THE YUKON CONSOLIDATED GOLD CORPORATION, LIMITED

METALLURGICAL INVESTIGATION OF DREDGE SAND TAILINGS

BY

K.R. RAWLING

DAWSON, Y. Terr.
November, 1938.

SUMMARY AND CONCLUSIONS

(1) Dredge # 11 has ample capacity for recovery of plus 100 mesh gold, but finer gold than this is not effectively recovered due to the velocity of water that must be maintained on the sluices.

(2) Gold recovery increases with the gold content of the gravel, provided the riffle system is not overloaded, but the content of the gravel has no definite effect on the tailings losses.

(3) An increase in riffle capacity would not be expected to increase the recovery of fine gold appreciably.

(4) The average gold content of the samples taken from the starboard long sluices # 1, 2, and 3 on Dredge # 11, was 2.1, 5.1, and 10.0 cent per ton respectively, but the latter two figures are considered to be high due to abnormal operating conditions.

(5) A composite tail sluice sand assaying 4.5 cent per ton and the consequent percentage recovery of gold at 95.6 percent have been calculated for Dredge # 11, to represent the average performance in a cut across Hunker Valley. The figure of 4.5 cent per ton is considered to be higher than normal for the Dredge, because of abnormal conditions during part of the sampling programme. An estimated figure of 3.1 cent per ton should be closer to the average performance. This figure indicates a recovery of 97.0 percent for the Dredge, which is closer to the average recovery of 97.9 percent determined accurately on the short sluices # 1, 2 and 3, and long sluice # 1.

(6) The performance of Dredges # 4, 6, 8, 9 and 10 has been estimated from the assays of hand samples of the tailings and the gold recovered during the sampling period:

Dredge	Tailings Assay (Cent per ton)	Estimated Recovery (Percent)
4	1.4	91.8
6	2.3	94.6
8	2.3	98.1
9	3.2	96.0
10	5.6	91.1

(7) The distribution of the gold recovered on the larger dredges between port and starboard sluices was found to be very unequal; the ratio being as high as 1:4 on Dredge # 8. A method for correcting this unequality is proposed, whereby the overflow level of the distributor box is built up on the overloaded side to divert more of the flow of sand and gold to the lean side.

(8) The pyritic content of the sands was found to be quite low, and the gold content of samples of pyrite from Bonanza, Hunker and Lower Sulphur Creeks varied around 70 cent per ton. Thus pyrite is not an economic source for additional gold recovery.

(9) On the basis of the low gold content of the sand tailings from the dredges studied, and from a consideration of operating labour costs, the cost of transportation of concentrates to a central plant at Bear Creek, overhead, power and maintenance costs, and the amortization of the capital investment for plant, the retreatment of these tailings for additional gold recovery cannot be considered an economic proposition.

RECOMMENDATIONS

(1) No further consideration should be given by the Company to the possibility of gold losses in tailings being economically recoverable.

(2) The proposed modifications to distributor boxes, in order to more equally distribute the gold recovered on the dredges, should be carried out before the next operating season.

I N D E X

INTRODUCTION

The placer reserves of The Yukon Consolidated Gold Corporation, Limited in the Klondike area have been proved by churn-drilling. Drill holes were placed at 200 foot intervals along lines 500 feet apart, except in special circumstances. The mud from each drill hole was "pumped" out and panned to recover the gold content. The value per cubic yard of the gravels was then estimated from the gold recovered by panning and the volume of the drill hole. However, since panning is not inherently a method of recovering very fine gold, the value of the reserves estimated by this method can only be considered to include a small fraction of the very fine gold present in the gravels. Neither can the reserve values be considered as anything but approximate since (i) the distances between drill holes is relatively large and placer gold is not very evenly distributed and (ii) the estimation of the gold content of the gravels by panning out the very small quantity from the drill hole mud cannot be considered very accurate as a small loss incurs a large percentage error.

Nevertheless, gold recoveries on the dredges have consistently exceeded estimates from the reserve values in prospected areas, to the extent of 9.0 percent over the period 1936 to 1957 inclusive. For #6 Dredge, however, where drill holes were further apart than standard practice, the recovery has been only 75.6 percent of the estimates, and for #12 Dredge, due to operating difficulties, only 73.3 percent. Gold recovered on the dredges has been shown to be approximately 99 percent plus 200 mesh Tyler and 90 to 99 percent plus 100 mesh (1), which sizing is considerably finer than could be expected of gold recovered by panning, so that recovery by the dredges of gold finer than that indicated by panning the drill holes would partly explain the overrun of the dredges beyond the estimates.

In general, it can be said that the Company's operations in the Klondike area are geared to the recovery of coarse gold both in estimation of reserves and processing of the gravels. Recovery of gold on the dredges has become standardized to the use of coco-matting and Hungarian riffles. This equipment has come to be considered the most satisfactory for the type of gold found in the Klondike placer deposits. Amalgamation was used in the early days of the field by the small miners who were working relatively rich claims containing coarse gold and on the early dredges, but mercury losses were high. An attempt to use amalgamation in conjunction with jigs on Dredge #12 for the gold on the bench ground on Dominion Creek was not successful, and a reversion to coco-mats and Hungarian riffles was carried out. The finer Klondike gold does not appear to be readily amalgamable with short contact times.

Based on occasional spot tailings samples taken during the life of the Company, the dredges were considered to be fairly efficient in the recovery of the gold established in the reserves, and were estimated at around

98 percent recovery of gold dug. However, since the drilling cannot estimate the quantity of gold in the deposits finer than that recovered by panning, the exact quantity of fine gold in the gravels and not recovered by the dredges was not known.

In addition to an unknown quantity of gold finer than 100 mesh, there was a possible source of gold, as yet untapped, in the pyritic content of the gravels. Hunker Creek in particular has shown considerable quantities in the riffles and visible in the tailings. Since pyrite is a frequent host mineral for gold, the possibility of recovering and processing the pyrite came to be considered.

PREVIOUS TEST WORK

Sealey (2) had shown high values of gold in sands dumped behind the dredges. Also, samples of pyrite taken from drill holes on several of the gold-bearing creeks and from three of the dredges were found to have gold contents up to several ounces per ton of pyrite. However, there was some doubt about the results due to the possibility of accidental salting.

Some test work had been carried out to determine the possibilities of recovering gold escaping in the sand tailings. Sands from Dredge #8 were tested by Haughen Metallurgical Processes Ltd. (3) using an electrolytic cyanidation process. High recoveries were obtained but the work was of very doubtful value.

Tailings losses had been found to be appreciable on Dredge #6. Miller (4) showed that by jigging the sand tailings, a calculated added recovery of 0.92 cent per cubic yard or 5.7 percent of gold saved on the dredge was possible. While the percentage loss might be considered high, the cost of recovering this 1 cent per cubic yard would be too high for profitable operation. However, although only 1 cent per cubic yard was recovered from the tailings by jigging, there was still the possibility of finer gold not being recovered by the jig.

Nixon (5) reviewed the operations metallurgically in June 1957. He concluded that the dredges were operating efficiently in recovering plus 100 mesh gold, but that the quantities of water, and hence the velocity, necessary to keep the coarse sands moving on the sluices would prevent settling and recovery of gold much finer than 100 mesh. "There is no disagreement that the fine gold is not being recovered, and it is well recognized that the initial sampling of the drill holes, the design and operation of the dredges, and the cleanup procedure are all geared to the recovery of coarse (e.g. plus 100 mesh) gold. It has always been the case in the Klondike field. The pursuit of the finer gold has been considered difficult and uneconomic, and an extensive investigation would be required to prove otherwise. The "reserves" of minus 100 mesh gold may, however, be substantial."

The results of Nixon's survey of the operations are set out in his summary and conclusions as follows:

i) The Yukon dredging operation appears to be making an efficient recovery of coarse (e.g. plus 100 mesh) gold at an extraordinarily low cost, considering all the complicating factors, of 4.7 - 21.0 pence sterling per short ton of material dredged (excluding Dredge #12).

ii) The whole operation, including the original sampling, is geared to the recovery of coarse gold, and little fine gold appears to be recovered.

iii) A scheme is proposed to recover some of the gold at present being lost. Whether it is practicable depends very largely on how much fine gold is present in the current tailings from the dredges and sluicing operation.

iv) It has been suggested that an automatic pulp sampler be installed to sample part of the sluice tailings from Dredge #11, and that samples so obtained in the current season be forwarded to a public analyst for assay. The sampler has been ordered and should be delivered about the middle of July, 1957.

v) A further sampling programme is suggested for 1958 if the results of (iv) are favourable, together with the appointment of a resident metallurgist, and the installation of sample preparation and assaying facilities.

vi) A sampling programme is recommended on the amalgamation tailings in the Gold Room to determine whether other valuable minerals, such as platinum, cassiterite, wolframite, gold in pyrite, and rutile, are being concentrated on the dredges.

vii) Consideration should be given to the mechanization of the Gold Room following the results of (vi).

viii) Immediate modifications to the gold saving equipment are suggested for Dredge #12, using two of the existing jigs as screens and recovering gold from the fine material in a new sluice.

As a result of (iv) above, a Gear-Jennings automatic sampler was installed, later than anticipated, in September 1957, at the end of #1 starboard long sluice on Dredge #11. Screens of 1/8" x 1/2" aperture were placed at the end of each of the first three short sluices, feeding #1 long sluice, to divert plus 1/8 inch material from #1 long sluice into a disused long sluice. The minus 1/8 inch material, approximately 80 percent of which passed through the screens, proceeded down #1 long sluice and was sampled every 10 minutes by the sampler. This material was continually sampled between October 2nd and 19th, 1957, and five bulk samples were prepared over the intervals between gold cleanups on the dredge. These five samples were dried, thoroughly mixed, cut down to 50 lb. lots by coning and quartering, and sent to G.S. Eldridge and Co., Vancouver, for assay.

The following results were obtained from the tests:

Test	Tonnage Sampled	Assays Au (Oz./ton)	Assays % S
1	1276	0.0025	0.15
2	2310	0.001	0.08
3	1857	0.0025	0.12
4	1856	0.005	0.17
5	1776	0.003	0.11
TOTAL	9074	0.0028 =9.8¢ per ton	0.12 = 0.22% pyrite

There was insufficient time available at the end of the operating season to obtain samples from the other two long sluices on the starboard side of the dredge.

