load without dismantling. As a result it was necessary for us to design something that was compact and could be trucked. Consequently, we took their technology and our design and mated the two. The result was a unit designed according to the flow sheet shown in Figure 13.

THE GOLD DUST CREEK OPERATION

The plant we had constructed to meet the needs of our gold placer operation on Gold Dust Creek was built by IHC in Holland. It required eight months (seven months for construction and one month for shipping) from the time we placed the order for the unit to be delivered in Fairbanks. Figure 14 shows the plant as it was offloaded from the rail car in Fairbanks. As shown, it arrived in several pieces which would conform to the requirements for road transport. A

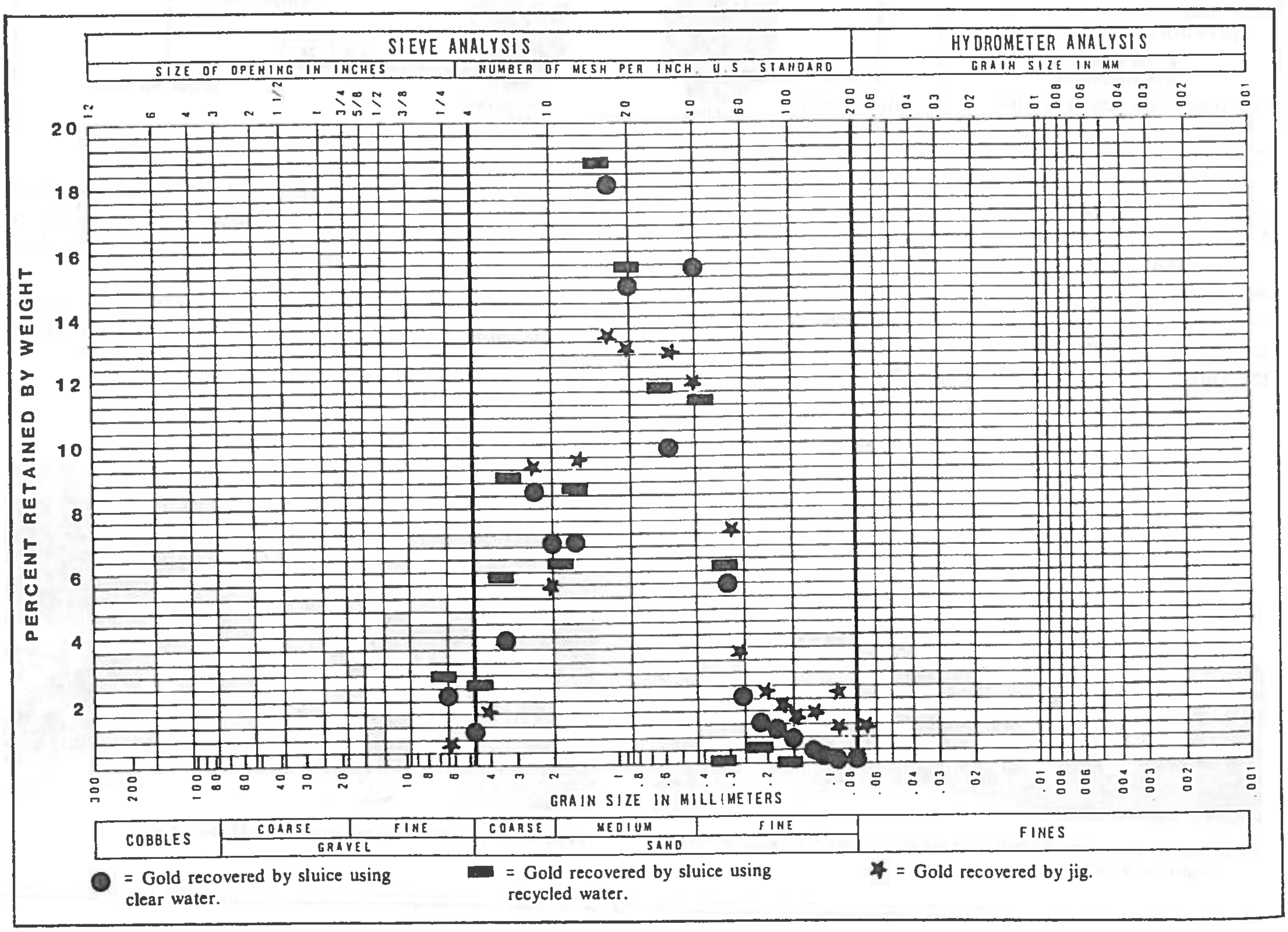
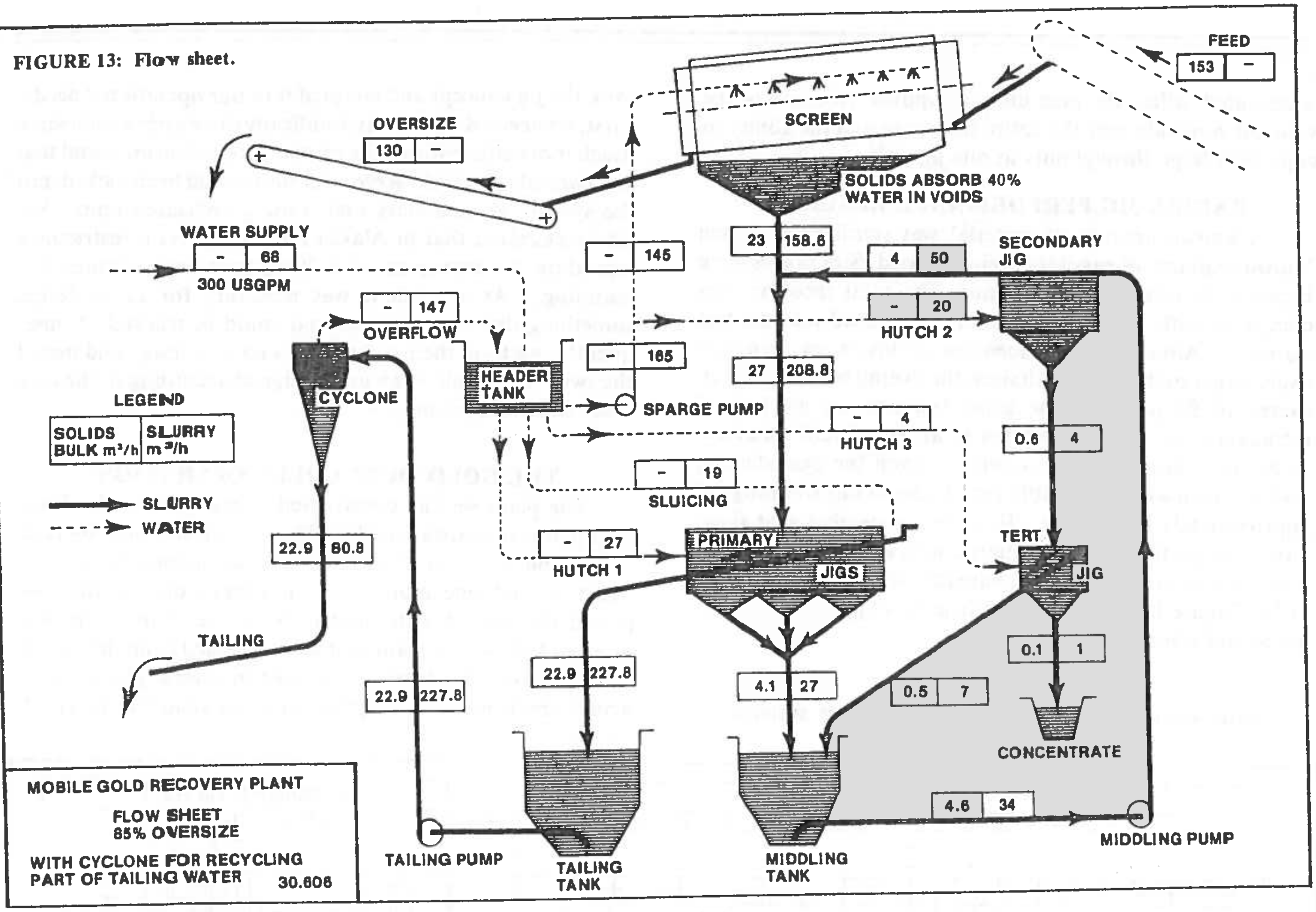


FIGURE 12: Efficiency comparison between jig and conventional sluice.

