KPMA

OUR GOLDEN HERITAGE

GOLD LOSSES AT KLONDIKE PLACER MINES

GOLD RECOVERY PROJECT

(PHASE I)

Prepared for

THE KLONDIKE PLACER MINERS ASSOCIATION

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1 SUMMARY AND CONCLUSIONS

1.1 SUMMARY

Placer mining has made significant contributions to both the history and lifestyle of the Yukon and continues to provide a stable non-governmental economic force. This year there are approximately 185 active placer mining operations with a combined reported contribution in excess of $65 million to the Yukon's small resource based economy. Placer gold recovery at many of these operations is not optimized due to a lack of access to current technology, training and testing facilities.

The objectives of the Gold Recovery Project (Phase 1) are to: collect representative tailings samples; recover the contained gold particles; evaluate and recommend improved recovery technology; and provide assistance in the selection and start up of recommended technology. The Klondike Placer Miners Association's (KPMA) Gold Recovery Project is unique because it is an industry initiated process research and development project which is under the direct control of the placer industry's only representative.

Current gold recovery technology is almost exclusively confined to sluice boxes. A limited number of operations additionally employ feeders and screens. Sluice boxes are very simple, reliable, inexpensive, and yield very high concentration ratios. Many factors contribute to improved recoveries with a sluice box including:

a) Controlled feed rates at less than 8 loose cubic yards/hr per foot of sluice width;

b) Screened pay gravels to at least 3/4 of an inch;

c) Adequate washing and liberation of free gold particles;

d) Water ratio of 17 Imperial gallons/minute per loose cubic yard of pay gravels/hr;

e) Use of both expanded metal and Hungarian riffles in every sluice run;

f) Utilization of a slick plate section before a riffle section to allow gold segregation in the slurry;

g) Even feeding through automation or strict manual control of loading equipment;

h) Sluice box gradients of between 1.5 and 2 inches/foot; and

i) Frequent removal of sluicebox concentrates for upgrading.

Sluiceboxes can recover up to 95% of gold particles as fine as 150 mesh provided that the previous conditions are realized. Sluiceboxes may not be adequate for placer deposits containing very fine pay gravels with abundant clays and fine silts, a high proportion of high density minerals, or extremely fine (-150 mesh) or flattened gold particles. For a White Channel deposit, an oscillating sluicebox kept riffles from packing and provided reasonable gold recovery.

Sampling design is critical for placer gold testwork because test results can be distorted by the "nugget" effect. The uneven distribution of gold particles in a placer deposit often produces large random sampling errors. A sluice tailings stream represents one of the easiest sampling locations, provided the samples are caught in the air.

Sluice tailings samples were collected from six operating Yukon mines at regular intervals with hand held buckets, sample cutters, or a large steel box depending on the coarseness of the discharge (Figure 1). The samples which represented between 1.5 and 7 loose cubic yards, were screened at 16 mesh and shipped to a tabling facility.

At each site, a large number of sample increments were alternately stored in two containers as interpenetrating samples over a period of 2 to 4 days. Comparison of these two samples indicated relative standard deviations (coefficient of variation) as low as 8 %. These errors soared 56% (nugget effect) for two sites with high losses, when a limited number of gold particles as coarse as 14 mesh were found in their tailings.

To evaluate gold recovery efficiency, the collection of head samples is an impractical task as well as being of dubious value. At each operation the sluice boxes were cleaned directly before and after the sampling period. A calculated head grade was determined by adding the gold recovered in the sluicebox and the gold lost to the tailings. The placer gold data recorded over the 2 to 4 day sampling periods represent only a snapshot of a total deposit's characteristics.

Due to the difficulty in evaluating the losses of sluices, some operators and researchers have based their conclusions on errors in logic. These myths (Section 6.5) must be dispelled before the industry can progress.

The field sampling program was very expensive considering that only six sites were sampled. However, the recoverable losses which have been identified at two of the Sites (C and F) would pay for the entire program after one year of operation. This research has a much greater benefit to the entire placer industry as other miners incorporate these recommendations into their sluicing systems.

1.2 CONCLUSIONS

1.2.1 Overall Losses

Overall losses ranged between 0.0006 ounces per loose cubic yard (oz/Lyd3) or $7.57 per operating hour at Site A to 0.0021 oz/Lyd3 or $130.00 per operating hour at Sites C and F. As much as 66% of these losses (or $140,000 per season) could be recovered chiefly by reducing feed rates to 8 Lyd3/hr per foot of sluice width and by screening the pay gravels to 3/4 of an inch.

At all of the sites the majority of the gold values lost to the tailings were coarser than 48 mesh. Tabling testwork detected very little gold finer than 150 mesh in the tailings. With the exception of Site E, gold lost to the tailings had the same shape (Corey Shape Factor) as the gold recovered by the sluiceboxes.

Estimated improvements to recoveries were based on comparisons between the sites, Poling (1986), and Zamyatin (1975) published data. There is a very limited amount of reliable data regarding the recovery of jigs and other gravity concentration devices.

1.2.2 Effect of Screening on Gold Recovery

The tailings sampling program clearly demonstrates the effect of screening on gold recovery with conventional sluiceboxes. Figure 2 illustrates the recovery of Sites A, D, E, and F versus gold particle size. Sites A, D, and E had evenly fed sluiceboxes with optimal feed rates. However, Site A was screened to 3/8 of an inch and recovered 95% of gold as fine as 150 mesh. Site D was screened to 2.5 inches and recovered gold coarser than 65 mesh fairly well.

Site E, an unscreened single run sluice recovered very little of the gold coarser than 65 mesh but still recovered more than Site F's triple run sluicebox. Even though Site F's overall feed rate was nearly optimal, its stationary punch plate distributor was a very inefficient screen (40 %) and directed the majority of the -3/4 inch gravels down the heavily overloaded center run, resulting in heavy losses (12 % overall).

These results agree with published data from the Soviet Union (Zamyatin 1975). Figure 3 illustrates the effect of reducing the top size of pay gravels from 5/8 inches to 5/32 inches on the recovery of a shallow fill (expanded metal riffles) sluicebox. Dramatic recovery improvements can result when the pay gravels are screened. Riffle packing can occur due to a high proportion of either fine silts and clays or heavy minerals in the pay gravels. These two conditions limit the application of conventional sluicebox recovery systems.

1.2.3 Effect of Riffle Type on Gold Recovery

Figure 4 illustrates the effect of riffle type on gold recovery. Data from Site E, F, and Zamyatin (1975) are represented. Site E's sluicebox consisted of a top section of expanded metal riffles directly under 3/4 inch punch plate and its bottom section was lined with Hungarian riffles. The expanded metal riffles were much more efficient at recovering gold finer than 1 mm (14 mesh), however they were unable to retain much of the coarser gold. The coarse gold which passed through the expanded metal riffles was caught by the Hungarian riffles.

These conclusions are mirrored in Site F's side runs (containing expanded metal) and center run (Hungarian riffles), although other factors such as overloading are additionally responsible for the center run's high losses. Many researchers have recommended the use of Hungarian riffles as a "Nugget Trap", however most did not realize just how small a nugget could be. The coarse gold losses with expanded metal are very dramatic as the particle size increases.

1.2.4 Site A

Site A's exceptional gold recovery is illustrated in Figure 8. Site A's gold losses were the lowest recorded at $7.57 per operating hour or 0.00055 oz/Lyd3. This was due to the following good recovery practices: removal of all +3/16 inch feed gravels; feed regulation by a hopper and vibrating screen; and use of efficient expanded metal riffles. Site A's water use is very low compared to recommended values but this does not appear to have affected gold recovery.

Figure 9 , Distribution of Gold Values, illustrates that Site A's gold is relatively fine sized and that most of the lost gold values are between 48 and 28 mesh in size. An unscreened sluicing system with poorer recoveries of fine gold would lead to extreme losses of gold at this site. Additional losses at this site were due to the loss of fine gravels which adhered to boulders rejected by the dry grizzly. The sluice contained only expanded metal riffles, therefore there may also be losses of gold coarser than 1 mm.

1.2.5 Site B

Site B's pay gravels are one of the exceptions to the applications of conventional sluiceboxes. The White Channel gravels located at this site contained a high percentage of clays and fine silts which are difficult to wash and tend to pack the riffles (Figure 6). Riffle packing must be avoided or extreme gold losses will occur. Figure 11, Distribution of Gold Values, indicates that Site B's gold is the finest that was sampled.

The operator used a rotary trommel to scrub the feed and screen it to 1 inch. After using conventional sluiceboxes for years, he decided to try out an oscillating version. The panning motion imparted by the box kept the riffles from packing and produced fair recoveries (Figure 10). The losses at this site were very high at 0.0022 oz/Lyd3 or $57.00 per operating hour. However, it may be very difficult to improve these losses unless a much more sophisticated recovery system is adopted.

A three stage jigging system was used on a White Channel deposit by Queenstake Resources. The system was relatively complex, expensive, and required frequent adjustment by qualified personnel. A very rough comparison with published jig recovery (Zamyatin, 1975) indicates that a jigging system would provide additional values of between $10 to $20 per hour. This is equivalent to additional revenues of $10,000 per 1000 hour operating season, not enough to justify the purchase and operating expenses of a jigging system without further full scale testwork. The oscillating sluice run used only expanded metal riffles. The operator intends to install a nugget trap to guard against coarse gold losses.

1.2.6 Site C

Site C's losses were very high at 0.0022 oz/Lyd3 or $126.04 per operating hour. These losses occurred despite the very coarse size distribution of its pay gravels (Figure 6) and gold (Figure 13) and the use of a 6 inch grizzly. Gold losses were primarily due to overloading the sluice runs with 4 to 5 times the optimum levels of pay gravels and process water. Sluicebox turbulence was worsened with the passage of minus 6 inch rocks. In addition the sluice run used Hungarian riffles which are less efficient at recovering gold particles smaller than 1 mm.

Increased recoveries should be achieved by making the sluicebox four times wider. Process water volumes would also have to be increased to move coarse rocks down the sluice run. Improved recoveries should also result from the use of expanded metal riffles over Nomad matting instead of Hungarian riffles. Hungarian riffles should be used in the last section of the sluicebox (at least 8 feet) to trap gold particles coarser than 1 mm.

Expanded metal riffles cannot tolerate the wear resulting from the passage of coarse rock. Therefore, the pay gravels should be screened to 3/4 inches. Screening could be accomplished by suspending metal punch plate above the expanded metal riffles (similar to Site E), or preferably by utilizing a vibratory screen ahead of the sluice. The vibrating screen would also promote better washing and reduce water requirements dramatically.

Comparison with Site F's side runs indicates that a screened sluicing system should recover half of the lost gold or $62.71 per hour ($50,200 per operating season). A further comparison with Site E's recovery indicates that an unscreened system with suspended punch plate would recover $26.37 per hour or $21,100 per season of lost gold values.

1.2.7 Site D

Gold losses at Site D amounted to only 0.0010 oz/Lyd3 or $34.49 per operating hour despite its relatively fine pay gravels (Figure 6) and abundance of heavy minerals. This operator attempted to minimize riffle packing problems by shutting down his sluicebox and hand raking the riffles every 2 hours and it appears to have worked. A Derocker provided feed regulation and screening to 2.5 inches. The operator used a combination of expanded metal and Hungarian riffles in the sluicebox which was fed at optimal pay gravel and process water rates.

A comparison with jig recovery data (Zamyatin 1975), and with Site C's oscillating sluice indicates that it is doubtful that Site D's sluicebox recovery (Figure 14) could be improved significantly except through finer screening.

1.2.8 Site E

Gold losses from Site E's sluicebox total 0.0011 ounces per loose cubic yard or $22.24 per operating hour. These losses are among the lowest, even in comparison to operations employing screening and feeding equipment and much lower volumes of wash water.

Site E's sluicing equipment (Section 1.2.3) recovered much less of the fine gold (-48 mesh) than operations employing screening equipment (Figure 16), however there was almost no fine gold in the original pay gravels (Figure 17). Feed rates, slope, and riffle layout were optimal, but the water ratio was three times higher than recommended. The high water levels which were required to push coarse rocks down the sluice run did not appear to affect gold recovery.

The majority of the gold lost at Site E was greater than 48 mesh and was significantly flatter than the recovered gold (eg Corey Shape Factor of 0.2 vs 0.4 for recovered gold). Flat gold is difficult to recover with any gravity concentration device, especially in the finer sizes. Site E's gold sluicing system was an optimal design for unscreened feed and was ideal for their deposit due to the coarseness of their gold.

Over a 250 hour operating season, the total gross value of gold losses is only $5,500. Even if half of these losses were recovered, the net amount of $2,300 per season would not justify the purchase of any additional equipment.

1.2.8 Site F

Gold losses from Site F amount to 0.0021 oz/Lyd3 or $132.00 per operating hour. Gold as coarse as 14 mesh was recovered in the center and side runs of this triple run sluice (Figures 18 and 19). The recovery of the dump box and side runs was poor compared to other sites. The center run's recovery was much worse and it lost almost all of the gold finer than 28 mesh.

The pay gravels were pushed onto the side of the dump box which was covered with 3/4 inch punch plate. The gravels were forced down the box by a stationary monitor to the distributor. This 3/4 inch punch plate section directed the underflow to two side runs, while the overflow continued on to the center run. An undercurrent sluice run was located under the final section of the center run. Expanded metal riffles were used in the dump box and side runs, while Hungarian riffles were used in the center run. All of the riffles were underlain with backed Nomad matting.

These high losses were primarily due to the extremely poor screening efficiency of the distributor section. Despite the large area of its punch plate (46 ft2), less than 40 % of the fine gravels were distributed to the side runs. The poor screening efficiency (40 %) of the distributor resulted in a feed rate which was four times the optimal to the center run and one quarter the optimal to the side runs. The center run was overloaded and had to pass large boulders and high volumes of water which created excessive turbulence. In addition, the bulldozer spent only 5 % of its cycle time feeding the dump box and this resulted in surging feed rates.

If the recovery of the center run could be increased to that of the side runs, an additional $87 per operating hour or $140,000 per operating season of gold would be recovered. This could be accomplished by screening the pay gravels prior to sluicing with a rotary trommel or Derocker-type assembly. The side runs also lost gold as coarse as 14 mesh. These losses could be reduced by including a section of Hungarian riffles at the end of the sluice runs. Gold recovery could be improved further by controlling feed surges with a manned monitor and devoting more cycle time to feeding the dump box.

1.2.9 A Triple Run Box's Dilemma

Site F's triple run box was homemade, but it demonstrated the shortcomings of many triple run boxes. The distributor of any triple run box must provide large volumes of water (greater than 5 times the optimal flow rates) to force large boulders down the center run. In addition it attempts to direct the fine gravels to the side runs. It is unlikely that any stationary punch plate distributor can satisfy both of these objectives. The turbulent volumes of water which must remain above the distributor section will trap some of the fine pay gravels and carry them down the center run.

The center run is normally lined with Hungarian riffles and is extremely turbulent due to the passage of boulders and high water volumes. Therefore it is unlikely that reasonable gold recoveries will result in the this type of sluicing environment. Many operators have added an undercurrent to improve the screening action and recovery efficiency of their center runs. However, the recovery characteristic of Site F's center run did not appear to be improved by the inclusion of its undercurrent sluice.