The results were of considerable interest for two reasons. Firstly, the 9.8 cent per ton assay of the minus 1/8 inch material, which constitutes a fair proportion of the total sands washed over the sluices, was obtained while the dredge was digging through hydraulic tailings from Last Chance Creek and was in poor pay. It was thought that the assay of tailings might increase with return to richer ground. This alone suggested that further work in 1958 was warranted. The pyrite content of the sands sampled was less than might have been expected from a visual inspection of the dredge, so that, unless the pyrite was particularly righ in gold, it would not be of major importance in gold recovery.

Samples of amalgamation tailings from the Gold Room were sent for assay. These heavy sands concentrate with the gold in the riffles on the dredges, and consist largely of magnetite but were a possible source of other commercial heavy minerals. These sands are brought to the Gold Room in the gold concentrate and are scavenged for gold by amalgamation before being discarded. The assay results for the heavy sands from the various operations are set out below:

Dredge	Test Sample	Gold Oz./ton	Platinum Oz./ton	Tin %	Tungsten % WO_3	Sulphur	Radiometric U_3O_8
4	S 10	0.61	Trace	0.06	0.01	13.25	
	S 16	0.22	"	0.05	0.01	17.25	
6	S 13	0.27	"	0.11	0.03	5.31	
	S 18	0.20	"	0.11	0.12	6.28	
8	S 8	0.48	"	0.12	0.01	4.80	
	S 14	0.37	"	0.10	0.01	3.87	
9	S 9	0.68	"	0.10	0.01	19.37	
	S 15	0.40	"	0.07	0.01	23.20	
10	S 12	0.31	"	0.11	0.04	1.62	
	S 19	0.12	"	0.06	0.10	1.72	
11	S 7	0.29	"	0.10	0.01	7.36	
	S 17	0.23	"	0.04	0.01	6.75	
12	S 20	0.12	"	0.20	0.10	2.74	
Par.Hill	S 22	0.19	"	Trace	Trace	1.58	
	S 7 - 17 Composite						Trace
	S 18 - 22						Not detected

% Gold Recovered by Amalgamation	
S 18	69.8
S 19	59.0
S 20	25.0
S 22	47.1

The assay results from these amalgamation tailings showed that platinum, tin, tungsten, and radioactive minerals were present in negligible proportions when the high degree of concentration, to which these minerals have already been subjected, is considered together with the actual small tonnage of these sands brought to the Gold Room each year. However, the gold and sulphur content of the samples was considerable. Assuming that all the gold and sulphur in the samples was in the pyrite (i.e. that there was no free gold in the amalgamation tailings), Nixon calculated pyrite assays of up to several ounces of gold per ton for the various dredges, even allowing for the fact that up to 60 percent of the gold in samples S 18 - 22 was shown to be amalgamable (free).

As against a calculated figure for pyrite on Dredge #4 of 1.66 ounce of gold per ton, two assays of hand-picked pyrite from #4 had shown 0.02 ounce per ton (October 1956) and 0.04 ounce per ton (March 1958). These latter figures would suggest that in fact the amalgamation tailings do contain free gold which has not amalgamated. When one considers that feed to the amalgam barrel contains of the order of 400 ounces of gold per ton, a tailings assay of 0.5 ounce per ton is far less than might be expected from the operation and could reasonably be all free gold which does not readily amalgamate even under the rigorous conditions pertaining in the Gold Room. If this is the case, such difficultly amalgamable gold would not be prone to amalgamate in a half hour test as run by Eldridge, so that a 50 percent recovery of amalgamable gold from the S samples was very probably far below the free gold ratio of the sands.

In view of the conflicting evidence on the importance of the pyritic gold content of the dredge tailings, and also of the gold values found in the tailings from #11 Dredge, it was decided to proceed with the sampling programme during 1958.

Concerning Nixon's other recommendations, no change in Gold Room practice has been implemented, partly because none of the minor minerals tested for were of commercial importance to require special equipment for separation, but largely because of considerations of manpower. One of the most cogent reasons for mechanization of the Gold Room was a reduction in the labour force both for a reduction in labour costs and an improvement in the security risk. However, with the number of dredges and benches at present being operated, the present 2 two-man crews, which are the normal complement together with the Gold Room superintendent, are necessary to clean up the gold at these operations. Thus, without the reduction in labour costs, and consequently with a force large enough to cope with the present hand concentration methods in the Gold Room, there is no gain to be had in capital expenditure on mechanical equipment, especially when the envisaged life of the Company is limited.

Nixon's suggested alterations to the gold saving equipment of Dredge #12 by using two of the existing jigs as screens and new sluices was not implemented, because of the necessity of elevating the jig underflow by means of a pump to the new sluice. In view of the very abrasive character of the sands in the #12 area, it was decided that the wear on the pump would be excessive and costly. Thus the jigs were completely discarded in favour of a system of sluices and riffles similar to the other dredges. This arrangement is now working satisfactorily.

OUTLINE OF TEST PROGRAMME - 1958

The testing programme for 1958 consisted of two main sections:

i) A detailed investigation of the sands from #11 Dredge;

ii) A qualitative appraisal of the performance of the other dredges.

i) The testing programme initially concentrated on Dredge #11, since it is one of the richest producers of gold and the Hunker gravels are the richest in pyrite. Little head room is available beneath the tail sluices, so it was not considered practicable to place an automatic sampler to collect sands from the tail sluices. However, there was sufficient room inside the dredge to sample the sands at the end of each individual long sluice. The automatic sampler was already fitted at the end of #1 long sluice on the starboard side from the 1957 programme, and provision had been made to fit the sampler to #2 and #3 long sluices on the same side.

It was considered reasonable to assume that the port side sluices would perform similarly to those on the starboard side and that the grade and characteristics of the gravels were similar on both sides of the dredge although it was known that the tonnages were not equal. Thus, as there was only one automatic sampler, no attempt was made to sample the port side, except by hand sampling.

The plan evolved for testing Dredge #11 was to sample #1 starboard long sluice for a complete cut across the valley, and #2 and 3 starboard long sluices on the return cut for shorter periods. From this data, an average gold content and tonnage of sands could be calculated as representing the average performance of the dredge in Hunker valley.

Before the dredge turned, however, the Company decided that the dredge should proceed into Last Chance Creek, which it was approaching, as far as it could possibly dredge before returning to the flat of Hunker Creek. Thus there was no possibility of sampling long sluices #2 and 3 while the dredge was treating Hunker gravels. Nevertheless, these two sluices were sampled for the period of one week each, while treating the combination of hydraulic tailings overlaying natural gravels in Last Chance Creek. Using the accurate results from #1 long sluice as a basis, these results for the shorter testing periods on #2 and 3 long sluices were useful for comparative purposes, and calculation of an approximate tail sluice sand. Thus conditions approximating those pertaining in the Hunker Valley was determined, for although large quantities of virtually worthless hydraulic tailings overlaid the natural gravel, this natural material was very similar to that in Hunker Creek.

The sand samples to be taken from the Dredge were to be sent to G.S. Eldridge & Co. for assay for gold, by fire and by amalgamation, and for sulphur. The object was to determine the total gold in the samples by fire assaying, the amount of free gold by amalgamation assay, the amount of gold associated with the pyrite by difference, and the pyrite content from the sulphur assay.

ii) An extension of the testing programme to the other dredges was made possible by the availability of a three-man team of vacation students. While it was not possible to obtain detailed information from these dredges, as in the case of Dredge #11, it was possible to conduct a hand sampling campaign on these dredges for one week each during all three shifts. Thus qualitative information on the performance of the dredges and the values in the dredge tailings was made available, and some indication of the potential of the combined company operations for retreatment of tailings was obtained. Dredge #12 could not be sampled, as access to one of the tail sluices was through the gold saving equipment which must be kept locked at all times.

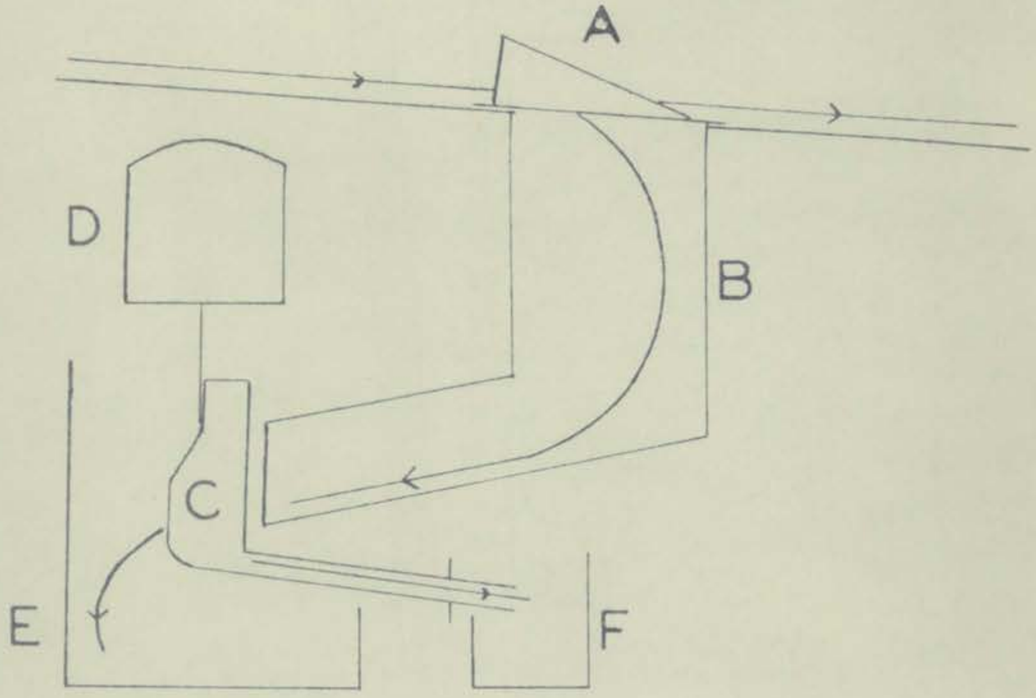

FIG. 1 AUTOMATIC
SAMPLING SYSTEM

PROCEDURE

i) Hand Sampling.

Samples were cut from the tail sluice material with a specially constructed scoop on a long handle. Using walkways at the end of the tail sluices, the sample crews cut fractions from the overflow, by cutting into and through the stream with the scoop and withdrawing it before it filled completely. In practice, there was always some spillage, due to the high velocity of the tail sluice overflow, from water and sands splashing back out of the bucket. This spillage would cause an excess of coarse material in the sample bucket for the fines would splash out again in preference to the coarse sands. However, the above procedure minimized the spillage and the consequent sampling error.