FIGURE 13: Flow sheet.



point of interest is that these pieces were small enough to be transported in certain types of large aircraft.

Upon arrival at Gold Dust Creek, as shown in Figure 15, we began putting the plant together. It fitted together with surprising ease, requiring five people and a crane - only twenty-six hours from start to finish. Using a forty-ton crane to swing the various components into place proved to be extremely efficient. Each component was machined and pinned which permitted bolting and piping to be assembled quite rapidly. In addition, the electrical harnesses were all plug-in types which permitted quick and simple hook-up.

Figure 16 shows the grizzly as it is moved into the trommel mouth. The coarse material was separated at this area.

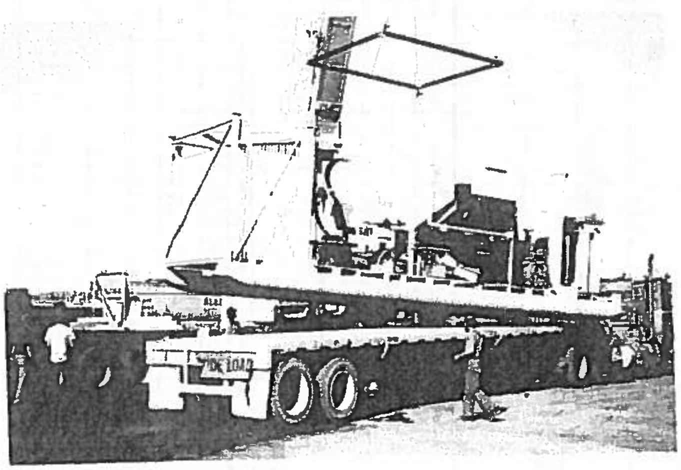


FIGURE 14: Transferring plant from rail car to truck for transportation to Gold Dust Creek.

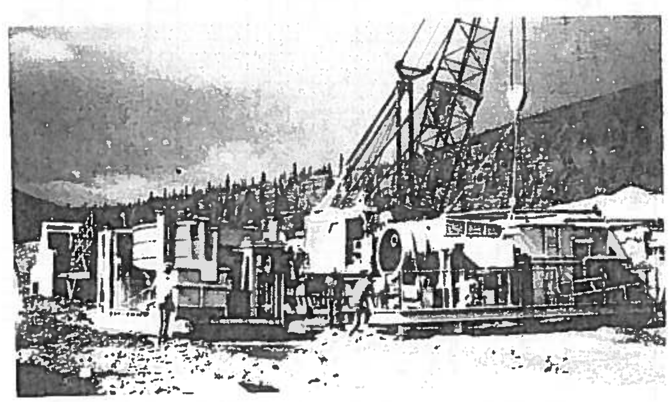


FIGURE 15: Assembling the plant at Gold Dust Creek.

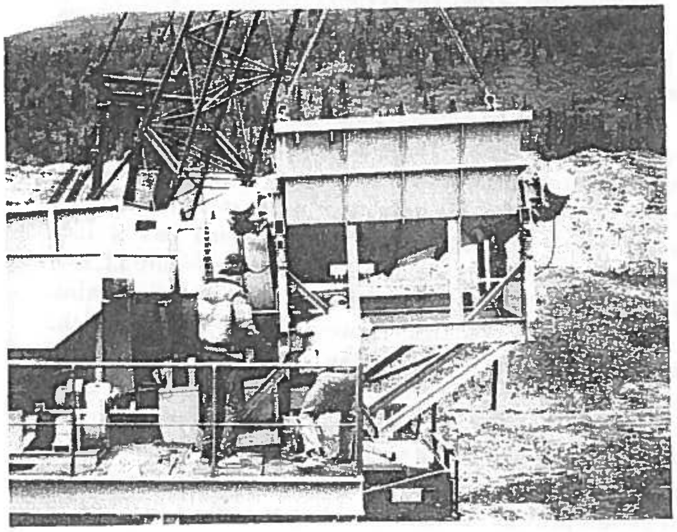


FIGURE 16: Coupling the Mogensen grizzly to the plant.

Rock Flow

To feed this plant we designed and built a track feeder. It consisted of a fold-out conveyor and track. The track unit itself is four feet wide and 14 feet long. The feeding capacity of this particular component is remarkable. It is equipped with a variable speed motor which, of course, provides easy adjustment of the feed rate. It is possible that if the speed were cranked up, that the unit would easily deliver between eight and ten thousand yards per hour. We had absolutely no problem with this unit. To sum it up, in terms of efficiency and reliability it surpassed expectations.

From the feed conveyor, the material is deposited on the Mogensen grizzly, as shown in Figure 17. The Mogensen actually retards the rock's flow across the sizing bars for as long as 20 seconds. This, of course, permits a longer period under water and, consequently, a more thorough wash.

When we first designed the Mogensen grizzly it was set for 12 inch minus. However, we didn't have the proper flow angle in the receiving hopper beneath the grizzly. As a consequence we had to plate the chute with UHMW plastic to reduce the friction. Then we added more bars to the Morgenson thereby reducing its classification to six inch minus.

From the hopper the six inch minus material passes across a short four foot run of riffles in an attempt to capture the coarse gold. However, this has proved to be inefficient and we are experiencing some coarse gold loss. Measures are under consideration to correct this particular deficiency.

From the riffle run the material enters the mouth of the trommel. Figure 18 shows a view from the top of the trommel. Note perforations in the lower half. The trommel itself is a 5½ foot diameter (54 inch i.d.), blinded, self-scrubber type. It is 26 feet long, 12 feet blinded and 14 feet open with ½ inch orifices. As shown in Figure 19, the blinded portion is the only section that has lifters. Also visible is an eleven inch retainer ring that serves to momentarily retard the flow of material thereby permitting more thorough scrubbing. The blinded section is fabricated from 1⅛ inch manganese cast plates which are very durable. The retainer

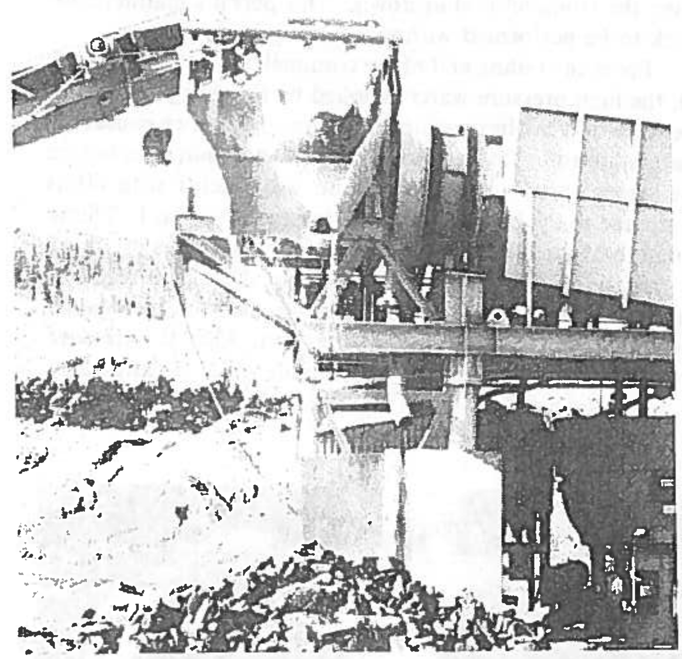


FIGURE 17: The Mogensen grizzly.