1.3 RECOMMENDATIONS

Each of the six site operators received detailed recommendations specific to their sites. Many of these recommendations were noted in the previous section (1.2). The greatest gold losses were due to overfeeding the sluice runs. Fine gold recovery improvements also resulted with screening and the use of expanded metal riffles. The recovery of coarse gold (+1 mm) was improved by providing a section of Hungarian riffles in each sluice run.

The recovery of placer gold in a conventional sluicebox can be optimized by:

a) Controlled feed rates at less than 8 loose cubic yards/hr per foot of sluice width;

b) Screened pay gravels to at least 3/4 of an inch;

c) Adequate washing and liberation of free gold particles;

d) Water ratio of 17 Imperial gallons/minute per loose cubic yard of pay gravels/hr;

e) Use of both expanded metal and Hungarian riffles in every sluice run;

f) Utilization of a slick plate section before a riffle section to allow gold segregation in the slurry;

g) Even feeding through automation or strict manual control of loading equipment;

h) Sluice box gradients of between 1.5 and 2 inches/foot; and

i) Frequent removal of sluicebox concentrates for upgrading.

Site A demonstrated that a sluicebox can recovery 95 % of the gold as fine as 150 mesh when feed control and fine screening are provided to a sluicebox. Site E demonstrated that sophisticated feed control equipment and fine screening is not necessary for creek deposits containing very coarse gold.

The size distribution and washability of pay gravels and the size distributions and shapes 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.

More pilot scale testwork similar to Poling (1986) is required to determine the best applications of different types of riffles and matting in sluiceboxes. Controlled testwork is also required to reliably determine the recovery characteristics of jigs and other gravity concentration devices for placer deposits containing a high proportion of clays and silts, heavy minerals, or extremely fine (-150 mesh) or flattened gold.

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

a) determine the additional gold recoveries resulting from the recommended modifications (Sites C and F);

b) optimize the performance of a three run box;

c) expand the knowledge of gold recovery at a greater variety of deposit types and recovery equipment.

The use of nuclear tracers in field testwork should be investigated. Nuclear tracers may be able to reduce sampling costs; increase the number of sites investigated; and provide quantitative data for operations where collecting samples is very difficult or dangerous due to the passage of coarse rocks.

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2 ACKNOWLEDGMENTS

2.1 FUNDING

The National Research Council's Industrial Research Assistance Program (IRAP) cost shared the following: salaries and administration costs for the project manager and his field assistants; sampling consulting services (Jan Merks of Matrix Consultants); and laboratory services (Bud Kenzie of Midas Concentrating Company).

The Canada/Yukon Economic Development Agreement provided an additional $60,600 towards costs incurred during the field sampling and analysis program.

The Canada Center for Minerals and Energy Technology (CANMET) provided funding for Matrix Consultants to consult regarding a tailings sampling design.

2.2 TECHNICAL ADVISORS

This project's technical advisors provided invaluable advice and recommendations and have devoted a considerable amount of time to ensure its success. They are:

a) Tad Cienski, Senior Mineral Processing Engineer for CANMET;

b) Ron Johnson, Vice President of the KPMA and a partner/operator of Beron Placers;

c) George Poling, Professor, University of British Columbia's Department of Mining and Mineral Process Engineering; and

d) Dan Walsh, Instructor, University of Alaska's Mineral Industry Research Laboratory.

2.3 ADMINISTRATIVE SUPPORT

Alan Fry, Executive Assistant to the KPMA, provided the administrative support for the Project.

2.4 ASSISTANTS

A mineralogical analysis of the samples was performed by Larry Carlyle Geological Services. The field assistants: Robert Rebinsky; Gary Pletcher; and Merle Wickstrand worked long hours in adverse conditions to collect the samples and site data.

3 ABSTRACT

The Yukon's 185 placer operations currently generate over Can $65 million and 10% of the employment of this small resource based economy. Placer gold recovery at these operations is not optimized due to a lack of access to current technology, training, and testing facilities. Phase 1 of the Gold Recovery Project is a field research and laboratory program which would facilitate the development and transfer of sampling and gold recovery technology to the Yukon placer industry.

Unlike previous Yukon placer research programs, the Gold Recovery Project is an industry initiated research project. It is under the direct control of the Klondike Placer Miners Association (KPMA) and is thus able to provide the maximum benefit to the Yukon's placer mining industry and assure a cooperative and confidential program with a minimum of overhead costs.

The tailings and concentrates from six operating placer mines were analyzed in 1988. This paper describes the background, procedures, and results of the testwork.

4 OBJECTIVES

The objectives of Phase 1, Field Sampling Program, are:

a) To conduct applied research in placer gold recovery technology;

b) To develop and implement a statistically based placer tailings sampling program;

c) To determine the magnitude and causes of gold losses of operating Yukon placer mines;

d) To recommend recovery improvements to individual operators where cost-effective;

e) To assist the operators to implement the recommendations.

f) To demonstrate the performance of improved recovery technology;

g) To distribute the results of its field and laboratory research for the benefit of the entire placer industry.

5 INTRODUCTION TO THE YUKON'S PLACER MINING INDUSTRY

5.1 ECONOMIC AND SOCIAL CONTRIBUTIONS

The placer mining industry continues to provide significant contributions to the economy of the Yukon. Placer gold mining was responsible for the accelerated early development of the Yukon and contributed over $65 million to the local economy in 1988.

Present placer operations also create significant spin-off benefits to the service and supply industries. Through the recent recession (1981-1985), every hardrock mine in the Yukon closed for the first time since hardrock mining began in the early years of the 20th century. Even through this recession, placer mining continued to provide a stable non-government economic force in the Territory. Rising gold prices have allowed a sustained growth in the industry and the highest production levels since 1917.

The Klondike Gold Rush (1896-1898) focussed world attention on the Yukon Territory. Placer mining operations from this era are responsible for the attraction of considerable numbers of tourists to the Yukon's frontier and to the Klondike in particular.

5.2 MINING METHODS

Mining during the gold rush was accomplished by hand methods relying mainly on underground excavations in permanently frozen gravels and on the surface with the use of hydraulic monitors. After the turn of the century, placer claims were consolidated by large companies and mined using bucket wheel dredges. The last of these dredges was owned by Yukon Consolidated Gold Company and ceased operations in 1966 because of rising operating costs and a fixed price of gold.

Current Yukon placer mines are relatively small by world standards, subsisting mainly on unmined remnants from the dredging era, extensions of existing deposits, and the reworking of historic placer creeks. Many deposits are worked under confined conditions in narrow gulches, and on small claim blocks.

Most of the present mining methods rely on the excavation and processing of large volumes of low grade material. This is usually accomplished with diesel powered earth moving equipment such as tracked dozers, rubber tired loaders, backhoes and scrapers. Mechanized underground mining is being utilized in three locations (Dago Hill, Jackson Hill, and Miller Creek) where it is justified by the greater depths and higher grades of placer deposits. A refurbished bucket wheel dredge was operated in 1986 and 1987 at Clear Creek and reported very low operating costs.

5.3 SLUICING PLACER GRAVELS

The sluice box has been used in the Yukon since the Klondike Gold Rush, and with very few exceptions, is still the only device used for primary gold concentration. A sluice box is a rectangular flume containing riffles, through which a dilute slurry of water and alluvial gravel is passed. Poling (1986) reports that turbulent eddies are formed around the riffles and form a dispersed shearing particle bed where particles of high specific gravity are concentrated. A complete shut down is required to remove the concentrate contained in the sluice's riffles. The frequent removal of concentrate is required to minimize riffle packing and gold migration.

The most popular sluice riffles include expanded metal, angle iron ("Hungarian"), punch plate, and grader blades (Mayo district). Larger riffles can better withstand the wear associated with coarse gravels but retain a larger volume of concentrate which requires more time to upgrade. In the Yukon there is a tendency to use the smallest readily available riffle which has reasonable wear characteristics. Matting is usually placed under the riffles and includes the traditional cocoa matting, and the more popular "Nomad" or artificial turf.

Several sluice box hoppers or dump boxes have a punch plate deck which allows undersize gravel to pass through the holes and be distributed to an undercurrent or set of side runs (ie Ross Box, Pearson Box). An undercurrent is a flume located directly below the punch plate deck, side runs are located on either side of the main sluice run. In the side runs, a finer riffle, shallower sluice gradient and lower water volumes can be utilized to improve the recovery of placer gold particles.

Large quantities of water are often required to liberate agglomerated gold particles from placer gravels and to mobilize particles prior to the recovery process. Inadequate washing is a common cause of gold losses. Peterson (1987) reports increased recovery at several Alaskan placer mines due to improved washing from a hydraulic lift (jet pump).

Many of the Yukon's placer operations now use either a stationary grizzly or a moving deck grizzly such as the "Derocker" to eliminate coarse material from the sluice box feed. Grizzlies are most common in the glaciated placer areas and where the placer gravels contain large angular rocks. A few other operations are screening to finer sizes with vibrating deck screens, and with rotating trommel screens. Vibrating screens have lower capital and operating costs while trommels wash the oversize more thoroughly with less water (Peterson 1987).

When placer gravels are not screened, additional water and steeper sluice box 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 results in continuing gold losses. Screens also improve washing by breaking up clumps of clay and cemented particles.

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

Pay gravels containing a high proportion of high specific gravity minerals such as magnetite, or a high percentage of clay are prone to riffle packing. Extreme gold losses occur when a sluice's riffles become packed or excessively scoured. For these deposits or where insufficient quantities of water are available, the investigation of alternative recovery technology is advisable.

5.4 PLACER MINING SEASON

Placer miners have a very short (100 day) season in which to earn a year's income. Several obstacles to continuous mining include mining and processing equipment maintenance and breakdowns. Some operations are down for considerable periods of the season awaiting parts and repairs. Many operators working in space restrictions must frequently stop operations and move their processing plant with them as their mining face progresses. Other operators have water shortages at midsummer and autumn and must wait frequently for adequate water volumes before continuing to sluice.

It is this severe shortage of time, further aggravated by equipment breakdowns, which makes many operators reluctant to devote time to experiment with other forms of recovery equipment or to conduct research at their sites. Some of the operators are under severe financial constraints and any delays cannot be afforded.

6 PREVIOUS RESEARCH

6.1 THE REQUIREMENT

The magnitude and causes of gold losses from the Yukon's 185 placer operations was never established prior to the Gold Recovery Project. Many of the placer operators suspected potential losses but were inadequately trained and equipped to assess the nature and extent of these losses. In the hardrock mining industry, mineral concentration systems are designed for each particular deposit by specialists who utilize detailed metallurgical testing. This is generally not the case in the placer industry, due to:

a) The extremely limited amount of accurate data regarding the recovery mechanisms and capabilities of existing concentrators;

b) The absence of information about placer deposits due to the extreme difficulties and high cost of obtaining representative samples, especially where coarse gold particles are concerned; and

c) The high cost of placer gold recovery testwork and evaluation.

Significant gold losses are expected due to:

a) The application of inappropriate recovery equipment;

b) The inability to optimize equipment parameters such as screening, feed rate, and water addition (due to sampling difficulties, and a limited sluicing season); and

c) The lack of training in gold recovery technology.

Environmental restoration of placer mining areas in which significant gold reserves remain is futile. Several areas of the Klondike have been re-mined several times. This is partly due to the greater efficiency of materials handling equipment and the increasing price of gold, but is also due to ineffective gold recovery technology which left much of the gold behind.

Steve Morrison (Northern Affairs Program, pers comm) has estimated that there are less than 5 years of placer reserves in the traditional mining areas. There are many areas with good potential for further placer reserves including the Indian and the Forty Mile Rivers. Both of these areas, and gold reserves contained in deposits which are mined primarily for sand and gravel content, also contain a large percentage of fine gold which is difficult to recover with crude sluice boxes.

6.2 PREVIOUS RESEARCH IN THE YUKON

The most notable placer research projects completed to date include "Fine Gold Recovery of Selected Sluicebox Configurations" conducted at the University of British Columbia (Poling 1986) and a Placer Sampling Program (Berry 1985). A materials handling research program attempted to define the feasibility of pumping gravels but has been unsuccessful due to difficulties in obtaining sufficient operating data.

The U.B.C. study is the only known evaluation of a pilot scale sluice which was performed 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 recoveries of 95 % of placer gold as fine as 0.15 mm (100 mesh) should be achieved at the following recommended optimum processing parameters:

a) Feed rate of 8 loose cubic yards of pay gravels/hr per foot of sluice width (Gold recovery was most sensitive to variations in the feed rate, Figure 5);

b) Process water ratio of 17 Imperial gallons/minute per loose cubic yard/hr of pay gravels;

c) Process water rate of 170 Imperial gallons/minute per foot of sluice width ;

d) Main riffle section of expanded metal, followed by a "nugget" trap of Hungarian riffles; and a

e) Sluice slope of 1 5/8 to 2 inches per foot (Steeper slopes required less water flow but were more sensitive to variations in feed rate).

The practise of "running clean" (full water flow without any gravels prior to shut down) need not result in gold losses.

The Placer Sampling Program was conducted in the summer of 1984 and included the sampling of the feed gravels and tailings from four Yukon placer operations which utilized sluice boxes. A 200 liter (45 gallon drum) volume of each sample was transported to the Canada Center for Minerals and Energy Technology's (CANMET) laboratory in Ottawa. CANMET's laboratory analyzed the content and size distribution of the gold particles (Berry, 1985).

Samples from one operation on the Forty Mile River contained an appreciable amount of magnetite and very fine gold (92% -0.149 mm, 100 mesh, and 50% -0.037 mm, 400 mesh). This fine gold was very difficult to extract with a laboratory shaking table and a low concentration ratio of 53:1 was required to obtain an 83 percent recovery. The other creek placer samples indicated much coarser gold: Sulphur Creek and Hunker Creek at 98% +0.297 mm; and Upper Dominion Creek at 86% +0.297 mm. Several assay anomalies attributed to the particle effect indicated that the conventional laboratory splitting and sampling procedures which were employed by CANMET were not appropriate for placer gold material.

6.3 PREVIOUS RESEARCH IN ALASKA

The US $2.7 million Alaskan Placer Mining Demonstration Grant Project (Peterson 1987) allowed experienced placer miners to study and test new methods promoting fine gold recovery and pollution control. The program was beneficial, however many of the tests were performed in the field without controlled conditions, over short durations, and with incomplete or faulty sampling.

Peterson reported that riffle packing and the resulting gold losses were caused by prolonged periods between sluicebox cleanups and by increasing the total suspended solids concentrations in the sluicebox wash water. He also stated that the most cost effective alternative, with widespread applicability, for improving fine gold recovery over a crude sluice box is to provide:

a) Classification (Screening);

b) Thorough washing of pay dirt; and

c) Even feeding to a well designed sluicebox.

He determined that in general, gold recovery devices added to the end of a sluice box, or used to replace a sluice box, including jigs, Reichert spirals, and centrifuges are less feasible compared to this proven alternative.

Peterson also stated that "classification improves recovery in at least three ways: smaller sized material requires less water to wash it through the box, leading to a more controlled sluicing environment; smaller particles tend to promote more thorough washing; and reducing the size of the largest particle reduces the tendency for particles to become temporarily lodged between the riffles."