The samples were taken from each tail sluice of the dredges every 10 or 15 minutes. Which of the two intervals was used, was determined largely by the quantity of sands that would be obtained during the sample period. Four positions across the stream were used for taking successive samples, since it was not possible to take a single sample cut across the whole stream. Thus every 40 or 60 minutes a cycle of samples was taken across the tail sluice streams.

Port and starboard samples were kept separate in barrels. The excess water was carefully decanted, after the sands had settled, as no filter was available except on Dredge #11. The samples were then bagged and sent to Bear Creek.

ii) Automatic Samples.

During the 1957 sampling programme, the automatic sampler had been set up to cut from the minus 1/8 inch material in #1 starboard long sluice on Dredge #11. However, for the 1958 programme, it was decided to sample all the sands which normally are treated on the long sluices, rather than lose an indefinite quantity of minus 1/8 inch sands in the oversize from a screen. This increase in volume overloaded the existing sampling system, however, so splitters were constructed to reduce the volume of water and sands passing to the sampler.

The sampling system is shown schematically in Figure 1. The stream reaching the end of the long sluice was split by the riffle A, causing part to continue on down the tail sluice. The portion split out of the stream passed down the chute B and, on emerging from it, was sampled by the cutter C attached to the automatic mechanism D. The sands were then discarded into the pond via the sluice system E, while the sample cut from the stream was collected in the launder F.

On #1 long sluice the maximum size of sands was 1/2 inch; a $1\frac{1}{2}$ inch cutter opening was employed on the sampler and the splitter over the chute sent 50 percent of the stream to the sampler. On the #8 and 3 long sluices where the 5/8 inch maximum size necessitated the use of a $2\frac{1}{2}$ inch cutter, the splitter was constructed to divert 31.25% of the stream to the sampler, so that the total quantity of sample obtained should be approximately constant.

The two splitters used had only four openings across the stream since the openings had to be large enough to avoid bridging by stones. The application of a splitter in this position introduced an error into the samples, since the distribution of sands and values across the stream was very probably asymmetric, so that any sample split from the long sluice stream would not be an exact replica of the stream itself. However, provided that this asymmetry was not too great, no serious error to the accuracy of this investigation would be introduced.

The time interval set for the automatic samples was 30 minutes in all cases. This interval should have been shorter from the point of view of sample accuracy when one considers the cycle of operations on the dredge, but the quantity of sands involved would have been too large to handle. Nevertheless, the effect of any variations in the operating cycle on the samples would even out over the 3 and 4 day sample periods.

From the constants pertaining to the sampling system, the total weight of sands on the sluices being sampled may be calculated from the weight of samples collected, according to the formula:

$$\text{Weight of material obtained per cut (lb)} = \text{lbs. per sec in stream} \times \frac{\text{cutter opening in inches}}{\text{cutter speed (inch per sec.)}}$$

Whence

$$\text{lb. per sec in stream} = \frac{\text{Total weight of sample}}{\text{no. of cuts}} \times \frac{11.4}{\text{cutter opening (in.)}}$$

Whence

$$\text{Total weight of sands sampled in sample period (tons)} = \frac{\text{Total weight of sample (lb)}}{\text{no. of cuts}} \times \frac{\text{no. of hrs. of sample period}}{\text{cutter opening (in.)}} \times 20.52$$

Whence for 50% of stream to sampler and 1½ inch cutter opening

$$\text{Tons of material sampled} = \frac{\text{Total weight of sample (lb)}}{\text{no. of cuts}} \times \text{no. of hrs.} \times 27.35$$

and for 31.25% of stream to sampler and 2½ inch cutter opening

$$\text{Tons of material sampled} = \frac{\text{Total weight of sample (lb)}}{\text{no. of cuts}} \times \text{no. of hrs.} \times 26.27$$

The automatic samples were filtered daily on the pan filter located on the dredge and bulked over the 3 and 4 day periods between gold cleanups on the dredge.

iii) Preparation of Assay Samples.

All samples were dried on a large hot plate at Bear Creek. The aggregated sands were broken up on a 1/8 inch screen and then carefully screened on 35 mesh. The object of the 35 mesh screening was initially an endeavour to eliminate coarse barren sands to simplify the sampling; however, both fractions were sent for assay and in all cases the plus 35 mesh fraction had a small gold content. In addition, the result of screening each sample produced an indication of the size distribution of the sample and also what fraction of sands could be rejected by screening at this size in any subsequent retreatment of tailings. Since 35 mesh is almost the practical limit of commercial screening operations, and it was expected that the gold being sought would be substantially finer than 35 mesh, it was a practical size at which to work.

The separate fractions were thoroughly mixed by coning 16 times and then divided by quartering. After the initial coning 16 times, all subsequent quarterings were preceded by 8 conings. The larger fractions had to be coned and quartered 2, 3 or 4 times to produce a sample of 30 to 50 lbs. to be sent for assay. The remainder of the fractions were labelled and kept for reference.

RESULTS OF INVESTIGATION

DREDGE # 11

Sampling commenced on Dredge # 11 on May 29th, 1958, after a general cleanup. The dredge was on the right limit of Hunker Creek, and after the general cleanup entered lay ground on May 29th to commence a new cut across the valley. Samples # 1 to 13 were taken of the sands at the end of # 1 starboard long sluice over the period May 29th to July 12th. Samples # 14 and 15 were taken at the end of # 2 long sluice between August 18th and 23rd and samples # 16 and 17 from the end of # 3 long sluice between August 23rd and 29th. Details of these tests and the assays of the test products are set out below:

Table 1. Automatic Samples from Dredge # 11

Sample	Period From	To	Weight of Sample (lb)	Tonnage Sampled (Short)	Screen Fractions Mesh Size	% Dist'b'n	Assays Au. oz/ton	% G
1	2:30 PM 5/29/58	8:10 AM 5/31/58	150.5	2067	Whole Sample		0.001	0.24
2	8:45 AM 5/31/58	8:20 AM 6/4/58	325	4450	Whole Sample		0.0005	0.18
3	9:15 AM 6/4/58	8:15 AM 6/7/58	247.5	3386	+35	43.5	0.0010	0.24
					-35	56.5	0.0004	0.13
					Total	100.0	0.00066	0.18
4	10:05 AM 6/7/58	8:20 AM 6/11/58	309	4236	+35	54.2	0.0004	0.11
					-35	45.8	0.0004	0.15
					Total	100.0	0.0004	0.13
5	10:00 AM 6/11/58	8:20 AM 6/14/58	215.5	2976	+35	53.2	0.0004	0.09
					-35	46.8	0.0010	0.09
					Total	100.0	0.00068	0.09
6	9:40 AM 6/14/58	8:30 AM 6/18/58	256	3514	+35	53.2	0.0010	0.11
					-35	46.8	0.0010	0.10
					Total	100.0	0.0010	0.11
7	12:05 PM 6/18/58	8:25 AM 6/21/58	194.5	2574	+35	51.5	0.0003	0.10
					-35	48.5	0.0014	0.07
					Total	100.0	0.00083	0.09
8	9:00 AM 6/21/58	8:20 AM 6/25/58	349	4991	+35	48.3	0.0002	0.08
					-35	51.7	0.0014	0.07
					Total	100.00	0.00082	0.07

Table 1 (Cont'd)

Sample	Period From	Period To	Weight of Sample (lb)	Tonnage Sampled (Short)	Screen Fractions Mesh Size	Screen Fractions % Dist'b'n.	Assays Au. oz/ton	Assays %S
9	11:45 AM 6/25/58	8:25 AM 6/28/58	276	4181	+35	49.5	0.0002	0.10
					-35	50.5	0.0014	0.07
					Total	100.0	0.00081	0.08
10	9:50 AM 6/28/58	8:20 AM 7/2/58	479	6553	+35	51.3	0.0001	0.10
					-35	48.7	0.0006	0.09
					Total	100.0	0.00034	0.10
11	12:15 PM 7/2/58	8:15 AM 7/5/58	357	4884	+35	48.6	0.0002	0.09
					-35	51.4	0.0013	0.12
					Total	100.0	0.00076	0.11
12	9:05 AM 7/5/58	8:20 AM 7/9/58	410	5624	+35	52.5	0.0002	0.08
					-35	47.5	0.0006	0.11
					Total	100.0	0.00039	0.09
13	9:05 AM 7/9/58	8:15 AM 7/12/58	326	4459	+35	49.7	0.0002	0.10
					-35	50.3	0.0005	0.12
					Total	100.0	0.00035	0.11
1 - 13				53,997			0.00061	0.115
14	8:45 AM 8/16/58	7:40 AM 8/18/58	134.5	1770	+35	52.4	0.001	0.05
					-35	47.6	0.002	0.08
					Total	100.0	0.00148	0.06
15	9:35 AM 8/18/58	8:25 AM 8/23/58	336.5	4432	+35	54.3	0.001	0.09
					-35	45.7	0.002	0.08
					Total	100.0	0.00146	0.09
14 - 15				6202			0.00146	0.09
16	10:00 AM 8/23/58	8:30 AM 8/27/58	151	1983	+35	67.3	0.001	0.04
					-35	32.7	0.006	0.09
					Total	100.0	0.0026	0.06
17	10:25 AM 8/27/58	8:15 AM 8/29/58	56	741	+35	75.0	0.003	0.04
					-35	25.0	0.005	0.13
					Total	100.0	0.0035	0.06
16 - 17				2724			0.00287	0.06

Amalgamation assays were performed on certain of the above samples to determine the proportion of free gold in them.

Table 8. Amalgamation Assays on Selected Samples

Sample		Gold Content - oz./ton Fire Assay	Gold Content - oz./ton Amalgamation Assay	% Amalgamated
1	Whole Sample	0.001	0.00021 *	21
2	Whole Sample	0.0005	0.00022 *	44
	-20 +35 mesh	0.0005	0.00028 *	56
	-35 mesh	0.001	0.00028 *	28
3	-35 mesh	0.0004	0.00034 *	85
4	-35 mesh	0.0004	0.00027 *	67.5
5	-35 mesh	0.0010	0.00078 (1) 0.00026 (2) •	100
6	-35 mesh	0.0010	0.00078 (1) 0.00026 (2) •	100

* 1/2 hour amalgamation using lye addition

• (1) 1/2 hour amalgamation - no lye
(2) 1/2 hour reamalgamation of tailing from (1) using lye.

e Note: Larger result from amalgamation assay than from fire assay due to unavoidable sampling errors.