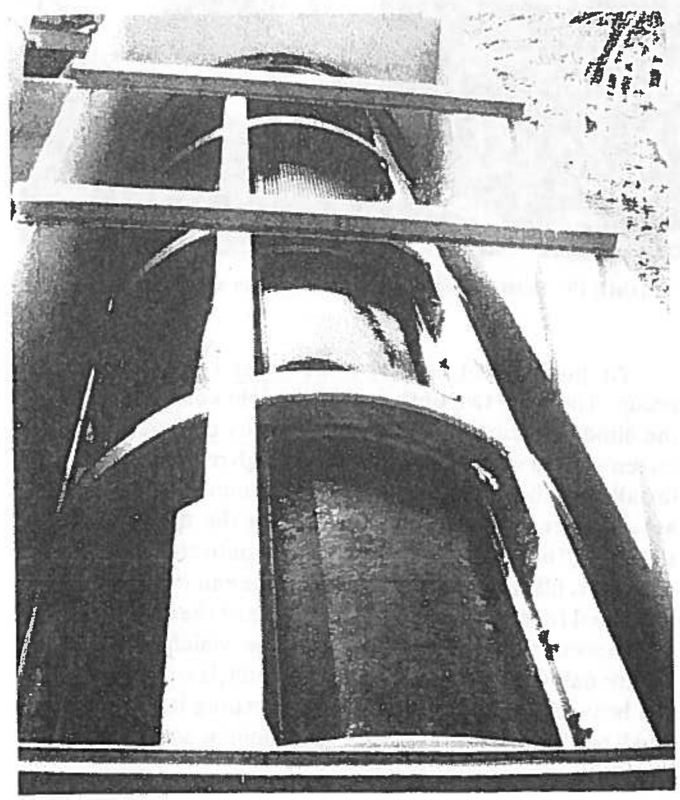


FIGURE 18: Trommel.

ring and the lifter bars are a hardened angle iron. Interestingly, there is a plastic coating on them that still exists after processing almost 8,000 yards of material. Needless to say, we have been fairly satisfied with how the section works.

The retainer is made in four sections and, as shown in Figure 19, the overlapping lips of the four sections meet each other. This was done so one section could let the water out when the trommel is shut down. This permits maintenance work to be performed without working in water.

From the tailing end of the trommel, as shown in Figure 20, the high pressure water supplied by the sparge system to the water jets can be seen directed at an angle which causes the material to roll. This is much more practical than lifter bars in the screen section because tests on wear factor with lifters compared to the blinded section is increased to 7 to 1. This is found to be directly related to the lower surface area provided by the punched manganese plate within the screen section. Another feature which increases the wear life of the trommel is that it can be driven in either direction. Also, the pressure nozzles on the sparge pipe are adjustable which permits them to point from one side to the other depending on which direction the trommel is turning.

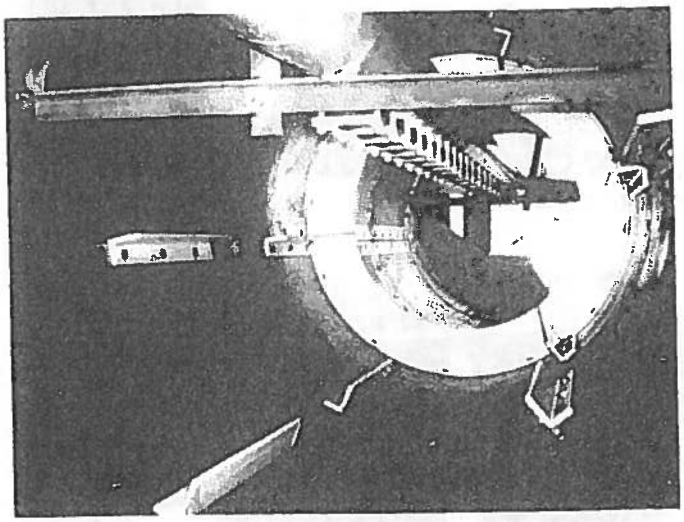


FIGURE 19: Retainer and lifters in blind section of trommel.

As pointed out earlier, some losses in coarse gold do occur. There are two options which might correct this. First, the blinded section could be shortened by inserting a ¾ inch screen section or second, a three inch high retainer ring can be installed at the discharge end of the trommel. Because there are no lifter bars in the screen section the material simply slides out the end of the trommel and onto the tailings belt. However, placing the retainer ring at the end would serve two beneficial functions. First, it would retard the material in the screen section for a longer period of time which would permit additional washing and sizing and second, it would hold back the heavier and larger concentrate, keeping it locked in the small bed load of the trommel. This could be easily checked at the end of each shift to determine if, indeed, the ½ inch plus concentrate is being captured.

As indicated earlier, the ½ inch plus material passes from the trommel onto a tailing conveyor. Simultaneously, the ½ inch minus material passes through the trommel screen into a chute leading to the primary jigs.

The primary jig system consists of three hydraulically driven trapezoidal cells. The ½ inch minus feed along with the wash water from the trommel is split between the three cells at the head of the jigs. The only other water is the hutch water that is forced up through the screen which has ⅜ inch openings. The ragging material, as shown in Figure 21, is ½ inch taconite pellets. Beneath the taconite we used tire chains. Originally they worked fine. However, it was found that if the system does not run 24 hours a day the chains have a habit of rusting and cementing the bed together. Since this plant does not run 24 hours a day the chains have been removed. Furthermore, it was found that the presence of large concentrations of high specific gravity minerals such as rutile, galena and pyrites provide sufficient bedding to replace that lost by removing the chain.

Clearly, the ½ inch minus to ⅜ inch plus gold is locked within the bedding of the jig while the ⅜ inch minus material passes through the jig screen into the hutch and subsequently falls to the bottom where it passes through a spigot and into a

FIGURE 20: High pressure water supplied by sparge pump to wash gravels as they pass through the trommel.

middling tank. Each of the three jig cells have independent diaphrams and hydraulic rams which are driven by a single hydraulic manifold.

Figure 22 shows the bottom of the jigs and the hoses that transfer the material to the middling tank. An important feature is the presence of a restrictor within the spigot which regulates the proper water volume passing through the bottom of the jig.

The product from this end is picked up by the middling tank and pumped up by a middling pump to the secondary jig. Figure 23 shows the receiving end of the secondary jig. Like the primary jig, it is also trapezoidal in shape. The ragging material which lays on a 5/16 inch screen consists of ½ inch taconite pellets with additional bedding supplied by the rutile, galena and pyrite in the concentrate. The tailings are directed

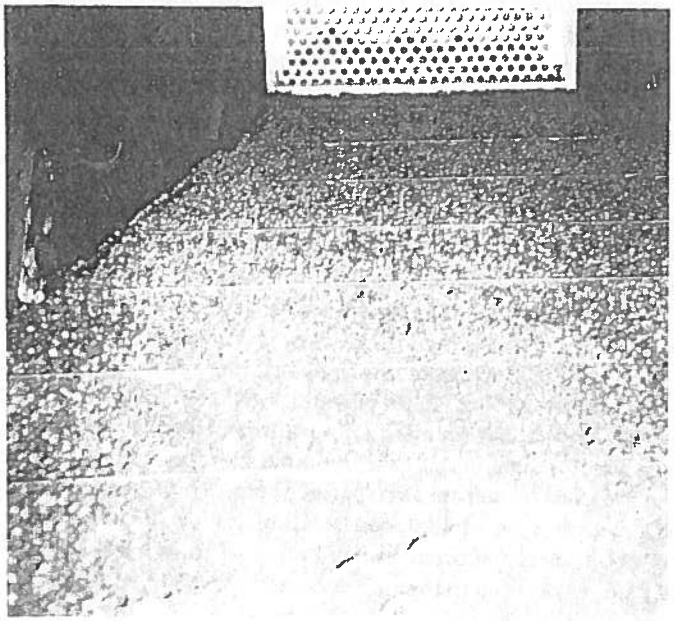


FIGURE 21: Taconite pellets and chain used as bedding material in primary jig.

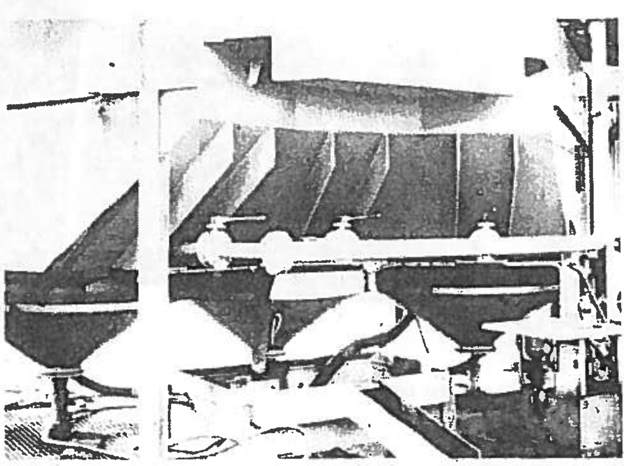


FIGURE 22: Diaphragms and concentrate transmission lines to middling tank.

back into the trommel chute and reenter the primaries. In comparison, the primaries have much better recoveries than the secondary. At the moment, the primaries are set to capture all the materials of specific gravity 6 or more while the secondary is set to capture material of specific gravity 12 or more.