6.4 PROPERTIES OF PLACER GOLD AFFECTING RECOVERY

The high specific gravity of placer gold (15-19) compared to common gangue minerals (2-5) usually makes it amenable to gravity concentration techniques. Factors reducing the gravity recovery efficiency of gold particles include: fine size distribution, flat particle shape, rough surfaces, hydrophobicity (preference for attachment to air surfaces), and low effective specific gravity.

The size distribution and shape of placer gold particles found in an alluvial deposit are a function of their geologic and hydrologic history. The gold particles from creek deposits are relatively coarse and may contain locked gangue minerals. The gold grains from river bar and beach deposits are generally uniformly fine and flattened.

Most of the data on the particle size distributions of alluvial gold have been obtained from concentrates derived from production and from grade evaluations. Grade evaluation procedures generally simulate the recovery efficiency of conventional placer technology such as sluices and jigs which falls off sharply below the 0.2 mm size. Regardless of the proportion of fine gold present, it may not be detected, particularly if obscured by clay.

Yeend (1985) evaluated the abrasion of gold particles using a tumbler to simulate natural high energy deposits. He concluded that gold absorbs the impact and abrasive forces exerted by sand and gravel and changes shape, unlike the common rock forming minerals, which fracture and break . He also observed that cobbles rather than sand are responsible for most of the abrasion and physical changes of gold. He found that velocity was more important as a factor in abrasion of gold than travel distance.

The natural hydrophobicity of placer gold causes it to float in direct conflict with gravity concentration. Wang (1983) states that gold surfaces which have been cleaned by leaching or contaminated by organics are naturally hydrophobic.

The leaching of silver or gangue minerals formerly locked in gold causes placer gold to be porous and have a low effective density. Ackels (1985) examined a jig concentrate and discovered several brown porous gold particles which had not been recovered previously by his sluicing operation. Both porous gold and gold which has locked gangue minerals has a lower density and is more difficult to recover.

6.5 SOME COMMON GOLD RECOVERY MYTHS

Due to the difficulty in evaluating the recovery and losses of placer gold concentrators, some operators and researchers have based their conclusions on errors in logic. The following nine myths must be dispelled before the reader can properly evaluate any data regarding gold recovery.

FINE GOLD FAIRY-TALE

a) I RECOVER SOME EXTREMELY FINE GOLD, THEREFORE I MUST BE RECOVERING ALL OF THE GOLD PARTICLES WHICH ARE COARSER.

Even the crudest sluicebox will recover some proportion of the fine gold present in a deposit, especially if there is a high proportion of fine gold in the pay gravels. Therefore this is not a valid recovery test.

NUGGET NOTS

b) I RECOVER SOME NUGGETS, THEREFORE I MUST BE RECOVERYING ALL OF THE NUGGETS.

The recovery of coarse gold is not as easy as it once seemed. Field data suggest that expanded metal riffles are better at recovering gold finer than 1 mm, whereas Hungarian riffles are better at recovering gold coarser than 1 mm. Many placer deposits contain a high proportion of gold coarser than 1 mm and may have high losses if they have not included a nugget trap in each of their sluice runs.

TURBULENCE TRAUMA

c) I RECOVER ALL OF MY GOLD IN THE FIRST FEW FEET OF MY SLUICEBOX, THERE MUST BE NO GOLD IN MY TAILINGS OR IT WOULD HAVE DROPPED OUT FURTHER DOWN THE SLUICE.

Analysis of sluicebox tailings completely discount this statement. Even relatively coarse gold particles which are caught in the turbulent flow of a sluicebox can remain suspended for the entire length of the sluicebox. Poling (1986) recommended a slick plate be installed in the center of a sluice run to allow the gold particles to segregate to the bottom of the slurry flow where they could be trapped by riffles.

SAMPLING SORROWS

d) I KNOW THAT THERE IS NO GOLD IN MY TAILINGS, BECAUSE I PANNED ONE PAN OF THE GRAVELS FROM THE TOP OF THE PILE.

One pan is a far too small sample and prone to the "nugget" or gold particle effect. Tailings segregate once they hit the ground, therefore the top is a very poor sample.

TRIAL AND ERROR TRIVIA

e) I SET UP THE BEST RECOVERY SYSTEM FOR MY PLACER DEPOSIT BY TRIAL AND ERROR, EVEN THOUGH I WAS UNABLE TO ESTIMATE THE HEAD AND TAILINGS GOLD VALUES.

Placer gold deposits may have wide variations in the gold values from one part of a deposit to another and testwork based only on the amount of gold recovered is erroneous.

EMOTIONAL ATTACHMENTS

f) THE EXCELLENT RECOVERY CHARACTERISTICS OF THIS DEVICE WERE PROVEN BY OUR ANCESTORS LONG BEFORE (THE SCIENTIFIC METHOD OF INVESTIGATION).

g) IF YOU CAN'T GET ALL OF THE GOLD THE WAY I'M PRESENTLY OPERATING, I'M NOT INTERESTED.

Many operators invest a lot of time and money in the design and construction of their sluiceboxes and are reluctant to abandon their inventions even when it is proven that they are very inefficient.

FATALIST PHILOSOPHIES

h) YES, I KNOW THAT I'M LOSING SOME GOLD, BUT THE AMOUNT OF GOLD THAT I AM CURRENTLY LOSING IS INEVITABLE.

i) SURE I'M LOSING GOLD, BUT I'M STILL MAKING MONEY (IN SPITE OF MYSELF).

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 may result in more gold at each cleanup and this extra gold can increase the profitability of an operation dramatically.

7 PROCEDURE

7.1 INTRODUCTION TO SAMPLING DESIGN

Sampling design is critical for placer gold testwork because pay gravels and tailings contain gold in trace amounts, commonly represented by only a few particles per sample. If very few gold particles are present in a given sample then the influence of a single gold particle can be too dramatic (the "nugget" effect). Poor sampling design and inadequate quality control render the results of any testwork program completely invalid.

The uneven distribution of gold particles in a placer deposit often produce large random sampling errors. A large number of small sample increments collected over a 2 to 4 day sampling period minimizes these errors and other variations in the sluicing cycle. The numerous increments are more representative than a single large sample. When they are stored alternately in two sets of containers, they represent two interpenetrating samples. When the two interpenetrating samples are processed and evaluated independently, their gold values can be compared to assess the precision of the sample.

Fortunately the tailings stream of a sluice run or jig represents an excellent sampling location provided the samples are collected before they touch the ground. In the tailings slurry, the material is already washed, it is suspended in a slurry, and has had most of the coarsest gold removed from it.

To evaluate gold recovery efficiency, the collection of head samples is an impractical task as well as being of dubious value. Very large samples would be required to reduce errors caused by the nugget effect. Instead, a calculated head grade can be determined by adding the value of gold recovered in the sluicebox to the value of gold lost to the tailings.

Sample size selection is a compromise between the higher costs of obtaining, handling, and processing large samples and the greater degree of confidence in testwork and analysis with larger samples. It is necessary to obtain the minimum sample size which will adequately satisfy the desired precision at a given confidence level.

A detailed sampling design was developed in the Gold Recovery Project Feasibility Study (Clarkson, 1988). This analysis estimated that duplicate samples of sluicebox tailings consisting of 200 liters each of -16 mesh tailings would yield error levels of 7 and 20 percent for average maximum gold particle sizes of 0.3 and 0.6 mm respectively. This sample volume represents a different proportion of the placer gold lost to the tailings at each site.

7.2 FIELD SAMPLING PROGRAM

The six sites chosen represented a good cross section of the Yukon placer industry by location and processing equipment. The recovery technology at the six sites varied from a single run sluice fed with a bulldozer to sites employing feeders, scrubbers, and screens.

At each of the sites (A through F), the following measurements and observations were completed:

a) pay gravel volumes through stadia surveying or bucket count/time study methods;

b) process water flow rates with the use of a Price current meter, or float methods;

c) sluice box dimensions, slope, type and configuration of riffles;

d) dimensions of feed hoppers, grizzlies, screens and related equipment where applicable;

e) sluice operation characteristics (such as evenness of feed and flow rates) and recovery characteristics (such as riffle packing);

f) effectiveness of washing and screening equipment from examination of discharged oversize and sluice tailings;

g) broad characterization of placer deposits and nature of pay gravels as these relate to gravity concentration; and

h) size distribution of pay gravels.

At every site the material flow rates and processing parameters were measured. Then preliminary sample increments were taken and analyzed to determine the optimal sampling procedures and the size, number, interval, and scheduling of these increments.

Gold is often distributed unevenly across the width of a sluicebox, therefore at each site the sampler cut across the full width of the discharge at regular intervals. A few sites required 24 hour sampling schedules. Some operations could be sampled with hand held buckets, others required the use of a remote sample cutter, (Figure 1), and site F's center run which discharged boulders required a steel container mounted on a D8 bulldozer. There was no single method which was applicable at all sites due to the wide variation in materials handling and processing equipment.

Between 125 and 1200 sample increments were collected alternately into two interpenetrating subsamples. The volumes, weights, and slurry density of these increments were recorded constantly for sites A and B and on a spot basis for the remaining sites.

The accumulated increments were then wet screened at 4, 8, and 16 mesh in an 18" diameter "SWECO" rotary screen. The -16 mesh material was transferred to the two 200 liter Teflon lined, steel barrels for shipment. At Site F two pairs of samples were taken, one pair for the center run and another for the side runs. The +16 mesh increments were measured and processed in the field with a 10" wide Long Tom containing a section of Hungarian riffles, a smooth slick plate, and a section of expanded metal riffles. Gold coarser than 16 mesh was recovered at Sites C and F.

Extreme care was exercised to minimize spillage during the collection and handling of the samples. Excess water was gently decanted from containers after the application of detergent. All sample screening and processing equipment was cleaned after each site and the accumulated material was panned to eliminate any carry over of gold values.

At each site the following samples were collected:

a) two final 200 liter samples of -16 mesh sluice tailings, (4 final samples at Site F);

b) gold particles and/or concentrates from the Long Tom and SWECO clean ups; and

c) an 800 gram sample of the pay material and coarse Long Tom tailings for sieving and mineralogical analysis.

The sluiceboxes were completely cleaned before and after the sampling period. This requirement forced some of the operations to do an additional cleanup and to clean up different sluicing sections separately. All of the operations except Site A, allowed us to thoroughly examine the placer gold recovered from the sluice box during the sampling period. The concentrates were screened, weighed, and examined under a microscope. The grade of the pay gravels was calculated from the sum of the weight of gold recovered by the sluicebox and the weight of gold lost to the tailings.

Microscopic examination revealed the maximum, minimum and average (mode) Corey Shape Factors for each size fraction, as well as other pertinent information. The operators at Site A misunderstood the requirements of the sampling program and allowed an examination of a portion of their "typical" concentrates.

7.3 LABORATORY ANALYSIS OF TAILINGS

Midas Concentrating Company of Dawson City was contracted to process the fourteen 200 liter (45 gallon) drums of samples on a full size mineral concentrating table over a 160 hour period. A fairly clean gold concentrate, middling, and a small sample of the tailings from each of the barrels were shipped to Whitehorse for final upgrading, weighing, and mineralogical analysis.

The tabling procedure was developed following testing conducted by Bud Kenzie (Midas) and the project manager. This testwork indicated that a relatively pure gold concentrate and a middling consisting of mostly sulfides and/or black sands could be recovered from the samples provided the samples were screened at 35 mesh and processed separately several times on a full size mineral concentrating table. Multiple runs on the table also eliminated the need for upgrading with a Superpanner. A finer screen was not used because the samples were 80% coarser than 35 mesh.

The following tabling procedure was considered thorough:

a) Process the +35 mesh and the -35 mesh samples separately after making the necessary adjustments of slope, action, and water addition to optimize recovery;

b) Table each sample, collect as Rougher concentrates and Rougher tails;

c) Retable Rougher tails and combine this Scavenger concentrate with the Rougher concentrate, Scavenger tails are now the Final tailings;

d) Retable Rougher concentrate until only the Middling consisting of Sulfides and/or Black Sands remains, collect the individual gold particles, table tails are combined with the Final tailings;

e) Continue retabling Middling and brushing gold particles into Final concentrate container until no further gold particles are observed;

f) Retain, dry and weigh all of the middling fraction, collect a one liter sample of the Final tailings and return the Final tailings to the original drums.

All of the sample barrels were analyzed individually and all apparatus employed was cleaned completely between each sample. Tabling was performed during the daylight hours to aid in the identification of extremely fine individual gold particles. Screening and sample handling was performed at all hours of the day.

7.4 FINAL UPGRADING AND ANALYSIS

The concentrates, middlings, and samples of table tailings were received in Whitehorse from the Midas Concentrating Company. The middlings and final table tailings samples were examined under a microscope by a geologist to determine their mineralogy, and especially to note any other valuable heavy minerals.

All impurities were removed from the table concentrates with the following procedure:

a) Each concentrate was individually sieved through 20, 28, 48, 65, 100, and 150 mesh screens;

b) Each size fraction was cleaned with Nitric acid and carefully hand panned to separate the free gold particles;

c) The rejected material was amalgamated to isolate any remaining gold particles and then examined under a microscope to guard against any gold losses;

d) Finally, all of the gold particles were hand sorted under a microscope to eliminate any gangue minerals, air dried, and weighed with an electronic balance.

The sulphide/black sand middlings produced when the sluicebox tailings were tabled were weighed, and fire assayed. The fire assay was performed to determine if there were substantial amounts of locked gold in the sulphide or black sand minerals. In many cases there was such a small quantity of middlings that the entire amount was fire assayed. No attempt at determining gold losses in the table tailings was feasible.

8 DATA

8.1 OVERVIEW OF PARTICIPATING MINES

8.1.1 Site A

Site A was a high bench river placer deposit containing very fine sized pay gravels and placer gold (Figures 6 and 9). The operators were cleaning out depressions in the bedrock horizon with a hydraulic excavator. The excavator piled the combined weathered bedrock and placer gravels. The wheeled loader mucked the pay dirt random distances from the piles to the grizzly/hopper.

The dry grizzly was located directly above the hopper and consisted of rails with an eight inch inside spacing. The grizzly was inclined towards the loader and larger rocks would often slide down in front of the hopper. When the grizzly was clogged with large boulders a hydraulic cylinder could be activated to raise one end of the grizzly and allow the wedged boulders to roll off. This inclination generally resulted in a loss of fines which adhered to the unwashed boulders.

The steel hopper was rectangular in shape and narrowed at the bottom to a chute. A double deck vibrating screen was located directly under the chute. The screen decks had 3/4" and 3/16" openings. The screen was inclined and its oversize was jettisoned directly opposite to the feed side of the hopper, the undersize fell through a small chute to the sluice run. The sluice was inclined at 3" to the foot and was only 22" wide and 15' long. Its riffles consisted of fine expanded over backed "Nomad" matting. The sluice discharged into a drain which was periodically mucked out with the hydraulic excavator. The sluicebox was cleaned up after 25 hours of sluicing. Its concentrates contained a high proportion of magnetite and were upgraded on a gold wheel.