During the sample periods # 1 to 13, when the sands reaching the end of #1 long sluice were being sampled, a detailed account of gold recovery on the test sluices was kept, i.e. gold recovered on starboard short sluices # 1, 2 and 3 and long sluice # 1. The gold cleanup on these sluices was carried out at the normal cleanup times for the dredge but was performed completely separately and in the same manner as a normal cleanup so that no artificial conditions were imposed on the gold recovery on these sluices. On an ordinary cleanup the coco-mats at the head of the test sluices were taken up every time together with the sands from those adjacent riffles which were overloaded with gold. This gold was cleaned up in the usual way, weighed after the usual Gold Room cleaning procedure and recorded against the preceding sample period. When a short sluice cleanup was scheduled for the dredge, the above procedure was carried out for the mats and overloaded riffles, and then the remaining riffles on the short sluices were cleaned up and the gold thus recorded was apportioned among those test periods since the preceding short sluice cleanup, according to the tonnage of sands treated in those periods. The long sluice was cleaned up only once, at the end of the 13th period and the gold recovered apportioned in the same way as for the short sluices, by tonnage.

Overpannings from the Gold Room cleaning of the test gold were amalgamated separately and the gold apportioned among the test periods according to the amount of gold already recorded for them. The weights of gold recovered and the calculated recovery for each test period are presented below:

Table 3. Details of Gold Recovered on Test Sluices

Sample Period	Mats & Sands	Short Sluices	Long Sluice	Overpannings	Actual Gold Recovered within Sample Period
1	63				71
2	155				173
3	102				115
4	115				131
5	122				135
6	117	43			131
7	90				97
8	107				118
9	70				78
10	179	16			195
11	58				65
12	64				71
13	146	8	11	71	157
	1,388	67	11	71	1,537

The gold recovered from the mats and short sluices between May 29th and June 21st was sized on standard Tyler screens, as was that recovered from the long sluice. These analyses do not include overpannings, but these latter were so small a proportion of the total that they would only slightly affect the sizing analysis.

Table 4. Sizing Analyses of Gold Recovered from Test Sluices

Mesh Tyler		Short Sluices # 1, 2 and 3	Long Sluice # 1
	+7	0.3	0.4
-7	+10	1.3	0.3
-10	+20	33.6	29.0
-20	+40	38.4	42.5
-40	+60	23.0	23.1
-60	+100	3.3	4.4
-100	+200	0.1	0.2
-200		-	-

Sizing and chemical analyses of the test sample # 2 were carried out down to 35 mesh, while the minus 35 mesh fractions of test samples # 3 and 4 were sized down to 200 mesh Tyler and analysed:

Table 5. Sizing and Chemical Analyses of Test Products

Sample	Mesh Tyler	% Dist'b'n	oz. Au/ton	% S
2	+ 6	8.0	0.0005	0.19
	-6 + 10	11.8	0.0005	0.20
	-10 + 20	10.9	0.0005	0.22
	-20 + 35	12.2	0.0005	0.20
	-35	57.1	0.001	0.13
	Total	100.0	0.0008	0.16
			0.0005 (by assay)	0.18 (by assay)
3 (-35 mesh)	+ 60	22.7	0.0004	
	-60 + 100	18.1	0.0004	
	-100 + 200	11.9	0.0012	
	-200	47.3	0.0010	
	Total	100.0	0.0008	
			0.0004 (by assay)	
4 (-35 mesh)	+ 60	32.2	0.0004	
	-60 + 100	19.5	0.0005	
	-100 + 200	14.5	0.0006	
	-200	33.8	0.0006	
	Total	100.0	0.0005	
			0.0004 (by assay)	

During certain of the test periods on Dredge # 11 (#5, 11, 12, 15, 16, and 17) hand samples were taken from the tail sluices for comparison with the automatic samples. During periods # 15 to 17, the sand elevators were by-passing some of the coarse sands from the long sluices to the stacker, so the elevator sands were also sampled to determine whether any gold was settling out of the tail sluice stream into the elevators. Details of these samples are set out below:

Table 6. Details of Tail Sluice Samples on Dredge # 11.

Sample	Weight of Sample (lb)	Screen Fractions Mesh Size	Screen Fractions % Dist'b'n	Assays oz Au/ton	Assays % S
5 Port Sluice	36	+ 35	74.1	0.0005	0.09
		-35	25.9	0.0840	0.11
		Total	100.0	0.0221	0.10
5 Stbd Sluice	59.5	+ 35	63.9	0.0750	0.10
		-35	36.1	0.0430	0.13
		Total	100.0	0.0634	0.11
11-12 Port Sluice	248.5	+ 35	59.8	0.0002	0.12
		-35	40.2	0.0008	0.11
		Total	100.0	0.00044	0.12
11-12 Stbd Sluice	365	+ 35	56.8	0.0005	0.15
		-35	43.2	0.0010	0.12
		Total	100.00	0.00072	0.14
15 Port Sluice	205	+35	57.9	0.0001	0.02
		-35	42.1	0.0005	0.06
		Total	100.0	0.00027	0.04
15 Port Elevator	576.5	+35	92.2	0.0002	0.02
		-35	7.8	0.0140	0.05
		Total	100.0	0.00026	0.02
15 Stbd Sluice	311	+35	54.7	0.0002	0.03
		-35	45.3	0.0020	0.05
		Total	100.0	0.0010	0.04
15 Stbd Elevator	498	+35	92.6	0.0002	0.02
		-35	7.4	0.0010	0.06
		Total	100.0	0.00026	0.02
16 Port Sluice	97	+ 35	50.0	0.0003	0.02
		- 35	50.0	0.0005	0.05
		Total	100.0	0.0004	0.035
16 Port Elevator	464	+35	92.9	0.0003	0.02
		-35	7.1	0.0012	0.05
		Total	100.0	0.00036	0.02
16 Stbd Sluice	251.5	+35	55.5	0.0003	0.02
		-35	44.5	0.0005	0.05
		Total	100.0	0.00039	0.03
16 Stbd Elevator	481.5	+35	93.5	0.0003	0.02
		-35	6.5	0.0013	0.05
		Total	100.0	0.00036	0.02

Table 6 (Cont'd)

Sample	Weight of Sample (lb.)	Screen Fractions Mesh Size	% Dist'b'n	Assays oz Au/ton	% S
17 Port Sluice	42	+35	47.6	0.0003	0.03
		-35	52.4	0.0170/	0.05
		Total	100.0	0.00040	
17 Port Elevator	133	+35	90.6	0.0003	0.02
		-35	9.4	0.0020	0.03
		Total	100.0	0.00046	0.02
17 Stbd Sluice	97	+35	44.9	0.0003	0.02
		-35	55.1	0.0005	0.05
		Total	100.0	0.00041	0.04
17 Stbd Elevator	175	+35	91.4	0.0003	0.02
		-35	8.6	0.0013	0.05
		Total	100.0	0.00038	0.02

// Obviously in error; 0.0013 taken
/ Obviously in error: 0.0005 taken

The gold recovered during the sample periods in Table 5 was cleaned up separately on each side of the dredge and recorded:

Table 7. Distribution of Gold Recovery during Hand Sampling on Dredge # 11.

Sample Period	Gold (oz) Port	Starboard
5	75	230
11 - 12	65	197
15	125	375
16	84	155
17	32	90

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DREDGES # 4, 6, 8, 9, 10

The sands from both port and starboard tail sluices on these dredges were sampled by hand on three shifts for one week each. Two separate weeks' samples were taken on # 4 dredge.

Table 8. Details of Hand Samples on Dredges # 4, 6, 8, 9, 10.

Dredge No.	Sample	Period From	Period To	Weight of Sample (lb)	Screen Mesh Size	Fractions % Dist'b'n.	Assays Oz Au/ton	Assays % S
4	1 Port	6/25/58	7/2/58	88	+35	54.0	0.0003	0.01
					-35	46.0	0.0007	0.03
					Total	100.0	0.00048	0.02
	1 Stbd	6/25/58	7/2/58	86	+35	59.6	0.0003	0.01
					-35	40.4	0.0007	0.03
					Total	100.0	0.00046	0.02
	2 Port	7/ 9/58	7/16/58	208	+35	66.7	0.0002	0.02
					-35	33.3	0.0006	0.03
					Total	100.0	0.00033	0.02
	2 Stbd	7/ 9/58	7/16/58	158.5	+35	70.7	0.001	0.03
					-35	29.3	0.002	0.04
					Total	100.0	0.0013	0.03
6	Port	7/17/58	7/24/58	495.5	+35	77.8	0.0002	0.02
					-35	22.2	0.0020	0.09
					Total	100.0	0.00060	0.04
	Stbd	7/17/58	7/24/58	565.5	+35	76.2	0.0003	0.04
					-35	23.8	0.0020	0.08
					Total	100.0	0.00071	0.05
8	Port	7/25/58	8/1/58	335.5	+35	68.8	0.0003	0.03
					-35	32.2	0.0015	0.06
					Total	100.0	0.00069	0.04
	Stbd	7/25/58	8/1/58	479.5	+35	71.0	0.0003	0.03
					-35	29.0	0.0015	0.05
					Total	100.0	0.00064	0.04
9	Port	8/1/58	8/8/58	413	+35	42.7	0.0015	0.05
					-35	57.3	0.0010	0.07
					Total	100.0	0.0011	0.06
	Stbd	8/1/58	8/8/58	343.5	+35	51.4	0.0002	0.05
					-35	48.6	0.0013	0.08
					Total	100.0	0.00073	0.06
10	Port	8/11/58	8/18/58	260.5	+35	30.3	0.001	0.09
					-35	69.7	0.002	0.08
					Total	100.0	0.0017	0.08
	Stbd	8/11/58	8/18/58	324.5	+35	44.9	0.001	0.10
					-35	55.1	0.002	0.07
					Total	100.0	0.0015	0.08

On each of these dredges the gold, recovered on both port and starboard sluices during the sample period, was cleaned up seperately for comparison with the tailings assay on the respective side.

Table 9. Gold Recovered during Sampling Periods on Dredges # 4,6,8,9,10.

Dredge No.	Gold (oz) Port	Starboard
4 - 1	152	74
4 - 2	44	21
6	123	105
8	154	628
9	192	195
10	113	251

Samples of the pyrite in the various creeks were hand picked where possible from the riffles and cleanup boxes on the dredges. Also at Dredge # 11, cubes of pyrite imbedded in the coarse stacker tailings were obtained. These pyrite samples were hand picked and sent for assay with the following results:

Table 10. Assays of Pyrite Samples.