The tertiary jig receives the material from the hutch bottom of the secondary jig. Unlike the ragging in the primaries and secondary jig, which is taconite pellets, the ragging in the tertiary jig is ¼ inch stainless steel ball bearings which lie upon a 1/16 inch screen. The tails out of the tertiary jig are carried out by the cross-flow into a funnel and transported back to the middling tank where they reenter the circuit to the secondary jig. Clearly, the tails from the tertiary jig are locked into the circuit thereby preventing any gold loss.

The final product is drawn off the bottom of the tertiary jig hutch into a five gallon bucket. Presently the plant is producing about two quarts of minus 1/16 inch gold bearing concentrate per 12 hour shift. Although it is not necessary to draw this product off until the end of the shift, it is recommended that it be checked every few hours. An additional benefit of an immediate product from the mined gravels is that it can be monitored more often to determine the location of the pay streak and the tenor of the ground adjacent to it.

The product captured on the jig screens surface is cleaned up only at the end of the season. The need for more frequent

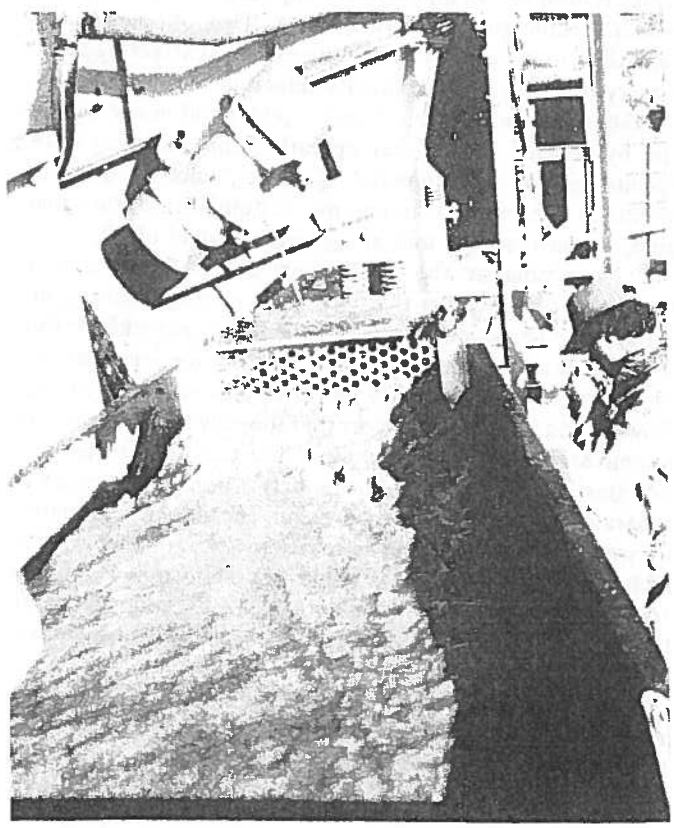


FIGURE 23: Secondary jig.

cleanups is not necessary because 70 percent of the total product is obtained from the concentrates processed through the tertiary jig. This would, of course, vary from site to site. This product covers all the immediate operational expenses. The size of the remaining 30 percent is directly related to the openings within the respective jig classification screens. Keeping in mind that the trommel only passes ½ inch minus material, the primary jig then only passes ⅜ inch minus, the secondary jig passes about 5/16 inch minus and the tertiary jig passes 1/16 inch minus material. It is clear that the remaining 30 percent of the gold consists of that portion locked within the bedding of the three jigs and coarse gold trapped in the short four-foot run of riffles at the head of the trommel.

This plant was designed to process up to 200 yards per hour at 15 percent ½ inch minus ground. In reality our ground was 60 percent ½ inch minus. At first we thought we would have to further reduce our trommeled reductions. Fortunately that wasn't the case because originally the plant was designed for minerals with a specific gravity of 4. Raising the specific gravity of the plant to 10 not only raised the capacity of the jigs but also permitted us to increase our input. At present we are running 180 yards per hour with this plant. And we are doing this with 60 percent of the material at less than ½ inch minus. Consequently, the type of jig in line after the initial hydraulic jig doesn't matter since the other styles have comparable recovery levels. Further, other than the tailings from the Mogensen and trommel, the only other tailings from the plant are the tailings from the primary jig.

This plant has the ability to recover a variety of ore minerals other than gold. This is a feature of the primary jig which is designed so that it can be adjusted to collect minerals with a specific gravity of as low as 4. This, of course, lends versatility to the unit by permitting its use in extracting a wide variety of lower specific gravity minerals such as scheelite, cinnabar and cassiterite. Clearly, gold could easily become the by-product of a placer operation concentrating lower specific gravity ore minerals. However, since we have not identified ore minerals in commercial quantities other than gold, we have set the unit at a specific gravity of 10.

To summarize the tailing flow and its benefits, it is important to remember that the tails from the tertiary jig are returned to the secondary jig and therefore are locked within the system and further, that the tails from the secondary jig are returned to the primary jig for re-processing. With this flow system in mind, it is clear that the only tails from the jig system are from the primary jig. These tails are directed to a dewatering cyclone where a major portion of the water is separated and returned to the system. The tails are eliminated by way of a tails chute. The remaining tails include the six inch minus, ½ inch plus material from the trommel which is transferred to the rear of the plant by a stacking conveyor and the six inch plus material which is separated by the Mogensen grizzly.

Although all the tails are currently being removed mechanically by loader, the separation of the tails into the three sizes permits advantages not available using the standard sluicing methods. The ½ inch minus material can be pumped. Although this method of removal is still in the development stage, once initiated it will be much more economical than mechanical removal. Perhaps one of the most noticeable immediate benefits of splitting the tails into the three distinct size ranges is that the larger material is stacked in elongate, strip-like tailing piles that serve as tailing filters. Because this plant uses such little water it will percolate through these filters and even without the dewatering cyclone running, there will be little or no visible water discharge.

Water Flow

Fresh water is supplied to the plant by a six inch Gorman and Rupp pump, driven by a 50 hp. electric motor which supplies 1,700 gallons per minute. This pumping system is larger than necessary. A three inch pump with a 20 hp. motor would supply sufficient water to run the entire plant.

The water is pumped to the header tank (see Figure 24). In the bottom of the center cell of the header tank is a fishing float which is attached to a flapper valve that serves to regulate the inflow of fresh water. The water from the center cell will flow into the lower cell on the right of Figure 24 which is the source of water for the sparge pump. The holes in the bottom of the center tank lead to and supply the hutch water for both the primary jigs and the tertiary jig. Because the secondary jig is on the same level as the header tank it is fed with pressure off the sparge system. As a result, a more sensitive water pressure control is required. In way of explanation, hutch water is critical where flow is concerned because if fluctuations of pressure occur there will be a corresponding fluctuation in the jig bed. As a consequence, it is recommended that all hutch water should be obtained from gravity flow which supplies a constant pressure. Unfortunately, plant design, in this case, required that the secondary jig use pressure flow from the sparge pump. However, restrictors within a valve that controls hutch flow are sensitive enough to permit a broad enough range of adjustment that if any surges occur in the sparge system it doesn't have a significant effect on the jig.

The cell on the left is the overflow. Any time the water

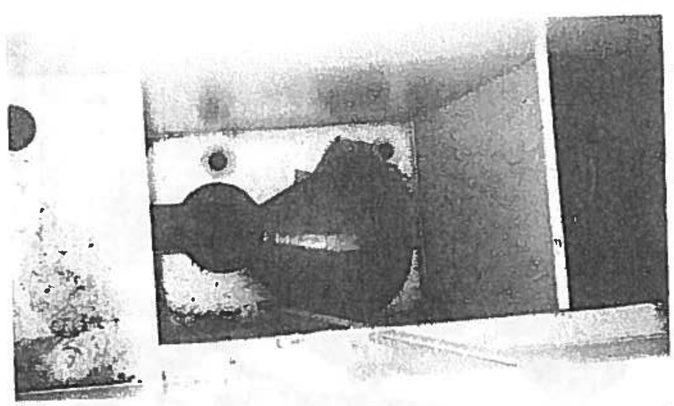


FIGURE 24: Header tank.

did get this high it would go out the drain.