The sluice operated at a very high solids/water ratio and the resulting tailings contained significant amounts of fine soils. The tailings samples were difficult to wash and were screened twice in the SWECO sample screen to yield liberated screen fractions. The sluice run was easily sampled using hand held buckets. Two hundred and eighty four individual sample increments were collected into two (interpenetrating) subsamples.

8.1.2 Site B

Site B was a high bench White Channel placer deposit containing abundant clays, very fine silts and extremely fine gold in combination with rounded quartz gravels. The operators mined by cutting the vertical gravel face and piling the pay gravels with a bulldozer. A small loader mucked the piled material a short distance to the hopper.

The pay gravels flowed through the feed hopper to a four foot diameter rotary trommel. There were spray nozzles located at the entrance and exit of the trommel. The majority of the trommel (20 feet) was a wash section, fitted with random lifters to tumble the feed gravels. The remaining 4 feet of the trommel consisted of a screen with 1 inch square openings. The well-washed quartz oversize discharged out the end of the trommel and was pushed away with the bulldozer.

The minus one inch screen undersize was split and diverted to twin sluices which were each 20 feet long by 4 feet wide and contained expanded metal riffles over Nomad matting. The sluices were suspended from a steel frame on cables. An electric motor mounted between the twin sluices created a horizontal circular "panning" motion. The circular motion was 5/8 inches in diameter and oscillated at 184 rpm. This panning motion kept the expanded metal riffles from packing with these extremely fine White Channel gravels. The sluice concentrates were cleaned up after 69 hours of operation and were upgraded with a Long Tom, and then by hand panning.

The sluice discharges were easily sampled with a hand held bucket and 420 sample increments were collected over a three day period. Individual sample increments were collected alternately into two (interpenetrating) subsamples.

8.1.3 Site C

Site C was a low bench river placer deposit. Coarse pay gravels were scraped and pushed from the valley floor up a loading ramp with a bulldozer. A large front end loader mucked from the opposite face of the ramp to the feed hopper.

From the feed hopper the material was pushed forward and washed by a manually controlled monitor. Well-washed oversize material slid off a round bar grizzly. The minus 6 inch diameter gravels and wash water continued to the single run sluice. A bulldozer pushed waste away from the end of the sluice and grizzly discharge.

The sluice consisted of a series of three sluice runs three feet wide and a total of 30 feet long. The sluice runs consisted of sections of 3/4 and 1/2 inch punch plate suspended 2 inches above 1.5 and 2 inch angle iron (Hungarian) riffles spaced 4 inches apart. The lowest 8.5 feet of the bottom run did not have any punch plate above the Hungarian riffles. Backed Nomad matting was located directly below the riffles.

The sluice operated with high turbulence in the sluice run due to the passage of coarse (minus six inch diameter) rock and the accompanying volumes of pay gravels and water. Despite this high turbulence the lowest Hungarian riffles were 80% full after 15 hours of sluicing. The sluicebox concentrates were cleaned up after 48 hours of sluicing and were upgraded with a duplex jig.

The tailings were sampled with a three inch pipe sample cutter (Figure 1) connected to flex hose which discharged into a 5 gallon container. There were 145 sample increments taken over 47.5 second periods which were collected alternately into two (interpenetrating) subsamples.

8.1.4 Site D

The Site D operator was mining right limit creek channel of his wide valley deposit. The large dozer scraped and ripped 5 feet of the schistose bedrock and placer gravels out of the channel and pushed it into piles.

A large front end loader mucked from the piles and fed the Derocker grizzly/hopper. The Derocker was 10 feet in width and 18 feet long. Its rollers were sloped at 3 inches/foot and spaced at 2.5 inches. The Derocker rotated and thoroughly washed the pay gravels, screened off the coarse material, and provided an even feed to the sluice runs. The Derocker's oversize and the sluicebox's tailings were carried away by a wheeled loader.

The undersize from the Derocker was directed through a series of sluice runs totaling 30 feet in length. The runs varied in width from 1.6 feet wide at the top to 8 feet at the discharge (a set of four 2 feet wide parallel sluice runs). Alternating riffle sections included Hungarian riffles over expanded metal and single and double thicknesses of expanded metal riffles. All of the riffles in the sluice runs were underlain by paper carton conveyer belting which was formed with small spherical pockets. The operator preferred this matting to Nomad or cocoa matting because it was easier to clean.

The pay gravels contained a high proportion of hematite and pyrite which tended to pack the sluices riffles. Packing was minimized by hand raking the riffles every two hours. The riffles were cleaned up after 38 hours of sluicing and upgraded in a small duplex jig. The operators commented that most of the larger nuggets had locked quartz.

Sampling was very tedious and was carried out over the entire 48+ hour mining period, 24 hours a day. Samples consisted of five consecutive cuts by a sampling pail from each of the four final sluice runs. Sample screening was very slow due to the abundance of -2.5 inch, +4 mesh material.

8.1.5 Site E

Site E was a low bench creek placer deposit with coarse placer gold. The operator monitored black muck and waste from the tall face of the left limit deposit. Pay gravels were bulldozed from the low bench into stock piles near the sluice box. Large boulders were sorted out of the feed as practical. The sluice box was fed slowly from the end with a small bulldozer. This same dozer would periodically travel to the sluice discharge to push the accumulated tailings away.

Pay gravels were washed with a manned monitor. A concerted effort was made to even the feed to the box by signaling the dozer operator and directing the spray of the monitor. The feed rate was relatively even except at the start of each feed cycle when a small wave of slurry would flow down the sluice .

The sluice box consisted of a slick dump box area 7 feet wide and 9 feet long, followed by two sections of riffles. The top riffle section was a total of 20 feet long and varied from 6 feet to 3 feet in width. Its riffles consisted of punch plate directly above expanded metal and backed Nomad matting. The punch plate was drilled with a variety of round and slotted holes up to 3/4 inches wide.

The bottom sluice box section was 3 feet wide and 16 feet long. These riffles consisted of 3 inch angle iron (Hungarian) riffles at a 6 inch spacing on top of backed Nomad matting. The entire sluice run operated with relatively high turbulence due to the high volumes of water which were required to push the coarse rocks down the sluice runs.

The top sluice section was cleaned up after 20 hours of sluicing. The bottom sluice section was cleaned up after a total of 42 hours of sluicing. The sluice box concentrates were upgraded with a Long Tom and gold wheel. The concentrates from the top and bottom sluices were retrieved, weighed, and screened separately.

The tailings were sampled with a three inch pipe sample cutter (Figure 1) connected to flex hose which discharged into a 5 gallon container. The 125 sample increments were collected alternately into two (interpenetrating) subsamples. The feed rate was determined by surveying the feed stock pile before and after the sampling period.

8.1.6 Site F

Site F was a low bench river placer deposit. Pay gravels were scraped and pushed from the valley floor up a gravel ramp with a bulldozer. As the dozer pushed, the gravels spilled over the front of the ramp onto the side of the dump box. Only a small fraction of the bulldozer's time was spent feeding the sluice and it was unable to separate large boulders from the sluicebox feed. This resulted in wide feed rate variations.

The triple run sluice box consisted of an upper dump box area which received the pay gravels and wash water, a distributor, a center run and two side runs. Every section except the distributor contained expanded metal riffles directly above backed Nomad matting. The dump box was 7.6 feet wide by 24 feet long and covered with 3/4 inch punch plate. Water was directed down the dump box from a stationary horizontal monitor. Some of the minus 3/4 inch material and slurry ran through the distributor's 8 by 7.6 feet wide punch plate and was split into the two side run sluices. Coarse pay gravels and slurry which remained above the distributor were directed into the center run sluice.

The two side run sluices were each 20 feet long and 2.7 feet wide. The center run sluice was also 20 feet long and 2.7 feet wide. Its riffles also included a 16 foot length of 2 x 2 inch angle iron (Hungarian) riffles spaced at 4 inch intervals. The lowest 4 feet of the center run contained 5/8 inch punch plate which directed the underflow to an undercurrent sluice run. The high volumes of water which were required to push the large boulders down the center sluice resulted in extreme turbulence in the center run sluice.

The dumpbox, center and side runs were cleaned before and after the 60 hour sluicing period. The side runs were also cleaned up twice during the sampling period. The operator upgraded the sluicebox concentrates with a duplex jig. The concentrates were retrieved, weighed and screened separately.

Separate tailings samples were collected from the side runs and the center runs during the day shift over a four day sluicing period. The individual sample increments were collected alternately into two (interpenetrating) subsamples. The side runs were easily sampled with a hand held bucket and 210 increments were collected over the sampling period. The center run was much more difficult to sample due to the frequent passage of very coarse rocks. These samples were collected in a large steel clean up box which was connected to the front of a bulldozer. Only 10 increments were collected due to the large volumes and time required for each increment from the center sluice.

8.2 PROCESSING EQUIPMENT

The following is a brief tabulation of the processing equipment at each of the sites sampled (A-F). Site E is divided into its top and bottom sluice runs. While Site F is divided into its dump box, side runs, and center run. All measurements are in Imperial units. The previous section (8.1) provided some additional details.

TABLE 8.2 PROCESSING EQUIPMENT

FEEDING	Site A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Wheeled Loader	yes	yes	yes	yes	Bulldozer		Bulldozer		
Hopper	yes	yes	yes	Derock	N/A	N/A	N/A	N/A	N/A
Volume yd3	4.3	7.2	?	?	N/A	N/A	N/A	N/A	N/A

Notes: Derock refers to a Derocker grizzly/screen used at Site D; N/A means not present or not applicable.

GRIZZLY	Site A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Length ft	6.5	N/A	4.9	18.0	N/A	N/A	N/A	N/A	N/A
Width ft	9.0	N/A	6.8	10.1	N/A	N/A	N/A	N/A	N/A
Area ft2	58.5	N/A	33.3	182.3	N/A	N/A	N/A	N/A	N/A
Spacing in	8.0	N/A	6.0	2.5	N/A	N/A	N/A	N/A	N/A

SCRUBBER	Site A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Type	N/A	Rotary	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Length ft	N/A	20.0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Diameter ft	N/A	3.8	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Speed RPM	N/A	9.2	N/A	N/A	N/A	N/A	N/A	N/A	N/A
% Critical Speed	N/A	45%	N/A	N/A	N/A	N/A	N/A	N/A	N/A

Notes: The critical speed of a scrubber is the speed at which the rocks do not tumble, but remain against the wall of the scrubber due to centrifugal force.

TABLE 8.2 Continued

SCREEN	Site A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Type	Vibrt	Rotary	PP*	Derock	PP*	N/A	PP*	PP*	N/A
Number of Decks	2	N/A	1	1	1	N/A	1	1	N/A
Length ft	4.0	3.9	21.5	18.0	19.7	N/A	24.5	8.0	N/A
Width (Dia) ft	5.0	(3.8)	3.1	10.1	4.9	N/A	7.6	7.6	N/A
Area ft2	20.0	44.2	66.7	182.3	95.9	N/A	186.0	46.1	N/A
Final Spacing in	0.19	1.00	0.5	2.5	0.75	N/A	0.75	0.75	N/A
Efficiency**	99%	99%	?	99%	?	N/A	?	41%	N/A

Notes: Vibrt refers to a vibrating screen deck; *PP refers to the use of stationary punch plate as a screen. The punch plate at Sites C and E, and in Site F's dump box were located in the sluice runs, directly above the riffles.

**Efficiencies for sites A through D are estimated based on visual examination of the screen oversize and undersize products. Site F's distributor efficiency has been calculated from the size analysis of the center and side run tailings samples.

SLUICEBOX	Site A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Total Length ft	15.0	23.5	30.0	29.5	19.7	15.8	24.5	20.1	20.3
Average Width ft	1.8	7.3	3.1	6.8	4.9	3.1	7.6	5.4	2.7
Total Area ft2	26.9	172.1	93.0	199.6	95.9	48.5	187.0	110.0	54
Overall Slope in/f	3.0	1.1	2.0	1.5	2.3	2.3	2.0	1.3	2.61
Riffle type	Exp	Exp	Hunga	Both	Exp	Hunga	Exp	Exp	Hunga
Matting	Nomad	Nomad	Nomad	Belt*	Nomad	Nomad	Nomad	Nomad	Nomad
Oscillating Sluice	No	Yes	No	No	No	No	No	No	No
Slick Plate	No	No	No	No	Yes	Yes	No	No	No

Notes: Exp refers to expanded metal riffles; Hunga refers to Hungarian riffles; Belt* refers to paper box conveyer belting which is formed with spherical pockets. A slick plate is a smooth section of steel plate located in front of riffles to calm the slurry flow.

8.3 PROCESSING PARAMETERS

The following is a brief tabulation of the pay gravel and process water flowrates. Water and feed flowrates are compared to those recommended by Poling (1986):

a) Feed rate of 8 loose cubic yards/hr per foot of sluice width.

b) Water ratio of 17 Imperial gallons/minute per loose cubic yards of pay gravels; and

c) Water rate of 170 Imperial gallons/minute per foot of sluice width.

TABLE 8.3 PROCESSING PARAMETERS

PAY GRAVELS AND PROCESS WATER FLOWRATES

Site	A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
Feed Rate Lyd3/hr	28	57	120	69	41	41	128	12	116
Sluiced Proportion	50%	89%	90%	81%	100%	100%	100%	10%	91%
/ft Sluice Width	7.9	7.0	34.8	8.2	8.4	13.2	16.8	2.3	42.9
Percent of Optimum	98%	87%	434%	103%	105%	166%	211%	28%	536%
Is Feed Surging	No	No	No	No	No	No	Yes	Yes	Yes
Process Water Igpm	85	1089	2733	1923	1755	1755	3635	988	2646
Per Lyd3/hr feed	6.0	21.4	25.4	34.5	43.0	43.0	28.4	81.3	22.8
Percent of Optimum	35%	123%	146%	198%	247%	247%	163%	467%	131%
/ft Sluice Width	47	149	882	241	360	570	478	183	980
Percent of Optimum	28%	89%	528%	144%	216%	341%	286%	110%	587%

Notes: The sluiced proportion refers to the proportion of the pay gravels which are screen undersize; The surging of feed was determined by comparison of sample increment volumes.

The volume of pay gravels sluiced was 4 times the optimum at Site C. This same overloading is evident in terms of water volume per width of sluice even though the mix of pay gravels and water was satisfactory.

Site F's stationary punch plate distributor was very inefficient and directed most of the fine gravels to the center run. This resulted in overloading the center run and underutilizing the side runs.

Extremely high volumes of water were required at Site F to push boulders down the center run without raking. Lower volumes were used at Site E, however several of the larger boulders had to be raked by hand.

8.4 SAMPLING STATISTICS

The following table displays the total number of sample increments that were collected per pair of cumulative (interpenetrating) samples and the number of operating hours or loose cubic yards of pay gravels each pair represents. The standard deviation is a measure of the variability of the amount of gold present in each pair of tailings samples. It is expressed as a percent of the average (mean).