Dredge No.	Oz. Au/ton	% Fe	% S	Pure Pyrite Oz. Au/ton	¢/ton
4 (Bonanza)	0.02		51.05	0.021	73
6 (Lower Sulphur)	0.02		50.46	0.021	74
11 (Hunker)	0.006	46.41	51.06	0.0062	22
11 (Stacker Tailings)	0.01	46.44	51.08	0.0104	36

DISCUSSION OF RESULTS AND CALCULATIONS

DREDGE NO. 11

(a) Performance of the Riffle System.

Nixon concluded (5) that the dredges were operating efficiently in recovering plus 100 mesh gold from the gravels, but there was little possibility that finer gold could be saved by the riffles under the necessary operating conditions. This has been borne out by the distribution and sizing of the gold recovered on short sluices # 1, 2 and 3, and long sluice # 1 on the starboard side of Dredge # 11 (hereinafter referred to as the "test sluices") during the testing programme.

The sizing of the gold recovered on both the short and long sluices (Table 4) was 99.9 and 99.8 percent, respectively, plus 100 mesh, and with substantially the same distribution in the coarser sizes. It is thus evident that the long sluice does not recover any finer gold than the short sluices.

Of the 1537 ounces of gold recovered on the test sluices during the testing programme (Table 3), 1455 ounces were recovered on the mats and riffles of the short sluices and 11 ounces on the long sluice (the 71 ounces recorded as overpannings may be assumed to have been distributed between the short and long sluices in the same ratio, since the amount of overpannings is roughly proportional to the quantity of gold being cleaned). Thus only 0.75 percent of the gold recovered on the test sluices was from the long sluice. This compares with 1.5 percent of the total gold recovered during the 1957 operating season on the whole dredge sluice system, being from the long sluices. The difference between the two figures may have been due to slightly different operating conditions, or varying properties of the gold and gravel between the two seasons, or that a slightly higher proportion of the gold is recovered on the other long sluices compared with # 1 starboard long sluice.

This distribution of gold between the short and long sluices shows definitely that the riffle system is not overloaded, that is, that ample riffle capacity for additional gold recovery is available on the long sluices. Thus, from the evidence of the sizing of the gold and the extra riffle capacity available, it may be concluded that the riffle system, as operated, of necessity, with a high velocity of water to keep the coarse sands moving, is not capable of recovering any appreciable quantity of minus 100 mesh gold on Dredge # 11.

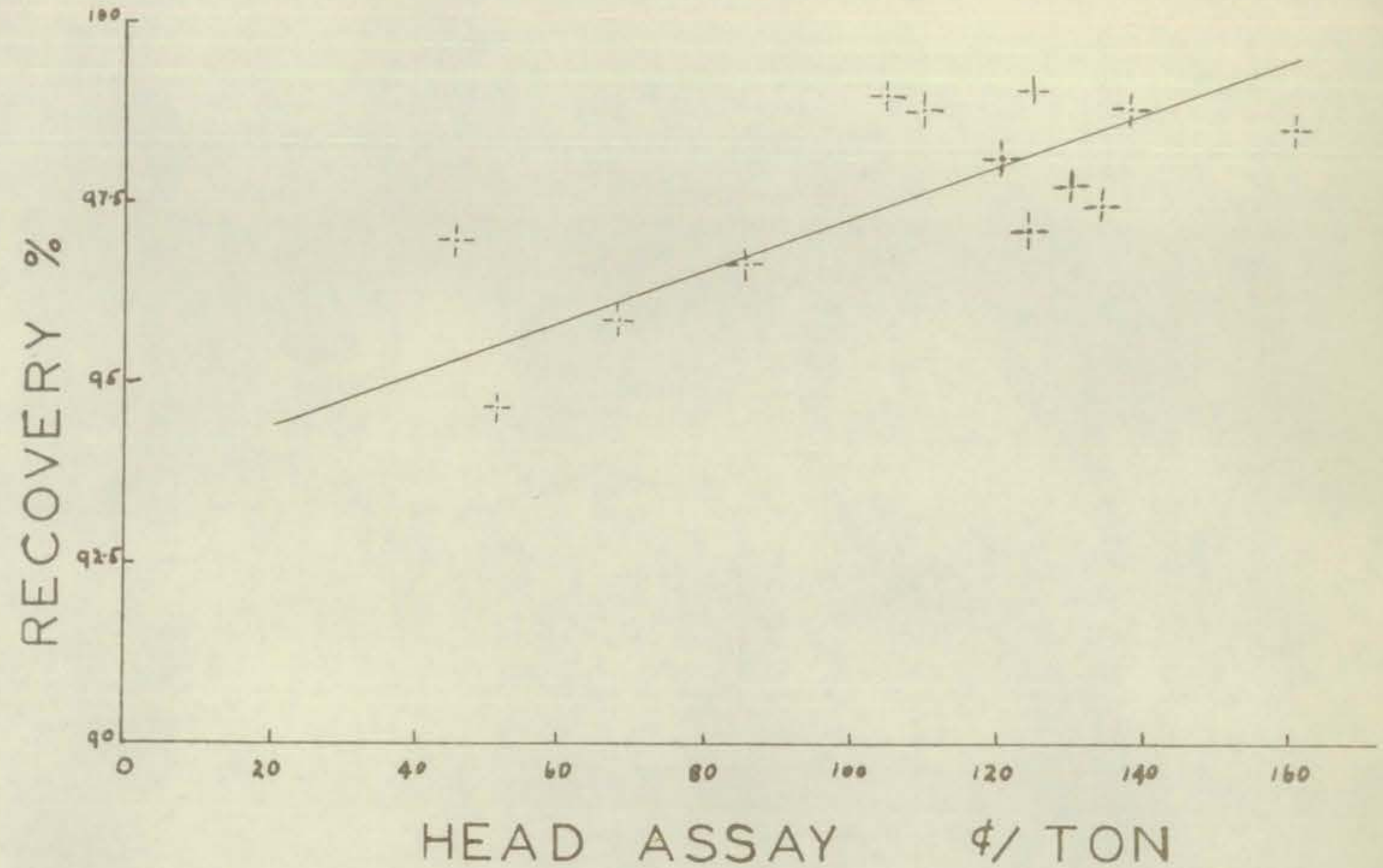
This conclusion may be extended to all the dredges since the same factors apply. The operating conditions on each dredge determine a critical size of gold particle below which the gold is not effectively recovered, since the velocity of water on the sluices prevents this fine gold from settling permanently into the riffles. Thus no additional riffle capacity will extend the size range over which gold can be recovered and hence increase the recovery on the dredges. From the size distribution of the gold recovered on the dredges as reported by Scott in 1941 (1), it is apparent that the critical size is nearer 200 mesh for dredges # 6, 8 and 10.

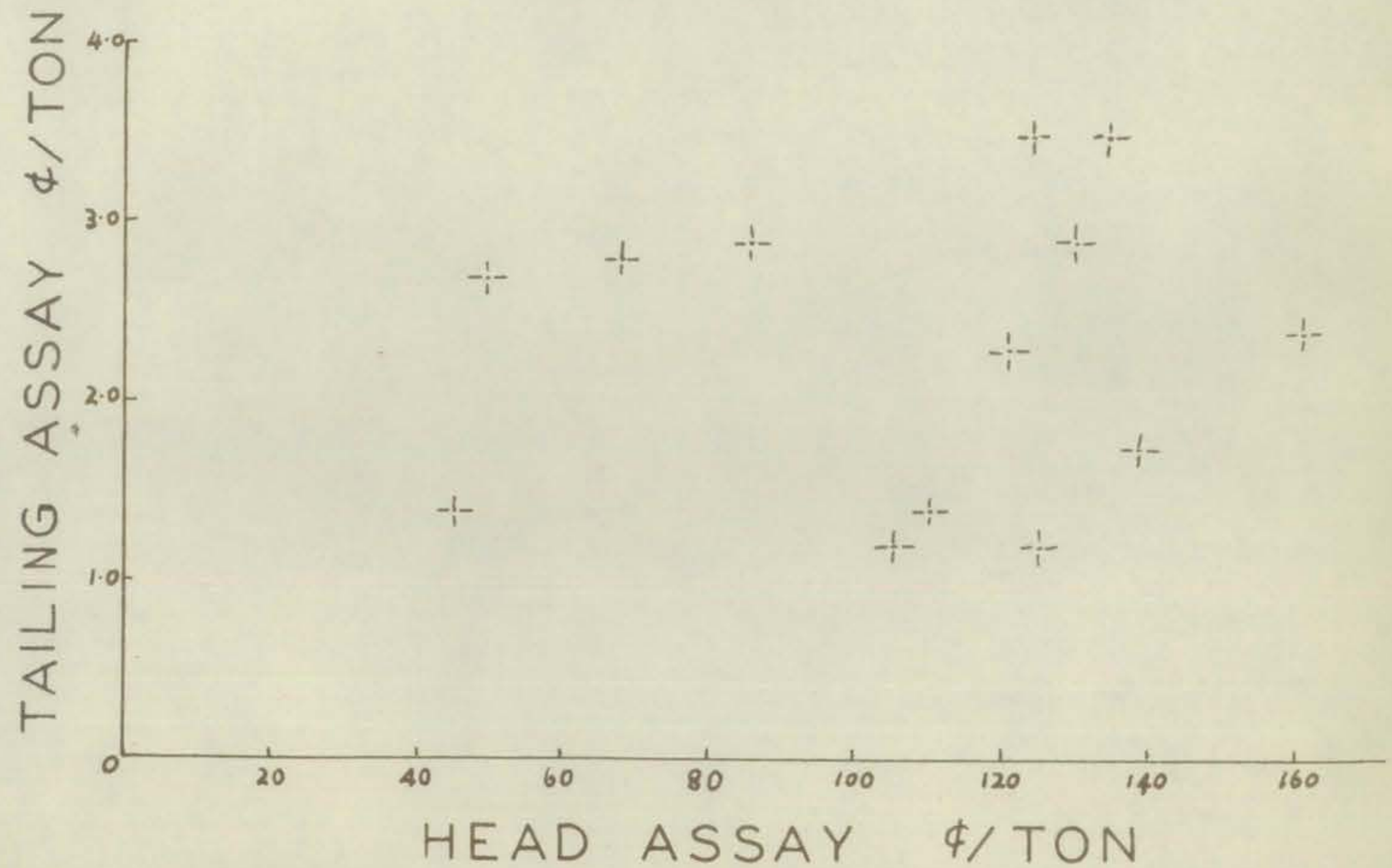
However it cannot be concluded that the sand tailings contain no free gold coarser than the critical size. The results in Table 1 show that the gold content of the tailings is concentrated in the minus 35 mesh fraction, although in certain cases the content of the plus 35 mesh fraction is appreciable. Table 5 shows that, while the gold values tend to concentrate in the minus 35 mesh fraction (Sample 2), and also within the minus 100 mesh fraction of the fine sands (Samples 3 and 4), nevertheless the plus 100 mesh sands do contain a considerable amount of gold.