The sparge pump supplies all the pressure water systems in the plant including the spray bars on the Mogensen grizzly, trommel and secondary jig hutch water.

As indicated, the hutch water for the primary and tertiary jigs is obtained by gravity flow from the center cell of the header tank. This provides about ten feet of head. To make the water flow adjustable would require about 14 feet of head. However, by running the system wide open a suitable but soft pressure flow is obtained which is ideal for this plant's operation.

The water flow from the jigs follows the same path as the tailings. Water from the tertiary is directed back to the secondary jig. Water from the secondary tailings is directed back to the trommel chute where it reenters the primary jig system, and tailings water and tailings from the primary jig are discharged by the launder chute or processed by the hydrocyclone (Figure 25).

FIGURE 25: Launder chute - note hydrocyclone on second deck.

When using the hydrocyclone the material that would normally be discharged through the launder chute would be routed through a trap door at the head of the chute and into a tank. All the tails from the primary jig including the gravel, sand and water are then pumped to the cyclone on the second deck. The cyclone separates a portion of the water and 100 mesh minus material which is returned by an eight inch pipe to the header tank. The recycling of the water to the header tank causes the float to stay up, thereby restricting the amount of fresh water input by about half.

It should be pointed out that this plant is designed to operate on a certain amount of solids and turbidity. Using a settling pond as a source of recycled water is not as efficient. The problem is that there is no way to control the amount of silt taken from the pond and therefore it builds up in the system and the recoveries decline. In contrast, the cyclone regulates the amount of solids and turbidity to the level appropriate for maximum recovery within the system. This eliminates the need for excessive monitoring.

CHARACTER OF THE PRODUCT

The plant's high level of efficiency in the capture of fine gold revealed some rather interesting results not evident while using the conventional sluicing system. In addition to the rather typical, easily recognized gold with the proper color, a brownish gray residue which accompanied the pyrite began to appear, predominantly in the finer fractions below 120 mesh. The material had a relatively low specific gravity, compared to typical gold, of about 9. Furthermore, it would not amalgamate. However, because there was a considerable amount of this material accompanying the concentrate it generated enough interest to have it analyzed. It was shocking to find that it assayed 98 percent gold. What was so puzzling was that there were a number of characteristics that just didn't fit. One of the most inconsistent characteristics was the high fineness: the more typical gold obtained from our previous sluicing operations had always run about 85 percent fine. Furthermore the brown or gray color rather than the typical yellow color, the low specific gravity of 9 rather than the higher specific gravity of 16 to 19, and the fact that it would not amalgamate were, at first, a puzzle. However, after careful examination under a microscope the mysteries were revealed. It was thought, at first, that there might be an oxide coating on the gold to prevent amalgamation and produce variation in color. However, examination with the microscope revealed that the gold particles exhibited a highly pitted, spongy character which did not reflect light as did the more typical gold and, therefore was responsible for the brown to gray color. Also, the rather porous nature of the particles would contribute to the lower specific gravity observed, in addition to resisting amalgamation.

Although attempts at separation by tabling and amalgamation were unsuccessful, the material was removed successfully using a Super Panner. It separated out just in front of the magnetite and was followed by the more typical gold extracted from Gold Dust Creek.

Some of the more noticeable characteristics observed in the various size ranges examined include the following:

In the 120 to 160 mesh range in addition to the bright colored gold there was an increase in porous brown-gray particles. There was a larger percentage of elongated and wire gold, over the finer material, in this fraction. Some waste material accompanied the gold.

In the 160 to 180 mesh range there was a notable increase in porous gold characterized by the accompanying brown-gray color. There was a noticeable decrease in waste and foreign material. Also of interest was an increase of bright silver crystals in this fraction.

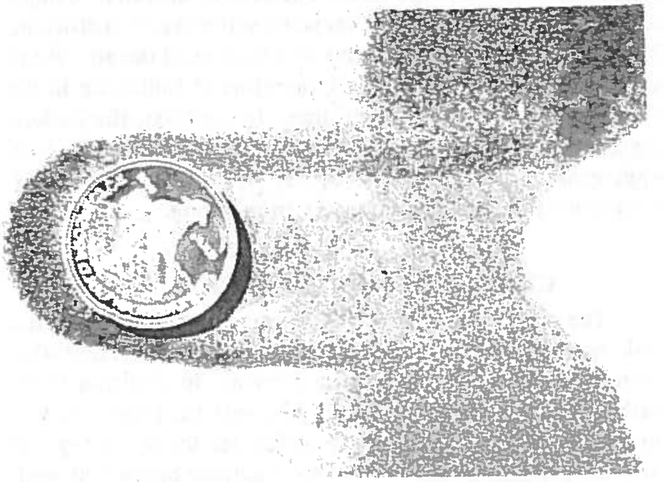


FIGURE 26: 120 mesh minus gold.

In the 180 to 200 mesh fraction, there were amounts of spongy gold comparable to that observed in the 160 to 180 mesh range. Although some crystals were present, the general character seemed to be much more rounded. There also seemed to be a greater percentage of silver crystals in this size range. Foreign waste material decreased.

In the 200 to 250 mesh fraction the high percentage of porous gold continued. Some crystals were observed although the general shape was toward a more rounded gold particle. Waste material decreased, although some bright green crystals resembling emeralds were present. Also there were some other crystals which exhibited a deep indigo color similar to a piece of heat-treated metal.

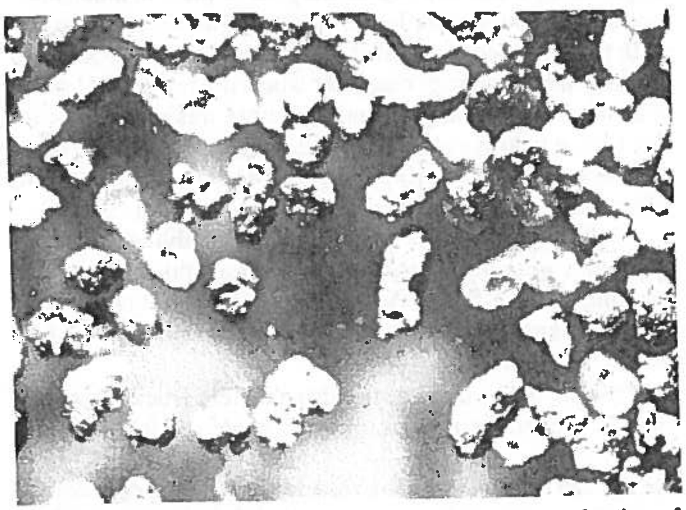


FIGURE 27: Pourous, brown-gray gold found in finer fraction of concentrates.

In the 250 to 400 range there was very little waste; it was almost all gold. It consisted of all rounded spongy particles with a noticeable absence of any crystalline structure or wire gold.

The minus 400 mesh material was comparable to the 250 to 400 mesh material.

To summarize the characteristics of the gold finer than 120 mesh: in addition to the bright colored gold typical of Gold Dust Creek some of the gold took on a color change from brown to gray; particle shape went from an elongated, sometimes crystalline wire gold down to a granular bead with a decrease in size; with a decrease in size there was a corresponding increase in porous gold exhibiting a lower specific gravity.

CONCLUSION

In conclusion, the efficiency of the plant has exceeded our initial expectations. Several aspects of processing and recovery are of exceptional value:

1) Because of the adjustability of the primary jigs to accommodate minerals of widely ranging specific gravity it was possible, when set to extract gold, to process a much larger quantity of ½ inch minus material than originally expected.
2) The lower water volume required by the jig system combined with recycling through the hydrocyclone has resulted in a significant decrease in water use.
3) By separating the tailings into three distinct size fractions (plus six inch, minus six inch-plus ½ inch, and minus ½ inch) it is possible to use the larger two fractions to construct tailing filters which improve discharge water quality.
4) The extraordinary efficiency of the jig system in fine gold recovery and concentrate extraction has revealed the presence of fine gold particles of lower specific gravity not recognized in our previous, rather traditional sluicing operation.

In general the craftsmanship and performance of the plant are superb and the recovery is even more remarkable than expected. As indicated, we are well satisfied with the plant and the service provided by IHC.