TABLE 8.4 SAMPLING STATISTICS

Site	A	B	C	D	E	E Bot	F	F Side	F Center
No of Increments	284	420	145	1274		125		210	10
Both Pairs of Sample Represents									
Operating Hours	0.076	0.027	0.059	0.086		0.123		0.060	0.031
Loose Cubic Yards	2.2	1.5	7.1	5.9		5.0		0.7	3.7
Standard Deviation	8%	17%	56%	N/A		8%		48%	38%

Notes: The limited number of sample increments for Site F's center run is due to the use of a large steel bucket to sample the coarse material.

Sites C and F had a high standard deviation due to a limited number of very coarse gold particles "nugget effect" which were recovered from the tailings. Fortunately, these two sites also had the highest losses of gold.

8.5 SIZE DISTRIBUTION OF GRAVELS AND GOLD PARTICLES

The following table displays pay gravel size data determined by a combination of the results of: dry sieve test of small samples; wet screening of tailings samples; bulk screening tests; and bucket counts of screen oversize where applicable.

TABLE 8.5.1 SIZE DISTRIBUTIONS OF PAY GRAVELS (Figure 6)

Site	A	B	C	D	E	F
Proportion Greater Than Sieve						
+4 Mesh percent	50%	19%	81%	54%	57%	65%
+8 Mesh percent	5%	7%	6%	8%	8%	5%
+16 Mesh percent	2%	5%	5%	4%	4%	3%
+32 Mesh percent	8%	22%	5%	6%	7%	7%
-32 Mesh percent	35%	47%	3%	28%	24%	21%
Total	100%	100%	100%	100%	100%	100%

Site	A	B	C	D	E	F
Cumulative Percent Passing Sieve						
+4 Mesh percent	50%	81%	19%	46%	43%	35%
+8 Mesh percent	45%	74%	13%	38%	35%	30%
+16 Mesh percent	43%	69%	8%	34%	31%	28%
+32 Mesh percent	35%	47%	3%	28%	24%	21%

Notes: The table and accompanying Graph (Figure 6) indicates that Site B's White Channel gravels were much finer than the others. Site A had the next finest pay gravels and Site C had the coarsest gravels.

The following table displays the cumulative percent of gold particles which are finer than the indicated sieve. This table does not include information about gold particles coarser than 14 mesh to protect the operator's head grades.

TABLE 8.5.2 SIZE DISTRIBUTION OF GOLD PARTICLES (Figure 7)

CUMULATIVE PERCENT OF RECOVERED GOLD PARTICLES FINER THAN SIEVE

Particle Diameter	Site Mesh Size	A*	B	C	D	E	F
4.76							
1.19	14	92.7%		76.3%	90.7%	72.2%	77.8%
0.841	20	84.6%	90.6%	50.0%	71.8%	42.2%	48.3%
0.595	28	53.5%	64.7%	34.2%	44.5%	16.4%	27.3%
0.297	48	29.8%	26.7%	9.6%	5.2%	1.4%	4.2%
0.210	65	14.2%	15.3%	5.3%	1.4%	0.6%	1.5%
0.149	100	4.6%	5.4%	2.8%	0.3%	0.1%	0.3%
0.105	150	1.2%	1.9%	1.1%	0.0%	0.0%	0.0%

Notes: Site A data are derived from an assumed head grade. Site B has the finest gold and Site E has the coarsest.

B.6 COREY SHAPE FACTORS

The Corey Shape Factor is a measure of the flatness of placer gold. It is the ratio of the thickness of a gold flake to the square root of its area. For example, the C.S.F. of a sphere is 1 and of a dime is 0.05. The following tables display the C.S.F. of the gold recovered by the sluice and the gold lost to the tailings.

The following table displays the Corey Shape Factors of various sizes of gold particles recovered by the sluice and lost to the tailings.

TABLE B.6 COREY SHAPE FACTORS

GOLD RECOVERED BY SLUICEBOX

Particle Diameter	Site Mesh Size	A	B	C	D	E Top	E Bot	F Dump	F Side	F Center
1.19	14	N/A	N/A	0.13	0.10	0.30	0.30	0.15	0.09	0.10
0.841	20	N/A	N/A	0.09	0.12	0.30	0.35	0.20	0.15	0.10
0.595	28	N/A	N/A	0.24	0.18	0.30	0.35	0.20	0.18	0.15
0.297	48	N/A	0.30	0.24	0.18	0.30	0.40	0.25	0.18	0.40
0.210	65	N/A	0.20	0.33	0.22	0.40	0.40	0.25	0.25	0.40
0.149	100	N/A	0.30	0.41	0.32	0.40	0.40	0.30	0.25	0.30
0.105	150	N/A	0.25	0.59	0.35	0.50	0.50	0.30	0.35	0.30
	-150	N/A	0.45	0.39	0.40	0.50	0.60	0.35	0.35	0.35

GOLD LOST TO TAILINGS

Particle Diameter	Site Mesh Size	A	B	C	D	E Bot	F Side	F Center
0.297	48	0.20	0.20	0.20	0.20	0.20	0.20	0.30
0.210	65	0.20	0.20	0.20	0.20	0.20	0.20	0.20
0.149	100	0.20	0.25	0.20	0.25	0.25	0.20	0.25
0.105	150	0.25	0.50	0.25	0.30	0.30	0.35	0.35
	-150	0.35	0.50	0.40	0.40	N/A	N/A	N/A

Notes: The Corey Shape Factors of the recovered gold are similar to the lost gold, except for Site E where the gold lost to the tailings is somewhat flatter.

8.7 DISTRIBUTION OF GOLD VALUES

The following table displays the proportions of gold in each size fraction recovered by the sluice and lost to the tailings.

TABLE 8.7 DISTRIBUTION OF GOLD VALUES

(Figures 9, 11, 13, 15, 17, 19)

GOLD VALUES RECOVERED BY SLUICES

Particle Diameter	Site Mesh Size	A*	B	C	D	E Top	E Bot	F Dump	F Side	F Center
1.19	14	13.0%		23.7%	9.3%	25.9%	1.9%	12.2%	7.1%	0.8%
0.841	20	2.4%	9.4%	25.3%	18.1%	27.8%	1.9%	18.7%	8.6%	1.1%
0.595	28	22.4%	25.9%	15.8%	27.3%	24.2%	1.6%	15.3%	4.1%	1.6%
0.297	48	31.4%	23.3%	18.8%	36.7%	11.8%	0.8%	12.0%	3.8%	0.4%
0.210	65	15.1%	8.2%	2.6%	3.2%	0.4%	0.0%	1.3%	0.1%	0.1%
0.149	100	9.2%	7.8%	0.5%	0.7%	0.1%	0.0%	0.3%	0.0%	0.0%
0.105	150	3.2%	2.0%	0.1%	0.1%	0.0%	0.0%	0.0%	0.0%	0.0%
	-150	1.1%	0.8%	0.1%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Total Recovered		97.8%	77.4%	86.9%	95.4%	90.3%	6.2%	59.7%	23.8%	4.1%

Total Site E 96.4%Total Site F 87.6%

GOLD VALUES LOST TO THE TAILINGS

Particle Diameter	Site Mesh Size	A*	B	C	D	E Top	E Bot	F Dump	F Side	F Center
1.19	14	0.0%	0.0%	0.0%	0.0%		0.0%		0.1%	2.0%
0.841	20	0.0%	0.0%	1.1%	0.9%		0.4%		0.7%	0.4%
0.595	28	0.0%	0.0%	0.0%	0.0%		0.0%		0.0%	0.0%
0.297	48	1.0%	14.7%	5.7%	2.6%		2.4%		2.1%	4.8%
0.210	65	0.5%	3.2%	1.7%	0.6%		0.4%		0.1%	1.0%
0.149	100	0.4%	2.1%	2.0%	0.4%		0.3%		0.1%	0.8%
0.105	150	0.2%	1.4%	1.5%	0.2%		0.1%		0.0%	0.3%
	-150	0.1%	1.1%	1.1%	0.0%		0.0%		0.0%	0.0%
Total Lost Values		2.2%	22.5%	13.1%	4.6%		3.6%		3.1%	9.3%

Total Site F 12.4%

Notes: Site A data are derived from assumed grades.

Extremely high losses occurred in Sites B, C and F. The high losses at Site B are due to its White Channel's extremely fine gold and gravel distributions. The high losses at Site C and the center run of Site F are due to extreme overloading, extreme turbulence, the passage of coarse rocks and the use of Hungarian riffles. Site F's center run looses twice as much gold than it retains. Site F's side run has much lower losses but still misses coarse gold particles.

8.8 RECOVERY OF SLUICING SYSTEMS

The recovery of the various sluiceboxes is displayed in the following chart. Where the recoveries of consecutive sections of a sluicebox are considered, the gold removed by the first section is not included in the feed to the second section. The data from Sites E and F demonstrate the value of keeping the gold from different sluicebox sections separate.

TABLE 8.8 RECOVERY OF SLUICING SYSTEMS
(Figures 8, 10, 12, 14, 16, and 18)

Particle Diameter	Site Mesh Size	A*	B	C	D	E Top	E Bot	F Dump	F Side	F Center
4.76	4	100%	100%	100%	100%	34%	100%	100%	100%	100%
3.36	6	100%	100%	100%	100%	77%	100%	100%	100%	100%
2.38	8	100%	100%	100%	100%	85%	100%	91%	100%	100%
1.68	12	100%	100%	100%	100%	90%	100%	70%	100%	100%
1.19	14	100%	100%	100%	100%	93%	100%	55%	98%	29%
0.841	20	100%	100%	96%	96%	93%	83%	63%	93%	72%
0.595	28	100%	100%	100%	100%	94%	100%	73%	100%	100%
0.297	48	96%	61%	77%	93%	79%	24%	52%	65%	8%
0.210	65	97%	72%	60%	84%	46%	5%	50%	49%	7%
0.149	100	96%	78%	21%	66%	30%	6%	22%	39%	2%
0.105	150	95%	58%	8%	43%	24%	5%	3%	6%	2%

GOLD LOST TO THE TAILINGS	A	B	C	D	E	F
Ounces/hr	0.016	0.118	0.262	0.072	0.046	0.274
Can$ per Hour	$8	$57	$126	$34	$22	$132
Can$1000/season	$10	$73	$101	$62	$6	$238

RECOVERABLE LOSSES	A	B	C	D	E	F
Ounces/hr	?	?	0.130	?	?	0.182
Can$ per Hour	?	?	$63	?	?	$87
Can$1000/season	?	?	$50	?	?	$157

Notes: Site A has the best recovery with 95% recoveries of gold as fine as 150 mesh. With the exception of Site B's White Channel gravels, the recovery of the sluices improves with finer screening of the pay gravels.

Site E's bottom sluice contains Hungarian riffles which recovered more of the gold coarser than 1 mm while the top section containing expanded metal riffles has superior recoveries of gold finer than 1 mm. This characteristic is mirrored in Site F's center and side runs. The losses of coarse gold in Site F's side runs may be due to this deficiency of expanded metal riffles.
Site C and F's recoverable losses are based on a comparison with the recovery of Site F's side runs and Site D and E's recovery characteristics. Sites A and B may be losing gold coarser than 1 mm. Site E's recovery system is well suited to its coarse gold.

9 REFERENCES

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Table 1
A Comparison of Standard Sieve Sizes

Tyler mesh	National Bureau of Standards inches	A.S.T.M. Sieve Opening mm	A.S.T.M. Sieve Opening inches
	4	101.60	4.00
	3	76.10	3.00
	2	50.80	2.00
	1	25.40	1.00
	3/4	19.00	0.75
	1/2	12.70	0.50
1/4	1/4	6.35	0.25

Tyler mesh	N.S.B. number	A.S.T.M. open mm	inches
3		6.730	0.2650
4	4	4.760	0.1874
6	6	3.360	0.1323
8	8	2.380	0.0937
10	12	1.680	0.0661
14	16	1.190	0.0469
20	20	0.841	0.0331
28	30	0.595	0.0234
35	40	0.420	0.0165
48	50	0.297	0.0117
65	70	0.210	0.0083
100	100	0.149	0.0059
150	140	0.105	0.0041
200	200	0.074	0.0029
270	270	0.053	0.0021
400	400	0.037	0.0015

Table 2
Gold Value Equivalents

Grams/ Cubic Meter g/m3	Troy Ounces/ Cubic Yard oz/yd3	Canadian Dollars/ Cubic Yard Can$/yd3 @$482/oz
1	0.02459	$11.85
40.67	1	$482
0.08438	0.002075	$1.00

Table 3
Placer Mining Measurement Conversions

WEIGHT EQUIVALENTS

Milligram mg	Gram g	Kilogram kg	Pound lb	Metric Tonne t	Short Ton ton
1	0.001	1.000E-06	2.205E-06	1.000E-09	1.102E-09
1000	1	0.001	0.002205	0.000001	1.102E-06
1000000	1000	1	2.205	0.001	0.001102
453600	453.6	0.4536	1	0.0004536	0.0005
1.0E+09	1000000	1000	2205	1	1.102
9.1E+08	907200	907.2	2000	0.9072	1

LENGTH EQUIVALENTS

Micron u	Millimeter mm	Meter m	Inch "	Foot '	Yard yd
1	0.001	1.000E-06	3.937E-05	3.281E-06	1.094E-06
1000	1	0.001	0.03937	0.0032808	0.001094
1000000	1000	1	39.37	3.2808	1.094
25400	25.4	0.0254	1	0.08333	0.02778
304800	304.8	0.3048	12	1	0.33333
914400	914.4	0.9144	36	3	1

VOLUME EQUIVALENTS

Milli- liter ml	Liter l	Cubic Meter m3	U.S. Gallon USg	Imperial Gallon g	Cubic Yard yd3
1	0.001	0.000001	0.0002642	0.0002200	0.0000013
1000	1	0.001	0.2642	0.2200	0.001308
1000000	1000	1	264.2	220.0	1.308
3785	3.785	0.003785	1	0.8327	0.004951
4546	4.546	0.004546	1.201	1	0.005946
764600	764.6	0.7646	201.99	168.19	1

FLOWRATE EQUIVALENTS

Liter/ second l/s	Cubic Meter/ second m3/s	U.S. Gallon/ minute USgpm	Imperial Gallon/ minute gpm	Cubic Feet/ second cfs
1	0.001	15.85	13.20	0.03531
1000	1	15851	13198	35.31
0.06309	0.00006308	1	0.8327	0.002228
0.07577	0.00007576	1.201	1	0.002676
28.32	0.02832	448.9	373.8	1