A part of the gold in both coarse and fine fractions of the sand tailings is locked within pyrite and the other sand minerals, but it is not the larger part. Amalgamation of the 20 to 35 mesh fraction of Sample 2 has shown a high proportion of free gold and the minus 35 mesh fractions of Samples 2 to 6 contained from 28 to 100 percent free gold (Table 2). These percentages are probably lower than the actual proportions of free gold in the samples, although a figure of plus 100 percent amalgamated is obviously a result of sampling errors which are unavoidable when handling such low grade placer gold ores.

This small gold content in the tail sands is thus of the order of 50-75 percent free and a considerable quantity of this free gold is larger than 100 mesh. Since it has been determined above that there is ample capacity for recovery of plus 100 mesh gold, it must be concluded that the presence of such gold in the tailings is due to mechanical carry-over -- either stuck to undisintegrated mud, or scoured out of the riffles by the sands, especially if the riffles have been overloaded during operation.

Notwithstanding that fact that there are losses of both coarse and fine gold in the sand tailings, these losses are actually and relatively small. By transposing the data in Tables 1 and 3, the head assay of the sands entering the test sluices during the testing period and the resultant recovery may be calculated. It must be recognized, however, that the head assays for each test period are only an approximation, since the method of calculating the gold recovered in each test period, as set out above, is not absolute, but as close as could possibly be determined without departing from normal cleanup procedure.


RECOVERY %
100
97.5
95
92.5
90
0
20
40
60
80
100
120
140
160
HEAD ASSAY ¢/ TON


TAILING ASSAY ¢/TON
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3.0
2.0
1.0
0
20
40
60
80
100
120
140
160
HEAD ASSAY ¢/ TON

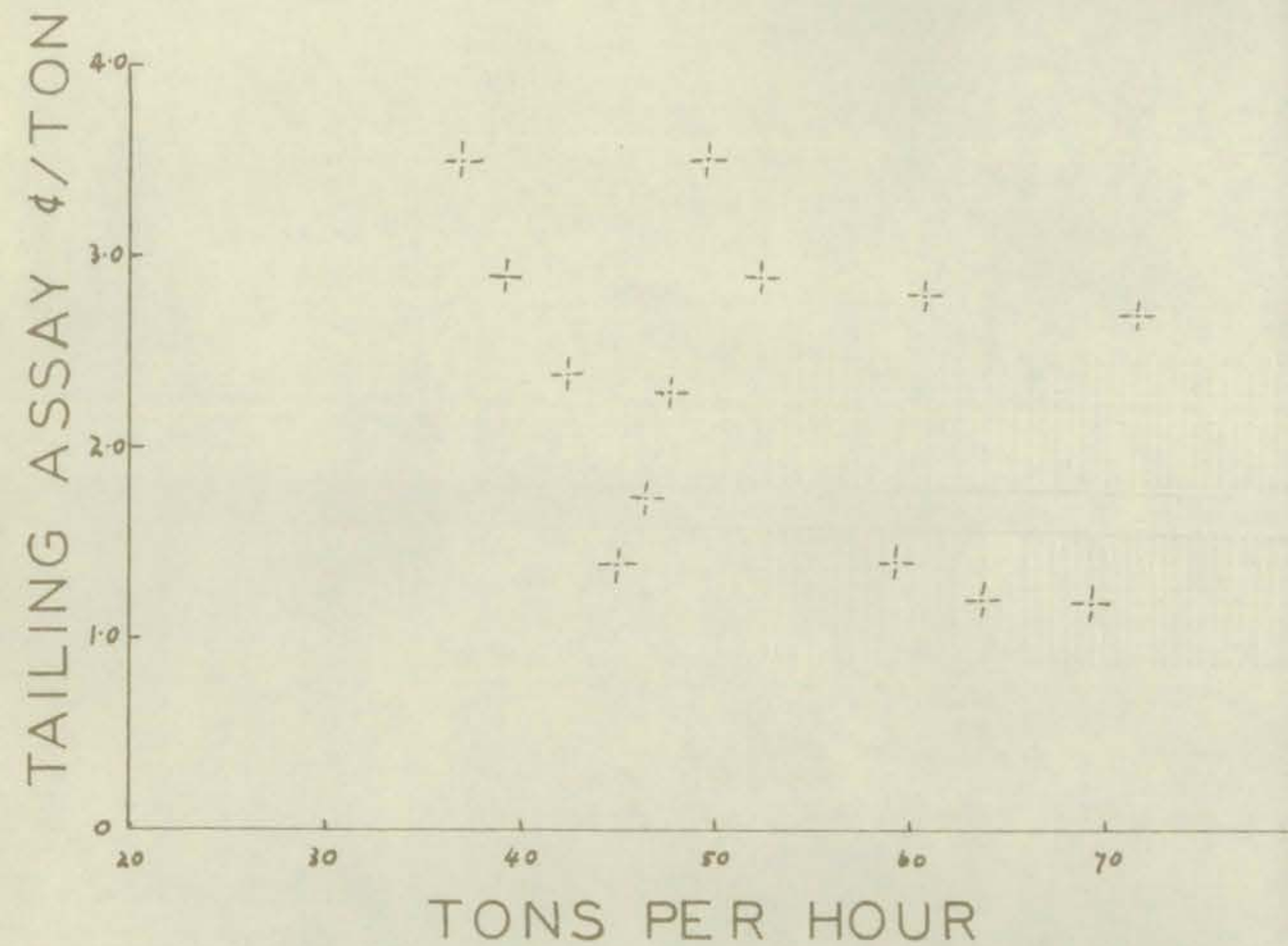
TAILING ASSAY ¢/TON
4.0
3.0
2.0
1.0
0
20
30
40
50
60
70
TONS PER HOUR

FIGURE 2

Table 11. Recoveries of Gold and Head Assays of Sands on Test Sluices.

Sample No.	Tons Sampled	Tons Per Hour	Assays of Tailings Au. oz/ton	¢/ton	S %	Total Gold Content of Tailings (oz.)	Gold Recovered (oz.)	Head Assays Au. oz/ton	¢/ton	% Gold Recovery
1	2067	49.6	0.001	3.5	0.24	2.1	71	0.0354	124	97.2
2	4480	46.5	0.0005	1.75	0.18	2.2	173	0.0394	138	98.8
3	3386	47.7	0.00066	2.3	0.18	2.2	115	0.0346	121	98.2
4	4238	45.0	0.0004	1.4	0.13	1.7	131	0.0313	110	98.8
5	2976	42.4	0.00068	2.4	0.09	2.0	135	0.0460	161	98.6
6	3614	57.0	0.001	3.5	0.11	3.5	131	0.0383	134	97.5
7	2674	39.1	0.00083	2.9	0.09	2.2	97	0.0371	130	97.8
8	4991	52.4	0.00082	2.9	0.07	4.1	118	0.0245	86	96.7
9	4181	60.9	0.00081	2.8	0.08	3.4	78	0.0195	68	95.8
10	6553	69.3	0.00034	1.2	0.10	2.2	195	0.0301	105	99.0
11	4884	71.8	0.00076	2.7	0.11	3.7	65	0.0141	49	94.7
12	5624	59.1	0.00039	1.4	0.09	2.2	71	0.0130	45.5	97.0
13	4459	63.7	0.00035	1.2	0.11	1.6	157	0.0356	125	99.1
1-13	53997	52.5	0.00061	2.1	0.115	33.1	1537	0.0291	102	97.9

Over the thirteen test periods, the recoveries on the test sluices were very high, varying from 94.7 to 99.1 percent, with an average recovery of 97.9 percent. Such a performance is extremely good when the very low grade of gravels is considered. It will be seen, however, that the average value of sands entering the test sluices, 102 cent per ton, was very much higher than the average value for the whole dredge during the tests (of the order of 87 cent per ton). Since the tailings values were virtually independent of head assay (Fig. 2) it might be expected that, from a lower average grade of gravels for the whole dredge, over all recovery would be somewhat lower. This contention is supported by the effect of head assay on recovery for the test sluices (Fig. 2). Although the individual points are somewhat scattered, there does appear to be a roughly linear increase in recovery with the head assay. Such a relationship would be expected since it has been shown above that there is ample capacity for gold recovery on the dredge, above the critical size.

The tonnage rate had no definite effect on the tailings assay (Fig. 2) although there does seem to be a slight tendency for the higher tonnage rates to be associated with lower tailings assays. This effect would be partly explained by the fact that, during the later testing periods on the # 1 long sluice when the larger tonnages were treated, the high bank of virtually barren hydraulic tailings from Last Chance Creek formed part of the throughput. Thus the ratio of barren overburden to gold bearing gravels was higher than usual with a nett decrease in the head assay of the gravel treated, as shown in Table 11, whereas if the dredge were operating close to its maximum recovery capacity with a normal grade of gravel, increasing tonnage would be expected to produce greater losses.

(b) Values in the Tail Sluice Sands.

The average assays and total tonnages of samples from each of the three long sluices are summarized below from Tables 1 and 11:

Table 12. Summary of Sand Distribution and Values on Starboard Sluices.

Long Sluice No.	Tonnage Samples	Cubic Yards Dug	ASSAYS Au. oz./ton	g/ton	% S
1	53,997	216,495	0.00061	2.1	0.115
2	6,202	38,536	0.00146	5.1	0.08
3	2,724	32,625	0.00287	10.0	0.08

The total number of cubic yards of gravel dug during the sample periods on each sluice has been calculated from the monthly and semi-monthly yardage figures as computed in the Engineering Department. An average daily yardage was calculated from the monthly and, when required, from the semi-monthly figures. Totalling the yardage from the calculated daily figures and the monthly totals, over the 6 weeks testing period on # 1 long sluice, has yielded a fairly accurate figure. However, for the shorter 7 and 6 day periods on sluices # 2 and 3, respectively, the yardages were calculated from the actual number of running hours, but are likely to be somewhat inaccurate because of variations in dredging rate throughout the month.

The figure tabulated for # 3 sluice must be considerably lower than was actually dug during the test period, for on August 24th Last Chance Creek was flooding with heavy rain and washed a large quantity of gravel into the dredge pond. This extra gravel was dug by the dredge for approximately 24 hours which is equivalent to about 6000 cubic yards, but since this quantity could not be determined accurately, it has not been included. Thus the tonnage sampled on long sluice # 3 was a smaller fraction of the total yardage dug than is apparent from Table 12.