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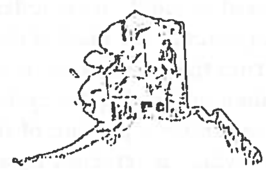


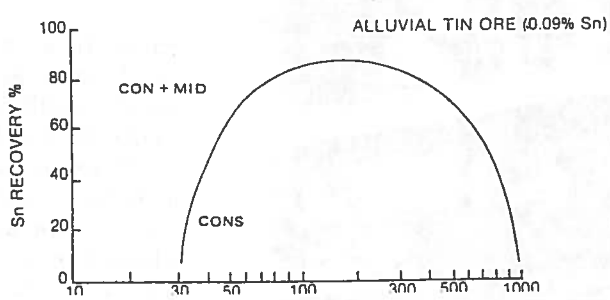
Considerations for Applying Reichert Mark VII Spirals in Alaska

Kelly Dolphin, Owner
The Miners' Company

Spirals have been used for gravity concentration for many years. The Reichert spirals were initially developed for the alluvial tin industries of Australia and Southeast Asia (Figure 1). Each of over twenty different Reichert spiral designs is contoured to separate materials of a given specific gravity range. The Mark VII has been on the market since 1982 and is designed to recover tin with specific gravity of 6 or 7. On the opposite end of the spectrum, the Mark X separates heavier waste shale from coal.

The crossectional shape of the Mark VII spiral changes from top to bottom. Material fed to the top of the spiral is progressively concentrated over its full length. Fine grained, high specific gravity solids are concentrated at the inside of the spiral's turns. The concentrate is separated from the middlings, which may be recirculated, and the tailings, by splitters at the bottom of the spiral. Depending upon the nature of the feed material, one usually obtains concentration ratios of 50:1 to 100:1.

The Mark VII Reichert spiral is a lightweight and low cost fiberglass - plastic unit. It has no moving parts. The spiral is designed to be used 24 hours a day, 365 days per year for many years before wearing out. A single spiral may be mounted on the support column for test work, or two or three identical spirals mounted on the



same column for production. A full scale production operation may require eight or more triple-start spirals.

The Mark VII spiral is fed a 40% (w/w) solids slurry of minus 10 mesh (or finer) feed (Figure 2). It achieves high efficiency when recovering gold between 40 and 350 mesh in size (350 to 40 microns). A four part split is made at the bottom of the spiral. Concentrates, which can be observed as a thin dark line on the inside edge of the spiral, are separated for further processing. The middlings may be recycled and act as a buffer, preventing loss of fine gold should the feed surge or fluctuate. The tailings are discarded. A water split, low in solids, is either discarded or recycled.

Concentrates from the spiral are processed either on a conventional Wifley or Diester table or on the new Gemini table. Tabling is employed to make the difficult separation of the heavy minerals from the gold.

DESIGN DATA

HEAD FEED (PER START)	
Capacity	up to 3TPH solids depending on application
Pulp Density (w/w)	up to 60% solids
Size Range	0.03 - 2mm
Pulp Volume (max.)	5.0 m^3/hr
CONCENTRATE REMOVAL (PER START)	
Rate	up to 0.3TPH solids
Pulp Density	30-60% solids w/w.

NOTE: SINGLE, TWIN, TRIPLE STARTS AVAILABLE.

Figure 2. Plant design data for the Reichert Mark VII Spiral.

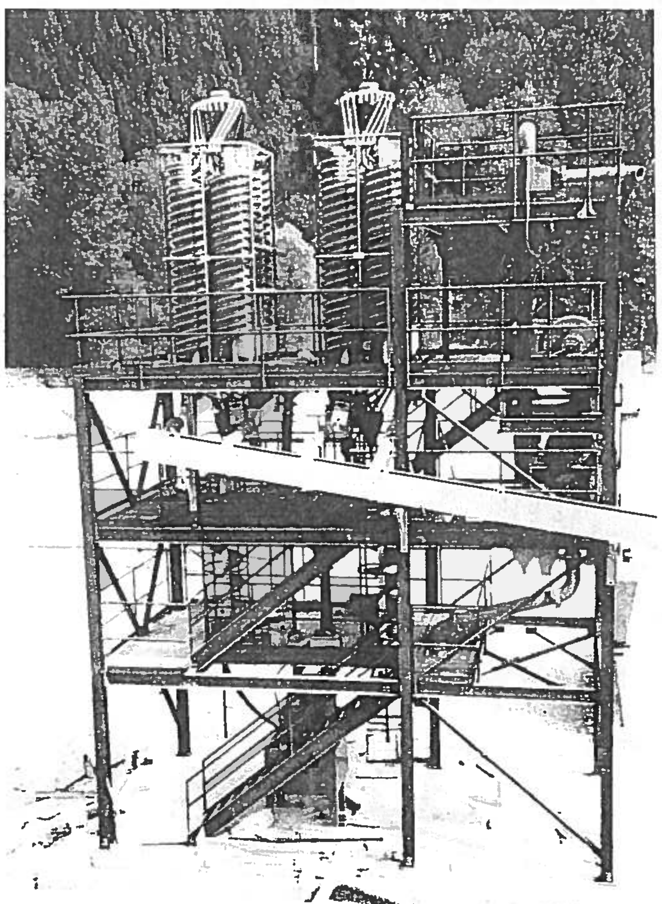


Figure 3. Spiral plant, Grass Valley, California.

A single spiral trough accepts a feed of up to three tons per hour (approximately two cubic yards per hour) of minus 10 mesh material. A triple start spiral can be fed up to nine tons, or six cubic yards, per hour.

Spirals are presently utilized in gravel pit operations in Arizona and California (Figures 3, 4 & 5). The Arizona operation feeds a 200 yd.^{3}/hr. plant with 50 ton dump trucks. The material is sized, with the minus 3/8" plus 20 mesh, passing across four double hutch jigs. The minus 20 mesh material is pumped through a 24" Linatex Hydrocyclone Separator then across a Dings double drum wet magnetic separator. The magnetics are stockpiled and the nonmagnetics are processed by eight triple start Reichert spirals for primary concentration, and 4 single start spirals for secondary concentration.

The gravel plant in California processes 200 tons/hour (130 yds.^{3}/hr.) of minus 3/8" material, which is screened to minus 16 mesh, dewatered in a hydrocyclone, the magnetics removed by a Dings wet

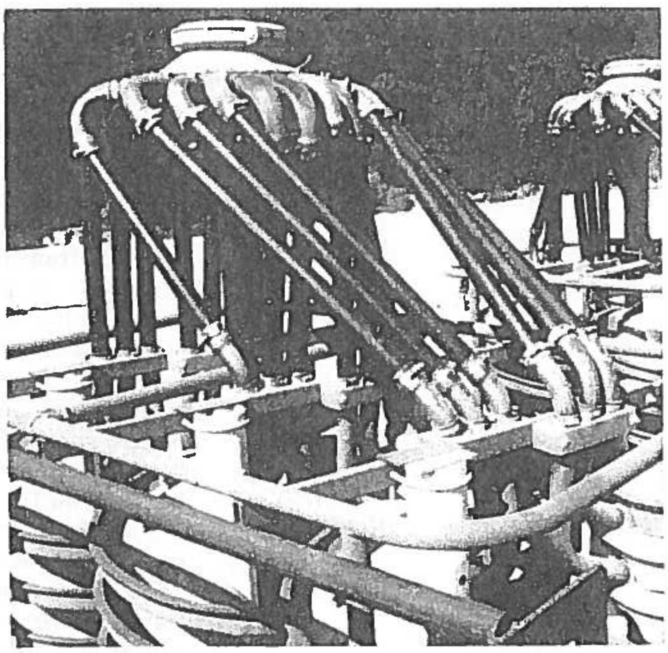


Figure 4. Pressurized manifolds feeding spirals. Grass Valley, California.

magnetic separator, and finally pumped to a bank of 12 triple start Reichert Spirals. Tailings will be sold as sand, middlings recycled through the spirals, and concentrates gravity fed to 2 single start Reichert spirals.

These are sophisticated, well engineered plants and may be good examples for setting up a new operation. Spirals are also adaptable as an add-on to existing sluice box operations (Figure 5). Prior to the design of a production system both lab testing and field testing of the spiral are essential to assure that the operation will pay for itself. The amount of fine gold, its size range, shape, and surface characteristics all play a part in this evaluation and vary considerably from one placer deposit to another.