Figure 1 Sampling Screened Gravèls

from Merks 1987

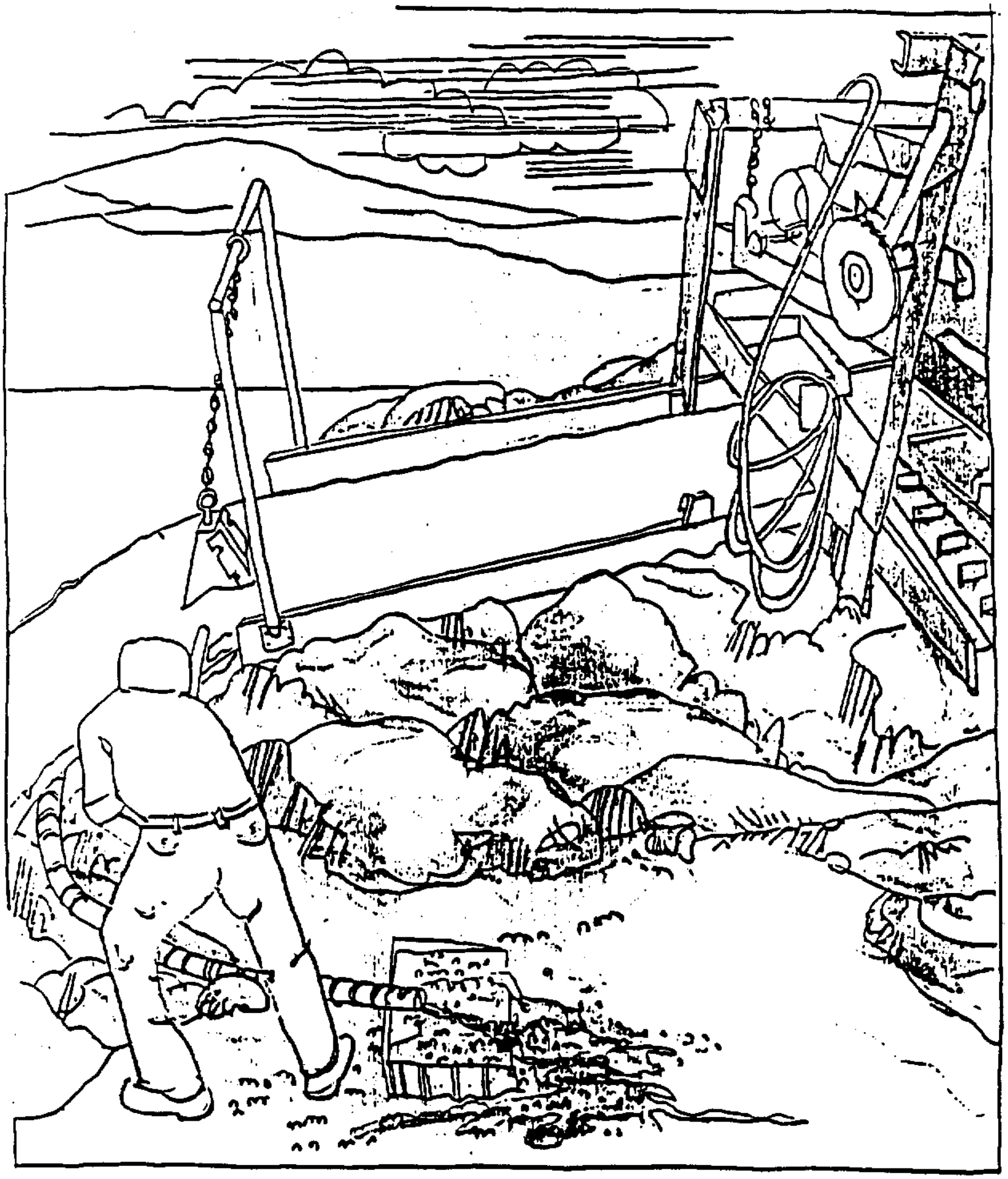


Figure 2 **EFFECT OF SCREENING ON GOLD RECOVERY** (Yukon)

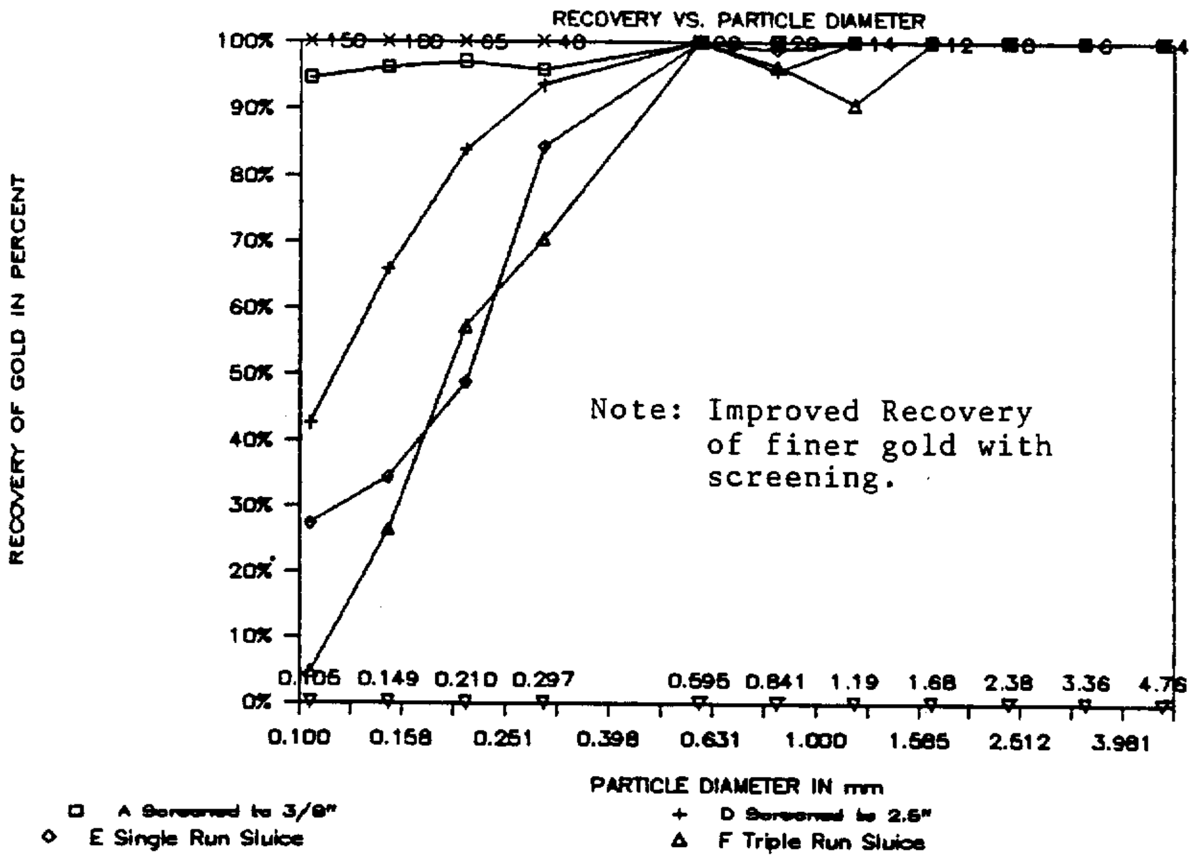


Figure 3 **EFFECT OF SCREENING ON GOLD RECOVERY** (Soviet)

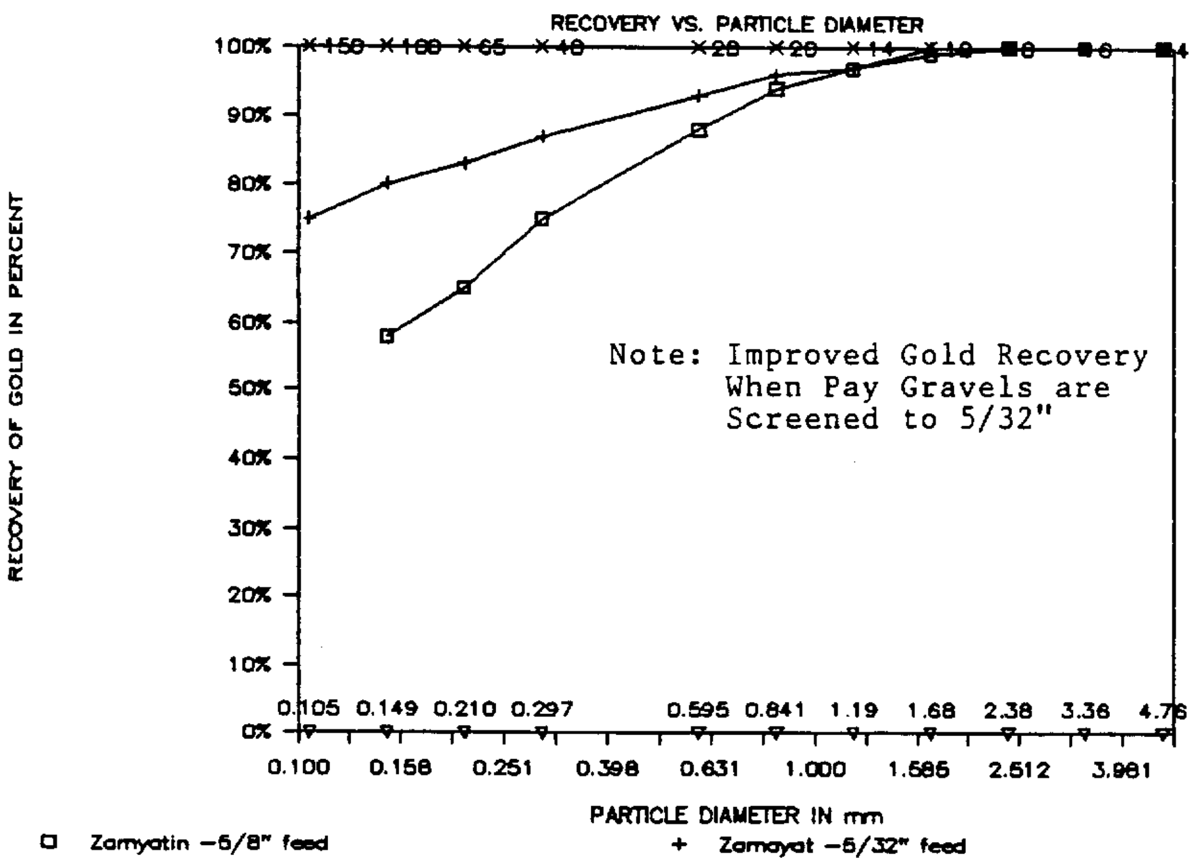


Figure 4 EFFECT OF RIFFLE TYPE ON GOLD RECOVERY

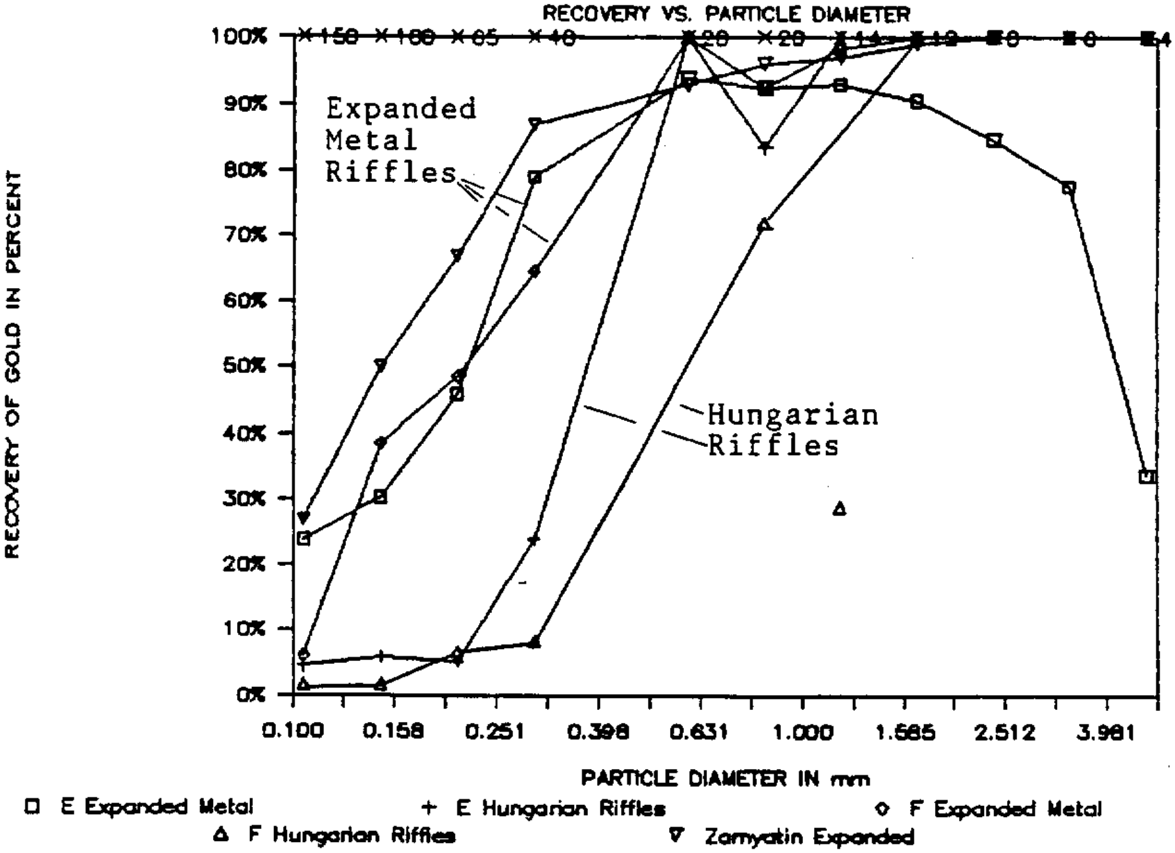
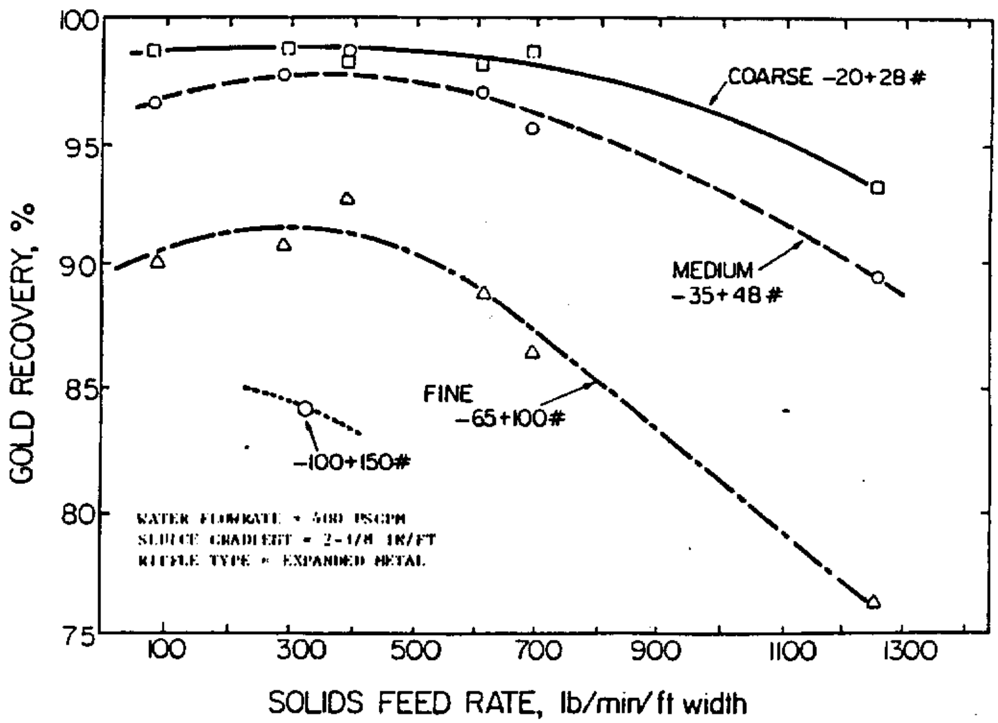