The much larger gold content of the tailings sands from # 2 and 3 long sluices compared with that in # 1 long sluice was unexpected, since gold recovery on the short sluices feeding them is lower, especially in the case of # 3 sluice. These higher losses were presumably of fine gold, but there seems to be no obvious reason why the finer gold should segregate to the end of the rotary screen unless it is associated with mud which does not readily disintegrate under the action of the water jets within the screen.

One factor which contributed to higher values on long sluice # 2 was the density of dredge pond water. During June and July when plenty of water was running in the creeks, the water in the dredge pond was not excessively muddy, and since water was flowing through the pond the mud content did not increase greatly. However by August, when the samples on # 2 and 3 long sluices were being taken, the dredge was on high ground and moving into Last Chance Creek which could not supply sufficient water. It was therefore necessary to pump water into the dredge pond to maintain its level, with the result that no large quantity of water was passing through the pond and the proportion of mud in the water increased considerably. The higher nett density of the water would hinder the settling of the finer gold, thus increasing the critical size of gold recoverable and the consequent losses of fine gold in the tailings. Such a condition of very high mud content in the pond water has long been known to produce higher tailings losses from the dredges.

Nevertheless, the muddy water cannot have greatly affected the samples from # 3 sluice, since at the beginning of these two sample periods, the dredge pond was flooded with clean water by Last Chance Creek. The mud would largely have been flushed out.

Under conditions of cleaner pond water, the losses on these sluices should be lower.

(i) Calculation of composite dredge tailings.

To calculate an average tail sluice sand for the dredge, from the results obtained for the 3 starboard long sluices, certain assumptions must be made:

(i) The port and starboard sluice systems produced similar tailings;

(ii) The properties of the gravels dug on the Hunker flat and in Last Chance Creek were similar.

(iii) The weight distribution of sands on the short and long sluices was constant and independent of tonnage rate.

The first assumption would appear to be reasonable, since the tail sluice samples for both port and starboard sides had assays of the same general order during test periods 11-12, 15, 16, 17 (Table 6). This principle also held for the other dredges sampled (Table 8).

The second assumption is necessary to make use of the samples obtained in Last Chance Creek for calculating the properties of the gravels on Hunker Flat. Since the percentages of plus and minus 35 mesh material in the sands sampled on # 1 long sluice did not vary greatly from 50 percent, between the right limit of Hunker Creek and the mouth of Last Chance Creek where a high proportion of hydraulic tailings was being dug, it may be assumed as an approximation that the size distribution of the total gravel dug, and hence the proportion passing through the screen, was fairly constant. The overlay of hydraulic tailing in Last Chance Creek contained less very fine sand than the gravelly mud on the natural flats of Hunker Creek, but being also barren, it should have had little effect on the gold recovery, except that it might have tended to scour gold out of the riffles more than the overburden on the flats, and thus increased gold losses. The pyrite content of the samples taken in Last Chance Creek, as seen by the sulphur assays, was considerably lower than in Hunker Creek (Tables 1, 6), but, since the pyrite is of minor importance to the investigation (see below), this difference has little bearing on the second assumption.

The third assumption, concerning the distribution of sands among the short sluices, is definitely incorrect, but necessary. As the tonnage treated increases, a greater proportion of the fine material moves down the rotating screen with the result that a higher proportion of the total sands on the sluices reports on the stern short sluices. Thus the proportion of sands measured on # 2 and 3 long sluices, when the tonnage rate was higher, must have been larger than the average for the cut across the valley. Hence, computing a composite tailing sand, by simple proportion, from weights of samples obtained overweights the contribution of # 2 and 3 long sluices, but no correction factor is available.

Accepting these three assumptions, with the above reservations, an average value of sluice tailings may be calculated from the tonnages sampled on the respective sluices and the number of cubic yards dug during the same periods.

Thus # 1 long sluice treated 53,997 tons of sands while 216,495 cubic yards were dug. Therefore in the same time # 2 sluice treated

$$5202 \times \frac{216,495}{32,636} = 34,843 \text{ tons}$$

and # 3 sluice treated

$$2724 \times \frac{216,495}{32,625} = 18,076 \text{ tons}$$

Thus the composite tailings sands may be calculated:

Table 13. Calculation of Composite Tail Sluice Sand.

		Assays		Units	
Sluice	Tonnage	Oz. Au/ton	% S	Oz. Au.	Tons S
1	53,997	0.00061	0.115	32.9	62.1
2	34,843	0.00146	0.08	50.9	27.8
3	18,076	0.00287	0.06	51.9	10.8
	106,916	0.00128	0.09	135.7	100.7
		= 4.5¢/ton		= 0.17% pyrite	

This figure of 4.5 cent per ton for the average tail sluice sand during the cut across the valley may be considered as an absolute maximum for Dredge # 11, since, as discussed above, the tonnage is probably too highly weighted toward sluices # 2 and 3 and the tailings losses for these sluices were apparently high due to abnormal operating conditions.

Hand samples taken on both tail sluices during test periods 11 and 12 (Table 6) had an average value of 0.00058 ounce per ton (2.0¢) which agreed closely with the average value of 0.00056 ounce per ton (2.0¢) for the automatic samples from # 1 long sluice for these periods. While allowing for the fact that the hand samples may not be particularly accurate, this comparison suggests either that the assays of the # 2 and 3 long sluices were not greatly different from that of # 1 long sluice or, if so, that their proportionate contribution to the tail sluice sands was not large enough to cause any great difference between these assays. The assays of automatic samples 14-15 and 16-17, from # 2 and 3 long sluices respectively, were consistent within themselves, and since there is no reason to doubt the accuracy of the assays, the high values were probably a result of the operating conditions in Last Chance Creek. In addition, the average values for these sluices of 0.00146 ounce per ton (5.1¢) and 0.00287 ounce per ton (10.0¢) were considerably higher than those of the average tail sluice - elevator samples at 0.00063 ounce per ton (2.2¢) - sample 15 only, and 0.00038 ounce per ton (1.3¢), respectively.

Thus it would appear that the proportion of sands shown on # 2 and 3 long sluices in the calculation above is far too high, since the values on these sluices have produced a composite tailing assay considerably different from the value on # 1 long sluice. Therefore, since the actual weights of sands sampled on these sluices is not in doubt, the yardage calculated as being dug during these test periods 14 to 17 must be very much in error and far lower than actual. Part of the error involved must also be attributed to the third assumption made for the calculation, for the variation with tonnage rate in the distribution of the sands between forward and aft short sluices may be quite large.

It is therefore evident, on the basis of these observations, that the calculated figure of 0.00128 ounce per ton (4.5¢) is excessive as an average value of tail sluice sands produced by the dredge in Bunker Valley. A value between 0.0006 and 0.001 ounce per ton is indicated, possibly 0.0009 ounce per ton (3.1¢).

(2) Calculation of dredge recovery.

Using the maximum tailings figure of 4.5 cent/ton, an average percentage recovery of gold by the dredge can be calculated. Total gold recovered by the dredge during the cut across the valley, that is, the first 13 test periods was 4465 ounces, as accurately as it can be determined from the cleanup figures.

There are three indefinite quantities that are pertinent to the calculation of dredge recovery.

(i) The weight of 1 cubic yard of gravel.

(ii) The proportion of gravels discarded by the stacker.

(iii) The relative proportions of sands treated on both port and starboard sluices.

None of these three quantities has been recorded as accurately measured. A figure of 1.5 short tons per cubic yard of gravel dug is generally used, when necessary;

however, figures of up to 1.9 short tons per cubic yard are used elsewhere. The proportion of gravel discarded by the stacker is estimated at an average of 50 percent, which figure, though never measured, seems to be fairly accurate. The relative proportion of sands on port and starboard sluices has likewise never been determined. On the basis of the ratio of gold recovery (Table 7), the proportions might be expected to be 1:3, port to starboard. This from visual inspection seems rather excessive, but a 1:2 ratio would appear to be approximately correct.

Two methods are available for calculating the percentage recovery of gold by the dredge. Firstly, using the calculated tonnage of 106,916 for the starboard side and the 1:2 distribution ratio, the total tonnage of the tail sluice sands for the cut across the valley is calculated as 160,324. Thus the gold content of 160,324 tons of sands containing 0.00128 ounces/ton

$$= 160,324 \times 0.00128 = 205 \text{ ounces}$$

$$\text{Gold recovered} = \frac{4455}{4660} \times 100 = 95.6\ \%$$

The second method gives the same result: 216,495 cubic yard of gravel dug, weighing 1.5 ton per cubic yard, and 50 percent passing to the stacker yields

$$216,495 \times 1.5 \times 0.5 = 162,371 \text{ tons of sands treated on the sluices.}$$

This result is very close to the figure arrived at above, which allowing for the fact that the calculated tonnage on the starboard side was somewhat high, as discussed above, suggests that the factors (1) - (3) taken above must be reasonably correct within limits.

The gold lost in the tailings = 162,371 x 0.00128 = 208 ounces

Gold recovered = 4455 ounces

$$\text{Percentage recovery} = \frac{4455}{4663} \times 100 = 95.5\ \%$$

As against these percentage recoveries, a figure closer to those obtained on the test sluices is obtained by using the value of 0.0009 ounce per ton (3.1¢), considered above to represent more closely the performance of the dredge across Hunker Valley.

Thus gold lost in tailings = 162,371 x 0.0009 = 146 ounces

Gold recovered = 4455 ounces

$$\text{Percentage recovery} = \frac{4455}{4601} \times 100 = 97.0\ \%$$

DREDGES NOS. 4, 6, 8, 9, 10

The hand samples of the tail sluice sands from these dredges show that in general both port and starboard sluice systems produce very similar grades of tailings (Table 8) which do not appear to be influenced by the relative gold recovery between the two systems (Table 9). Thus these assays for each dredge may be averaged to produce mean tailings assays which are tabulated below, together with the gold recovered on the mats of the respective dredges and the yardage dug during each test period.

Table 14. Mean Tailings Assays and Total Gold Recoveries on Dredges # 4,6,8,9,10

Dredge No.	Cu. Yds. Dug	Percent Stacker Tailings	Sand Tailings Assays oz. Au/ton	cent/ton	% S	Gold Recovery oz.	Recovery • Factor
4	144,016	50	0.00040 ≠	1.4	0.02	291	60.95
6	37,359	50	0.00065	2.3	0.04	228	73.88
8	34,391	45	0.00066	2.3	0.04	782	83.21
9	27,902	45	0.00091	3.2	0.06	387	85.14
10	29,826	40	0.0016	5.6	0.08	264	83.20

≠ No. 2 sample, starboard sluice, neglected as being in error; mean from No. 1 sample and #2 port sample taken.