To date, limited testing by the Miners' Company of the Mark VII spiral includes laboratory testing of placer samples and introductory on-site field work (Figure 6). Initial results of these tests were determined by fire assay. This method has proven unsatisfactory as it measures both free gold and gold locked in the minerals of the concentrates. Fire assay is also inappropriate for placer gold evaluation since it is very vulnerable to the "nugget effect," which in this case is produced by the coarser of the minus 10 mesh gold. The development

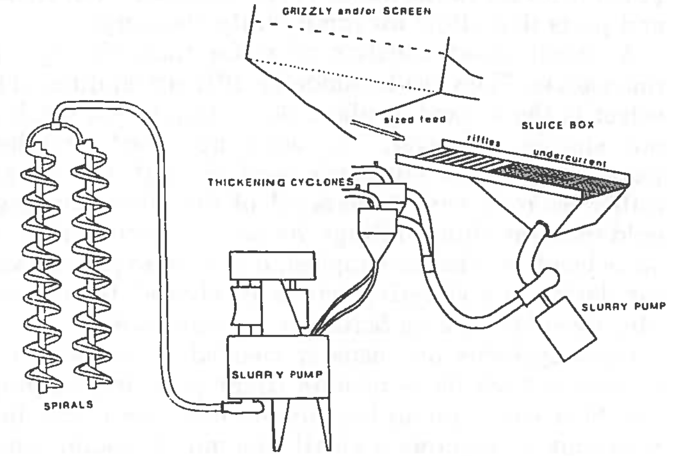


Figure 5. Equipment schematic for spiral processing of sluice box tailings.

and manufacture of the Gemeni Table by Wendell Rogers, (9725 W. 21st Ave., Lakewood, Colorado, 80215) should allow for a more accurate evaluation of all future spiral production testing (Figure 7).

Spirals, unlike sluices, are continuous production devices. They are not batch systems which need to be cleaned out at intervals. They have no moving parts other than the well proven slurry pumps. When used in conjunction with hydrocyclones, also well proven by the mining industry, up to 40 to 60% of the water in the system can be recycled. Spirals can be installed on a skid and moved easily from cut to cut during the mining season.

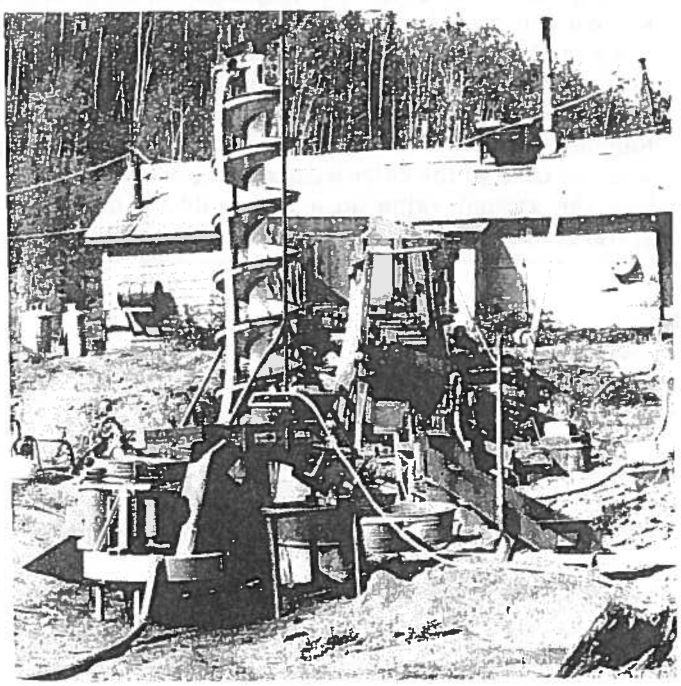


Figure 6. Portable spiral pilot plant being tested at Coal Creek, Alaska.

The gold lost by most sluice boxes (minus 60 mesh gold) can be recovered to a large degree by the addition of a spiral plant. If one already has a sluice, it may well be more cost effective to add a spiral system for secondary recovery than to discard the sluice and install an expensive jig plant. If one had both a sluice and a spiral plant and relocated to a placer deposit without an economic quantity of fine gold ($0.50 to $1.00 per cubic yard of minus 100 mesh gold) then one could simply set the spiral system off to the side (Figure 8).

In as much as any additional recovery system will require a substantial capital investment it is crucial that adequate testing precede the acquisition of any new system. In fact, very few placer deposits contain enough fine gold to warrant any additional expenditure on a recovery system. For most creeks an increase in the amount of material sluiced during the mining season will create the only cost effective increase in gold production.

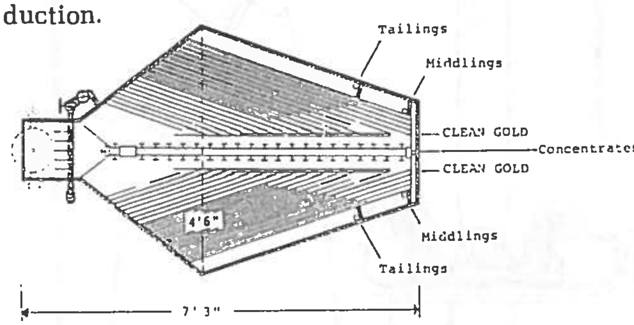


Figure 7. Plan schematic of the Gemini Table.

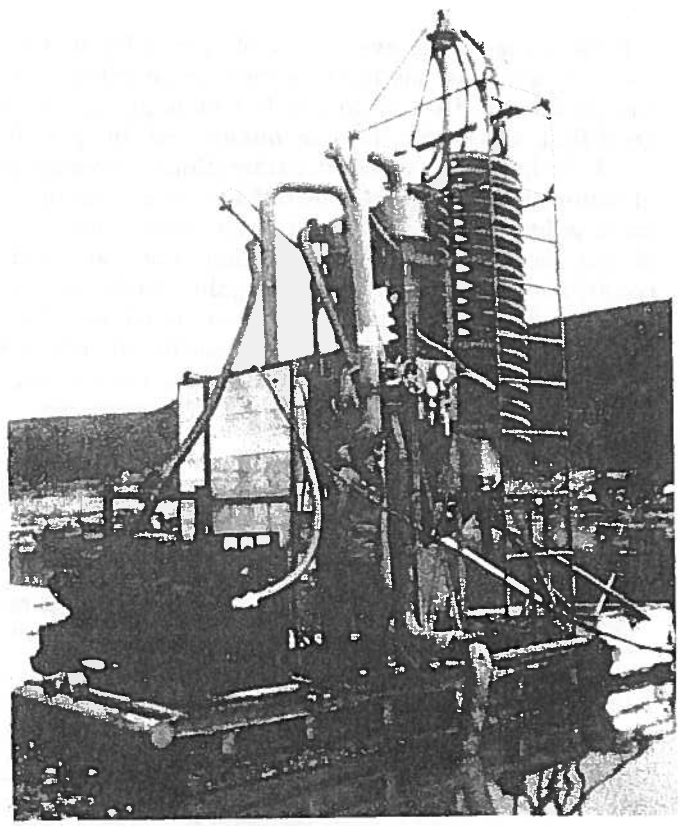


Figure 8. Portable skid mounted spiral plant in production on the Chena River, Alaska.

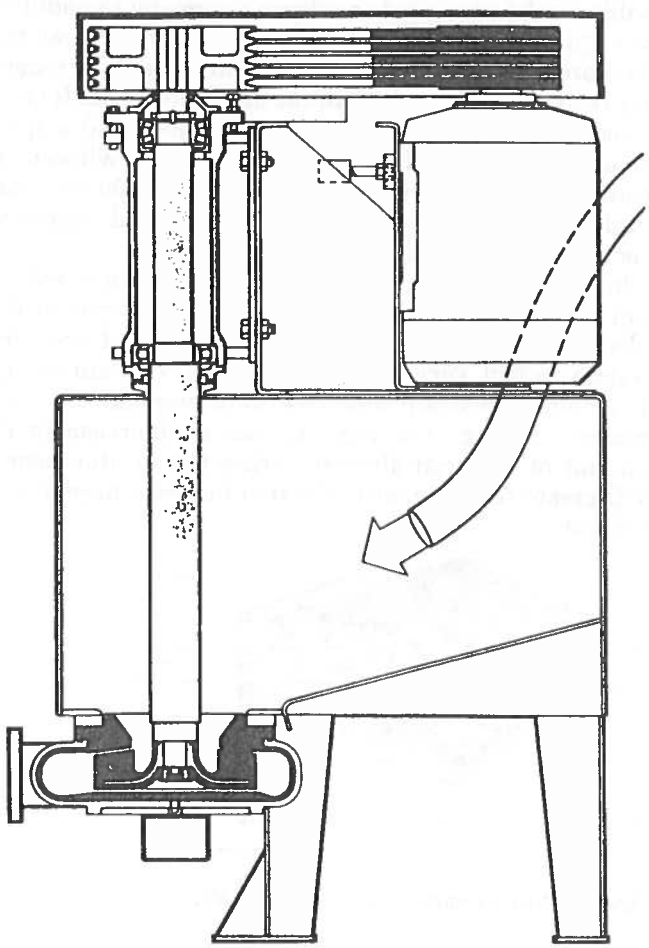


Figure 9. Sala Vertical Pump.