Figure 5 EFFECT OF FEED RATE ON GOLD RECOVERY



from Poling 1986

Figure 6 **FEED GRAVEL SIZE DISTRIBUTION**

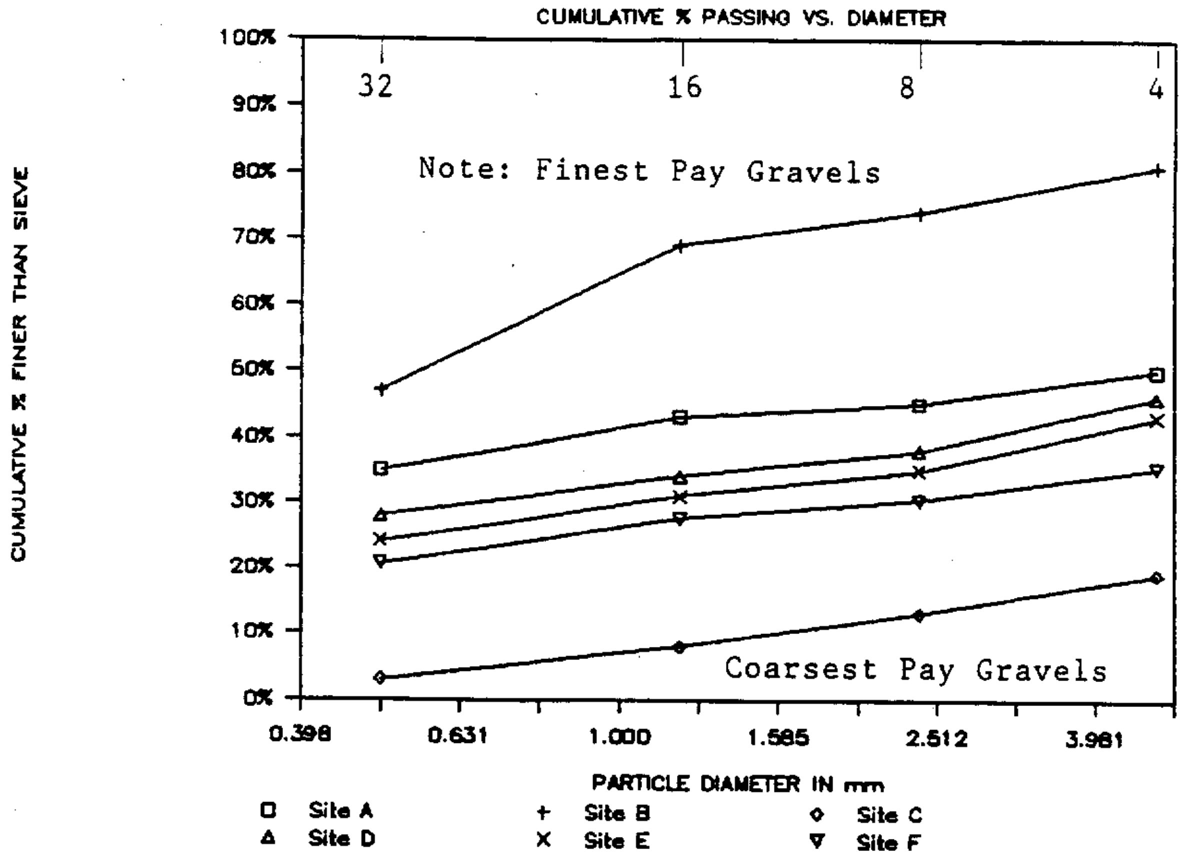


Figure 7 **GOLD PARTICLE SIZE DISTRIBUTION**

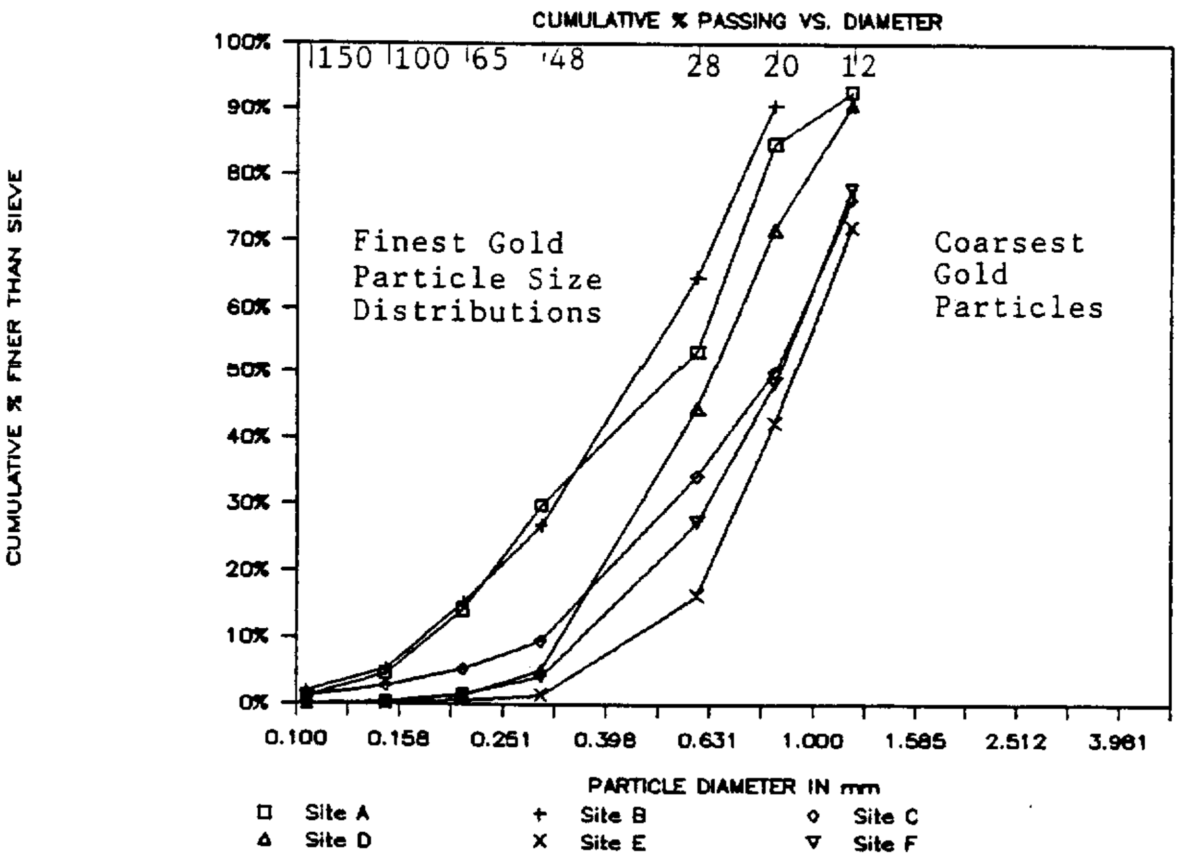


Figure 8 **SITE A GOLD RECOVERY**

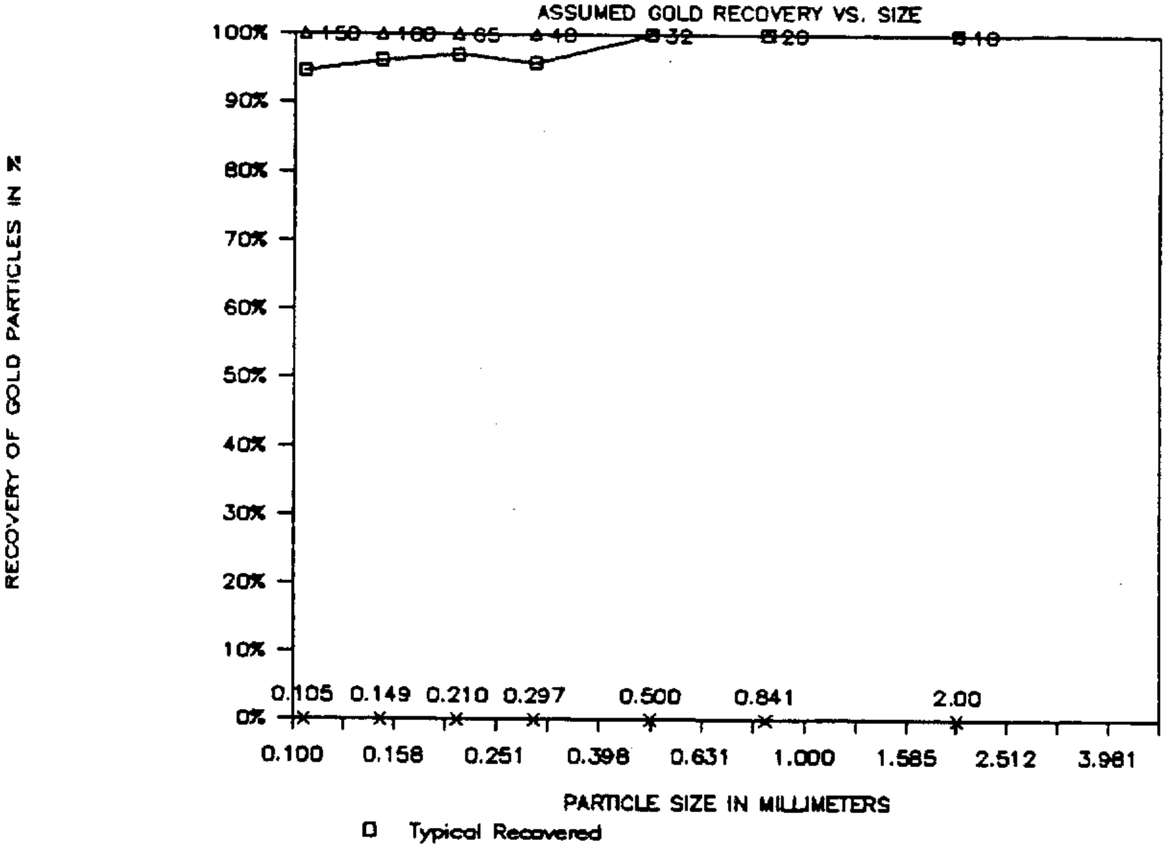
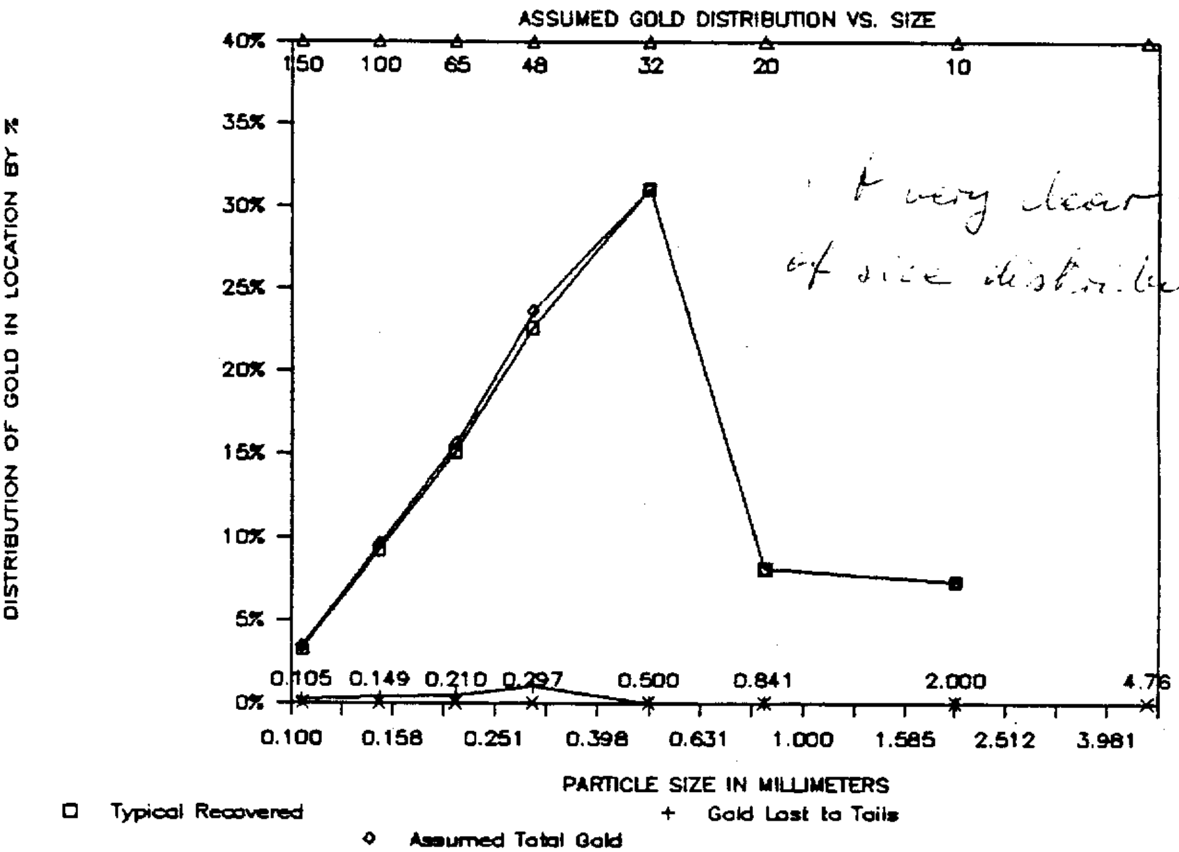


Figure 9 **SITE A DISTRIBUTION OF GOLD VALUES**



A very clear representation of size distribution

GOLD LOSSES AT KLONDIKE PLACER MINES

Figure 10 **SITE B GOLD RECOVERY**

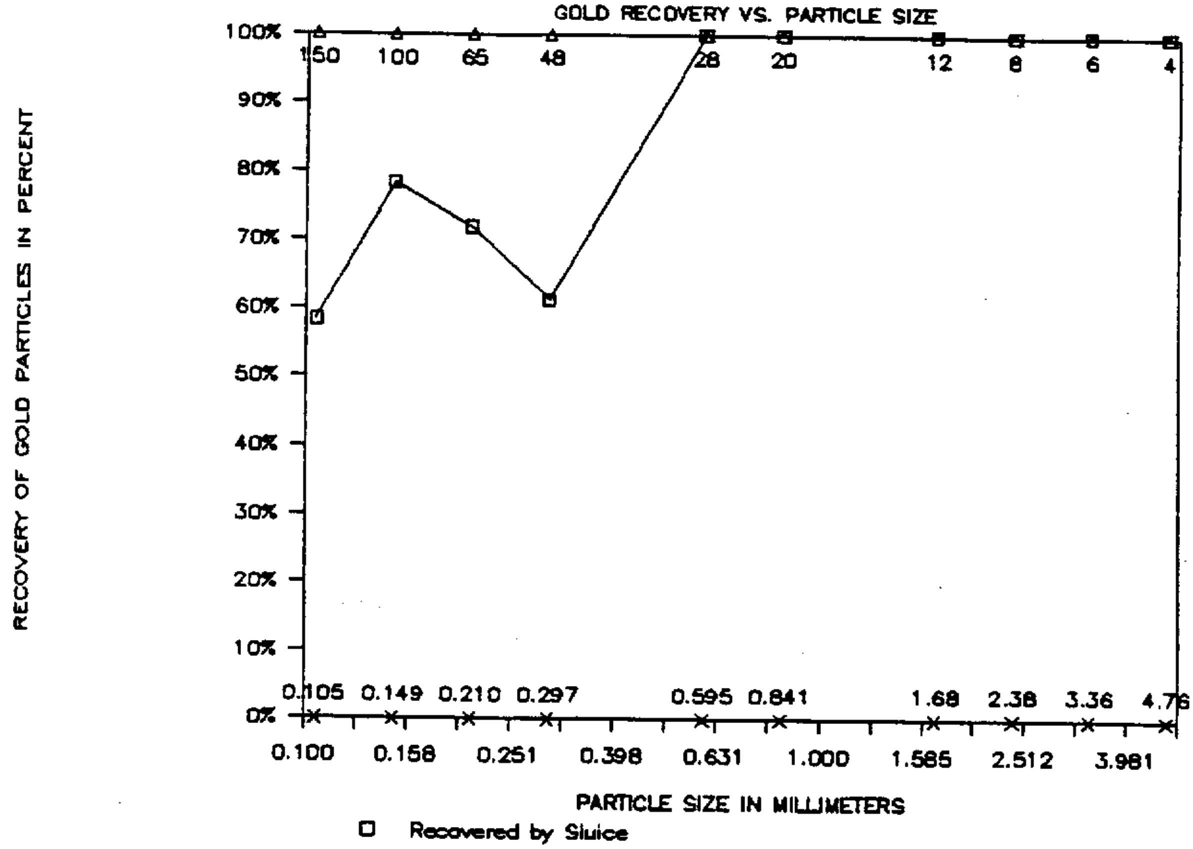


Figure 11 **SITE B DISTRIBUTION OF GOLD VALUES**

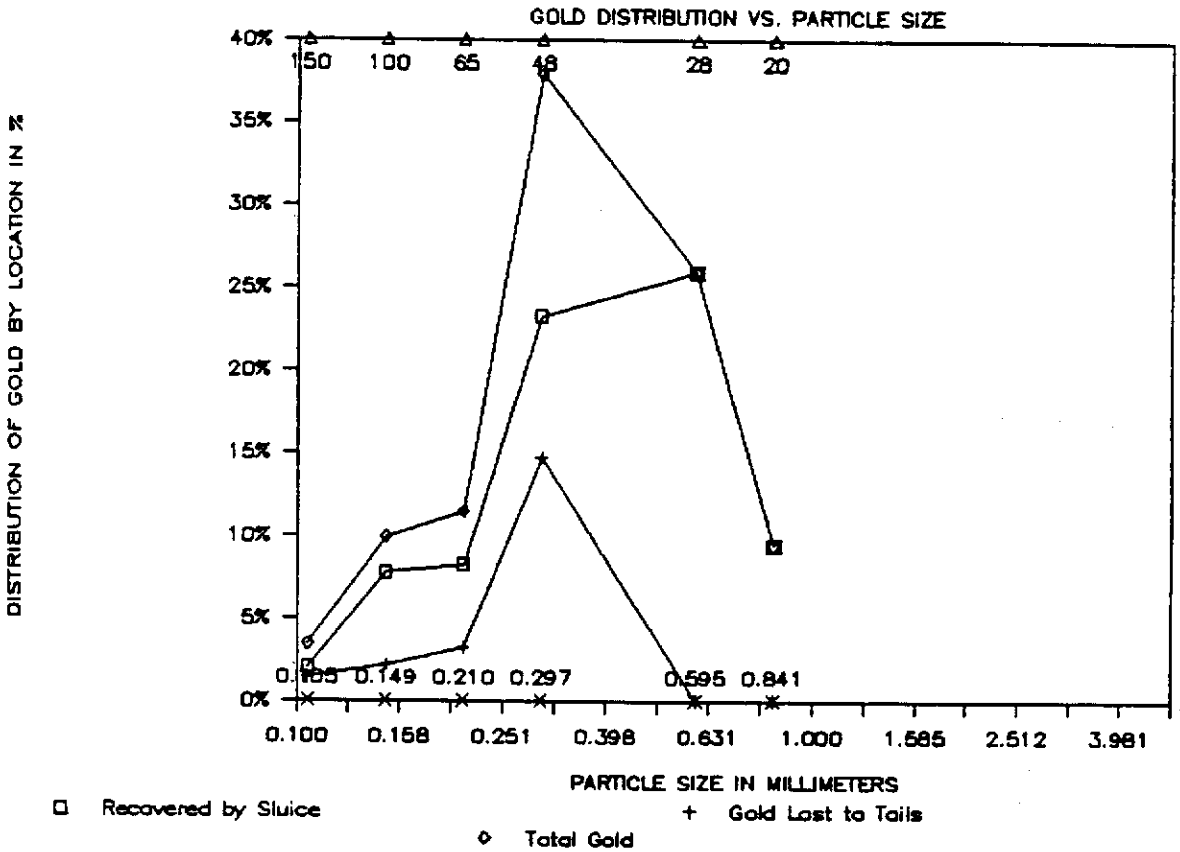


Figure 12 **SITE C GOLD RECOVERY**

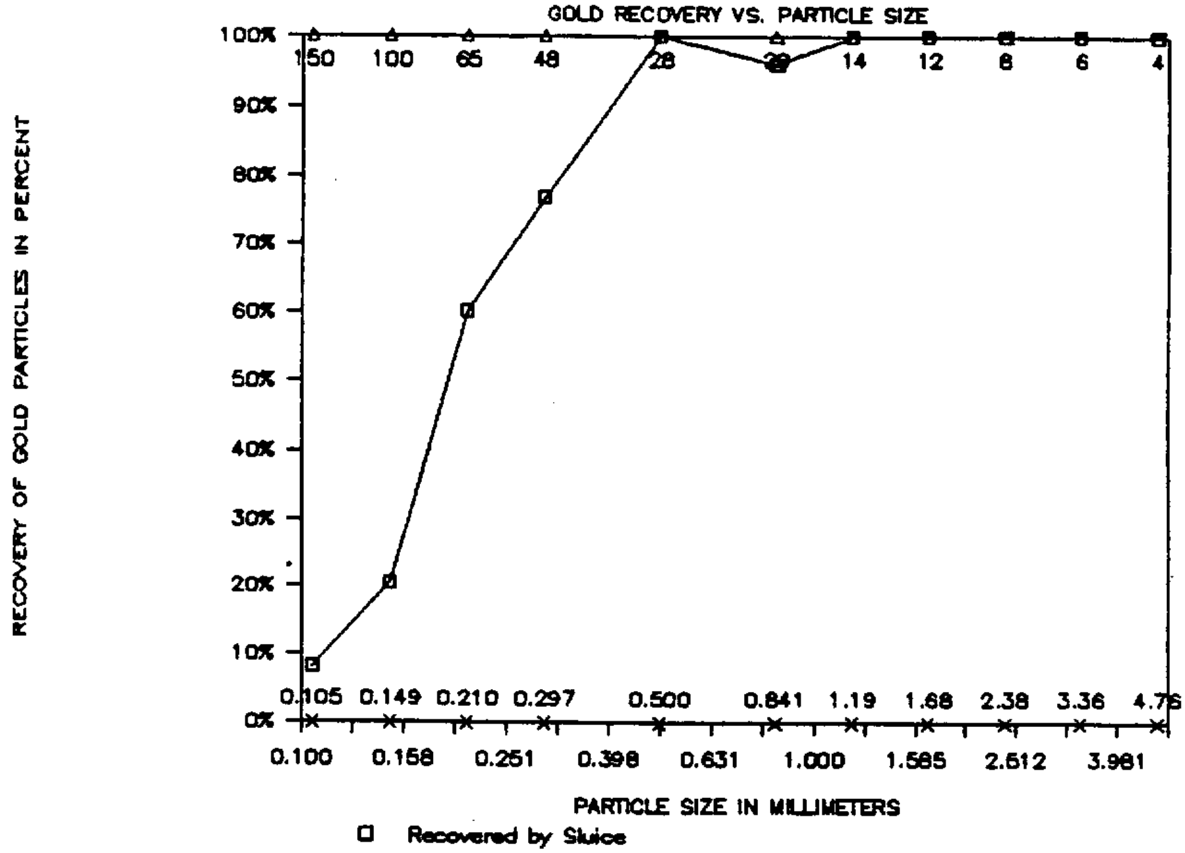


Figure 13 **SITE C DISTRIBUTION OF GOLD VALUES**

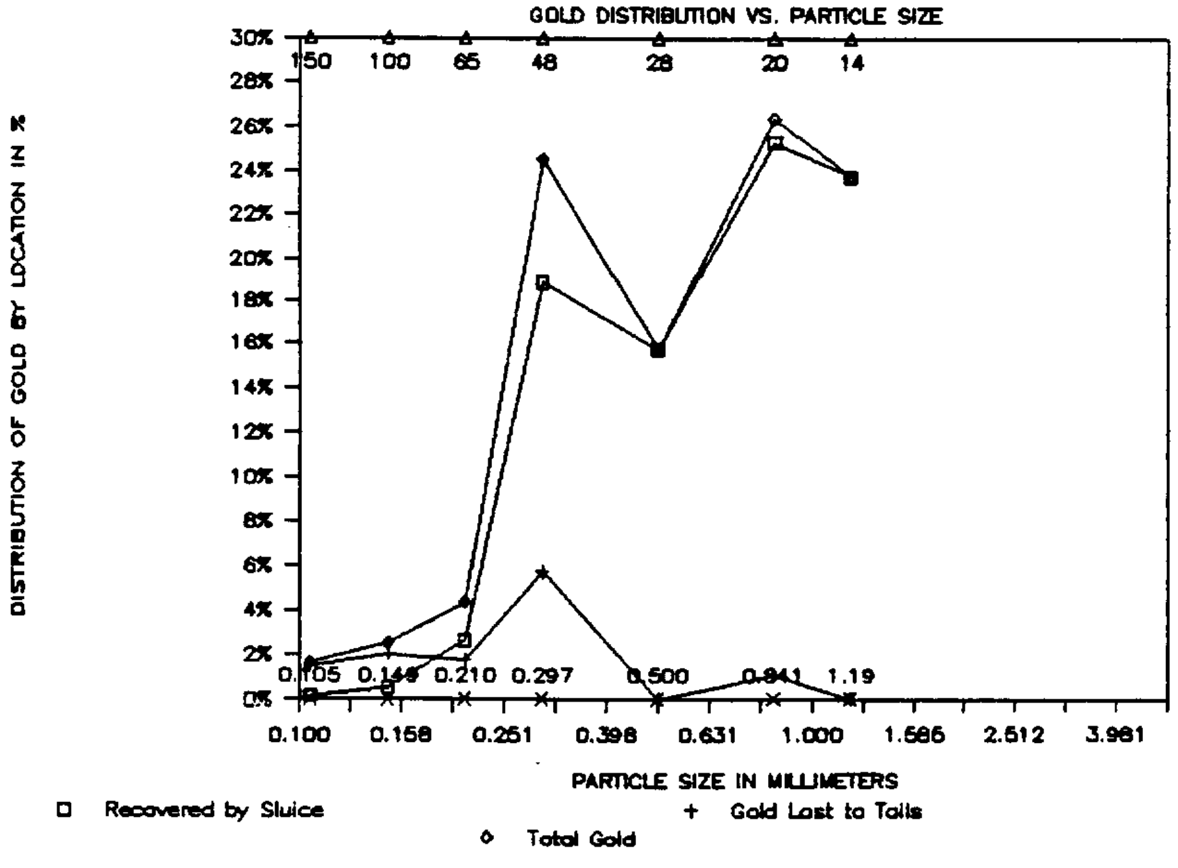


Figure 18 **SITE F GOLD RECOVERY**

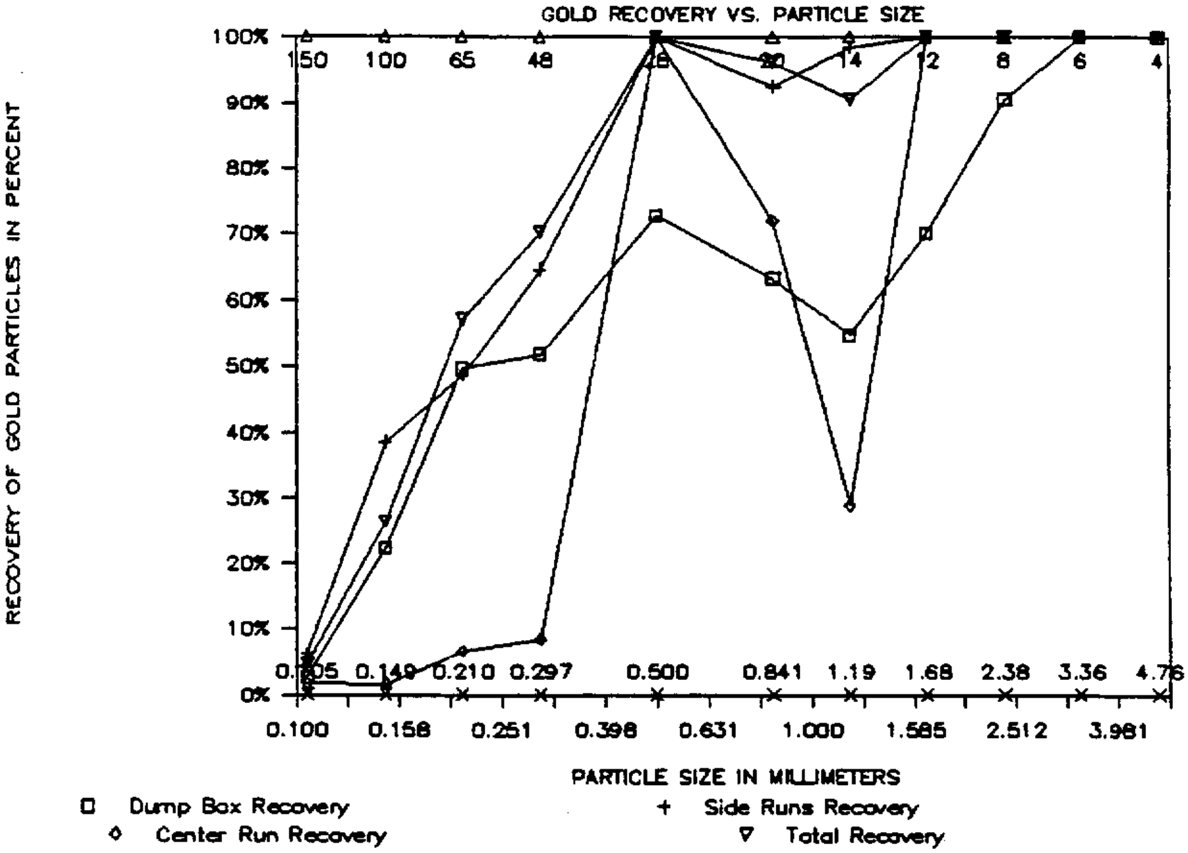


Figure 19 **SITE F DISTRIBUTION OF GOLD VALUES**