Experience in the evaluation of creeks for fine gold recovery systems has led to several conclusions. First, fine gold is not difficult to pan. In fact, panning is a very good first step to evaluate the quantity of fine gold in a creek. To be worth further consideration an average pan of material should contain 50 to 500 colors of minus 100 mesh gold; not merely ten or twenty colors. One should obtain more individual colors than one can readily count, and this quantity of fine gold should be found over much of the creek. Next, one must be able to demonstrate that a substantial quantity of minus 60 mesh gold is being lost by a sluice box. Feed sizing to -10 mesh for spirals requires several steps, therefore one should be able to classify to -1" or -1/2" prior to sluicing. This will improve recovery in the sluice box substantially and is a prerequisite to the consideration of a secondary recovery plant. Another step that will improve recovery in the sluice is the installation of riffles and ports that allow for rapid, daily clean-ups.

A spiral plant consists of more than the spirals themselves. The most technically difficult equipment to select is the screening plant. Screening to -10 mesh is not simple. However, one does not need complete passage of all the -10 mesh sand through the screen, rather we only wish to pass all of the minus 20 mesh gold from the sluice tailings. As such, a static screen, or sieve bend, may be the simplest, and least expensive sizing device. If a vibrating screen is advised, the Derrick Multifeed Wet Sizing Screen is recommended.

Hydrocyclones are usually needed to dewater the undersize from the screening plant prior to pumping. The Sala slurry pump has proven effective in our test programs. It requires a small amount of maintenance time and routinely handles the transport of the 40%-60%(w/w) feed slurry to the spirals (Figure 9).

Spirals recover fine gold and other heavy minerals. In some circumstances these heavy minerals may also be economic. Because of the presence of these other heavy minerals, the task of separating the gold from the spiral concentrate becomes a very formidable one. Amalgamation will work, but often is not cost effective for this material. The best piece of equipment presently on the market is the Gemini 12 Fine Gold Separator (Gemini Table). Although new and still undergoing structural modifications it is an effective method for separating fine gold from a heavy concentrate.

As examples of some of the field testing we are doing in Alaska with the Reichert Spiral, I'd now like to describe two such cases for you. The first is one we performed on a mine in the Koyukuk mining district. Known volumes of material were taken from three locations in the pay zone on bedrock. These samples were processed separately by screening each at 10 mesh and processing the -10 mesh material on a single start Reichert Mark VII Spiral. The +10 mesh gold weight was not used in the calculations since we desired to obtain the ground value on a fine gold basis only. The spiral concentrate from each sample was processed on a

Table No. 3
Manely Spiral Test Assay Summary

Sample No./ Volume	Concentrate Volume	Visible Gold	Fired Weight of Visible Gold	Fire Assay of Table Concentrates	Average Assay	Weight of Table Concentrates	Calculated* Weight of Gold in Table Concentrates	Total Gold in Sample	$/yd³ **
5/50 gal.	1/2 gal.	5 colors	12.05 mg	a) 0 mg b) 0.39 mg	0.195 mg	405 gms	2.7 mg	14.75 mg	$9.56
5/50 gal.	3/4 gal.	40-50 colors	111.25 mg	a) 1.53 mg b) 0.70 mg	1.115 mg	506 gms	19.38 mg	130.63 mg	$5.04

$$* \text{ Calculated gold weight} = \frac{\text{average assay Au (mg)}}{\text{29.1 grams}} \times \text{weight of concentrates (grams)}$$

** $/yd³ = weight of gold (mg) × $0.01/mg ÷ 1/4 yard per sample.

Gemini table and the gold weight recovered by the table's clean gold split was used in the subsequent calculations of ground value ($/yd³). Table No. 1 shows the results of this series of tests. All ground values were determined assuming a gold price of $300.00/oz. and no gold fineness correction was made.

Table No. 1

Results of Spiral Test Performed in Koyukuk Mining District

Sample No.	Sample Volume	Gold Wt. Recovered (mg)		Ground Value ($/yd³)
1	¼yd³	289.39		11.16
2	¼yd³	80.94		3.12
3	¼yd³	+20 -	86.52	
	Gold wt.	20 × 40 -	139.89	
	versus	40 × 60 -	37.71	
	ASTM mesh	-60 -	5.73	
		TOTAL	269.85	10.41

A second series of tests were run in the Manley mining district. The procedure for running the samples was as previously described for the Koyukuk test work. However, the tests at Manley were run at two different times during the 1984 mining season. In the Spring, before mining began, samples were taken from the previous mining season's tailings piles. Tables No. 2 and 3 show the results of this test series. Note that Table 3 also includes an estimate of the gold in the Gemini table's concentrate split. This gold weight was estimated by fire assay and hence these values should be viewed with caution.

During the 1984 mining season, a test was run on a 1/4 yd³ sample from the working sluice box's tailings. The test recovered a total of 214.96 mg of free gold, yielding a sample value of $8.29/yd³. This figure reinforced our earlier conclusions that the sluicing operation was losing gold values in the area of $5.00/yd³. We will follow up this test series with additional on-site work in the 1985 mining season.

Table No. 2

Manely Spiral Test Sample Results

Sample	Sample Volume (5 Gallon Buckets)	Oversize Volume (5 Gallon Buckets)	Concentrate Volume (Gallons)	Results
2	10	6½	1	Panned, no color
3	10	5	1	Panned, no color
4	10	1	½	Tabled. 5 colors
5	10	3½	½ to 1	Tabled. 40-50 colors
6	5	5	½	Panned 18 colors. Field estimate, 15 mg., $1.20/yd³
7	10	2	½	Panned 2 very fine colors

The Hydro-Laser and its use in Underground Placer Mining*

Hugh B. Fate, Jr., D.M.D.

I. Underground Placer Mining

A. Underground placer (drift mining) operation.
1. Low heat rise in frozen placer deposits.
2. Low water production.
3. Removal of frozen gravel in blocks.
4. Good production potential (especially if mechanized).
5. Small space requirements underground.

B. Advantages of underground placer mining.
1. Selective removal of pay gravel
2. Development of high quality placer deposits that are too deep for conventional operations.
3. Less water volume used for sluicing causing fewer settling problems.
4. Fewer environmental disturbances and/or problems.
5. Comparatively low capital costs.
6. Comparable or lower operating costs.

C. Disadvantages of underground placer mining.
1. Fairly high grade of ground is needed due to low volume output.
2. Very strict compliance with *Mining Safety and Health Administration (MSHA)* in mines with shafts over 100 feet vertical depth.

II. Hydro-Laser

A. Description of Hydro-laser.
1. Model 3100 SS with triplex plunger pump.
2. Powered by 4-71 Detroit Diesel Engine coupled to positive displacement pump.
3. Nozzle tip range (.05 - 1.5 mm)
4. Gun type nozzle with shoulder stock.
5. Operating pressure range (3000-16000 psi).
6. Entire unit is trailer mounted and can be pulled by light pickup.
7. Two hose reels with high pressure hoses.

B. Operating parameters.
1. Requires regular daily lubrication inspection.
2. Requires fresh water supply with at least 25 psi.
3. At high pressures, constant flow is needed to forestall rupture.
4. Nozzle is dangerous during high pressure operation.
5. Safety gear is required for splash-back.

C. Common uses.
1. Cleaning hardened deposits from radiators or louvered structures.
2. Cleaning large storage tanks.
3. Demolition of concrete.