• Recovery factor is the percentage of the total gold production of the dredge recovered on the mats and adjacent riffles, in the 1957 season.

In Table 14, the yardage dug during the test periods was calculated from the average daily rate for the particular month, and the percentage of stacker tailings is an estimate only, since no measurement has ever been made. From the recovery factor and the quantity of gold recovered on the mats, an estimate of the total gold recovered by the dredge during the test period is obtained, which can only be considered as approximate. The weight of 1 cubic yard of gravel is taken as 1.5 short tons which has been partly substantiated by the calculations for Dredge # 11, above.

Dredge # 4: Estimate of gold recovered during the test period $= 291 \times \frac{100}{60.95} = 478$ oz.

Loss of gold in tailings $= 144{,}016 \times 1.5 \times 0.5 \times 0.00040 = 43$ oz.

Estimated recovery of gold $= \frac{478}{521} \times 100 = 91.8$ %

Dredge #6: Estimate of gold recovered during test period $= 228 \times \frac{100}{73.88} = 309$ oz.

Loss of gold in tailings $= 37{,}359 \times 1.5 \times 0.5 \times 0.00065 = 18$ oz.

Estimated recovery of gold $= \frac{309}{327} \times 100 = 94.6$ %

Dredge # 8: Estimate of gold recovered during test period = $782 \times \frac{100}{83.21} = 940$ oz.

Loss of gold in tailings = $34,391 \times 1.5 \times 0.55 \times 0.00066 = 19$ oz.

Estimated recovery of gold = $\frac{940}{959} \times 100 = 98.1$ %

Dredge # 9: Estimate of gold recovered during test period = $387 \times \frac{100}{85.14} = 454$ oz.

Loss of gold in tailings = $27,902 \times 1.5 \times 0.55 \times 0.00091 = 19$ oz.

Estimated recovery of gold = $\frac{454}{473} \times 100 = 96.0$ %

Dredge # 10: Estimate of gold recovered during test period = $364 \times \frac{100}{85.20} = 427$ oz.

Loss of gold in tailings = $29,526 \times 1.5 \times 0.6 \times 0.0016 = 42.5$ oz.

Estimate recovery of gold = $\frac{427}{469.5} \times 100 = 91.1$ %

These percentage recoveries can only be considered as approximate, as the constants used in the calculations are only estimates, and there are considerable errors involved in the use of the hand sample assays to calculate the gold losses in the tailings. However, these calculations serve as an indication of dredge performance, although they will not necessarily indicate an average condition for the operating season, since the sample periods were short and operating conditions vary throughout the season.

It is evident from these calculations that the dredges have an operating range of recovery between 90 and 98 percent, which varies according to grade of gravel and operating conditions.

GOLD DISTRIBUTION ON DREDGES

An interesting feature of the hand sampling programme on all dredges arose in the distribution of the recovered gold between the port and starboard sluices. In most cases there was far more gold recovered on one side of the dredge than the other, as shown in the tabulation below:

Table 15. Distribution of Gold on Dredge Sluices

Dredge No.	Gold Recovered (oz.) Port	Starboard	Ratio	Rotation of Screen
4 - 1	152	74	2.05 : 1	Counter Clockwise
- 2	44	21	2.1 : 1	
6	123	105	1.15 : 1	Counter Clockwise
8	154	628	1 : 4.1	Clockwise
9	192	195	1 : 1	Counter Clockwise
10	113	251	1 : 2.2	Clockwise
11 - 5	75	230	1 : 3.06	Clockwise
- 11-12	65	197	1 : 3.03	
- 13	125	375	1 : 3	
- 16	84	155	1 : 1.85	
- 17	32	90	1 : 2.8	

While the smaller dredges, # 6 and 9, had an equal gold distribution, the recovery of gold was very disproportionate on the larger dredges. The cause of this inequality is the screen and distributor system. In theory, the sands, on passing through the screen, should gravitate into the distributor and overflow equally to the short sluices on either side of the dredge. However, due to the rotation of the screen, the sands issue from the screen casing at an angle and hence overflow the distributor unequally. It is evident that in those dredges where inequality exists, a clockwise direction of rotation of the screen is associated with the greater proportion of the gold being recovered on the starboard sluices, and a counter-clockwise rotation with the greater proportion on the port sluices (Table 15).

Since the ratio is as high as 4:1 on Dredge # 8, this condition could cause additional gold losses, if the dredges became overloaded. The larger dredges which are thus affected have plenty of riffle capacity, so that only in exceptional circumstances would danger of added loss exist, but any condition which tends to cause loss of gold will have a far greater effect on the overloaded side. The ratio of gravel distributed between the two sides is nowhere near as high as the gold ratio, but it is definitely uneven and should be rectified to improve the gold distribution.

The only proper solution would be to eliminate the distributor box and replace it with a riffle splitter under the screen casing. This arrangement has been tried, but heavy wear on the rubber lined riffles caused them to be abandoned. The advantage of the distributor box is that a bed of gravel in the bottom forms its own wearing surface. At present gates are placed on the appropriate side of the distributor to divert the flow to the lean side, but these are seldom used due to the danger of blocking the sluices.

A simple solution to the problem, although not perfect, would be to build up the overflow level of the distributor, on the side which recovers the bulk of the gold, to such a height that the gold recovery becomes more balanced due to the diversion of a larger part of the flow to the lean side. The distribution of gold between port and starboard sides is readily determined at each cleanup by weighing the gold separately on each side, as was done during the test programme. The level would probably only need to be built up 1 or 2 inches in each case to equalize the gold distribution. At equal gold distribution, the distribution of sands may be disproportionate in the opposite direction, but this would be of less consequence than an unequal gold distribution. Such a system would operate simply with no possibility of blockage, and the gates at present installed could be removed.

GOLD IN PYRITE

The assays of hand picked pyrite samples have confirmed the earlier assays which showed a low gold content. A summary of these assays and the pyritic content of the sand tailings from the dredges is presented below:

Table 16. Gold Assays of Pyrite and Pyritic Content of Dredge Tailings.

Dredge No.	Percent Pyrite In Sands	Pyrite Oz. Au. per ton	Assay Cent per ton	
4 (Bonanza)	0.04	0.021	73	
		0.02	70	(October, 1956)
		0.04	140	(March, 1958)
6 (Lower Sulphur)	0.08	0.021	74	
8 (Middle Sulphur)	0.08	?		
9 (Upper Sulphur)	0.11	?		
10 (Dominion)	0.15			
11 (Hunker)	0.23	0.0062	22	(Picked from riffles)
		0.0104	36	(Stacker tailings)

The pyritic content of the sands was lower than estimated, but as expected, Hunker had the highest content of all the creeks. The gold content of the pyrite from Bonanza, Lower Sulphur and Hunker Creeks was of much the same order, allowing for a small amount of possible salting, which suggests that the pyrite in these creeks had a common genesis during the mineralization of the area. Thus it might be predicted that the gold content of pyrite dug on Dredges # 8, 9 and 10 would also be of this same order.

The gold content of the pyrite, in addition to the very small pyritic content of the sands, renders it economically unimportant as a source of additional gold recovery. Even were the pyrite concentrated along with the free gold by mechanical means, the cost of processing the pyrite by roasting and cyanidation would not be repaid by its gold content.

SAMPLING ERRORS

Placer gold deposits along with alluvial tin ores are among the most difficult ores to sample and large sampling errors are often incurred. Several factors peculiar to this type of deposit magnify the sampling errors: (a) the large difference in density between the valuable mineral and the gravel, (b) the fact that the valuable mineral is completely liberated, and (c) the fact that in general such deposits are low in grade so that the ratio of valuable to non-valuable particles is extremely small. Large samples of the deposits and an extremely careful sampling technique are necessary to minimize errors during sampling.

The errors involved in such sampling problems are larger than in most base metal ores. Gy (6) has evolved a nomogram in which all the pertinent factors are incorporated to evaluate the statistical errors incurred in any given sampling operation. The nomogram takes no account of personal errors in the sampling operation, so that these errors which cannot be evaluated are additional to the statistical errors.

As described in the sampling procedure, 50 lb. lots of samples were sent for assay. Samples # 1 and 2 were sent as sampled, that is, maximum size 1/2 inch, whereas the rest of the samples were screened on 35 mesh and 50 lb. samples of each fraction were sent for assay. Assuming an average assay of 0.001 ounce/ton for the tailings, the nomogram shows that, for a 6 mesh top size, samples of 100 tons would incur errors within the limits plus and minus 20 percent of the actual assay value 95 percent of times, or that 95 percent of samples of 1000 tons each would incur errors within the limits of plus and minus 5 percent of the actual value. In other words, the statistical errors are extremely large in any practical sampling operation when such coarse, low grade material is involved. However, for the samples with 35 mesh top size and the same assay, a 50 lb. sample would be within plus and minus 50 percent of the actual value, 95 percent of times.

Thus it may be estimated that, due to the statistical and operating errors involved in the sampling of tail sluice sands during this investigation, the assay values of samples, calculated from the plus and minus 35 mesh fraction assays, were within the limits plus and minus 50 to 100 percent of the actual values for the samples. A certain assaying error was also involved since some of the assays were close to the limit of sensitivity of the assaying method; however, this error is only a small one compared with the statistical sampling error.

Some indication of the extent of these errors is obtained from Table 5, where head assays of samples have been calculated from the assays of screened fractions and compared with the actual assay of the whole sample. Thus for sample # 2, the assayed head was 62.5 percent of the calculated head, while for samples # 3, and 4 the respective ratios were 50 and 80 percent, for minus 35 mesh sands. These results are well within the limits determined above.

By comparison, it will be noted that the actual sulphur assay for # 2 sample was only 112.5 percent of the calculated assay. In this case, where the valuable mineral (pyrite) is a larger proportion of the sample and has a much lower density than gold, the errors are smaller - a 50 lb. sample insures a statistical error within plus and minus 30 percent of the actual value, 95 percent of times for 1/2 inch material.

The errors incurred during the actual taking of the samples have already been mentioned under the Procedure. There should have been no serious errors in the automatic sampling, except insofar as the use of a splitter in the stream to be sampled possibly introduced some bias into that section of the stream which passed through the automatic sampler. Cumulative with this error is a possible inaccuracy in the samples due to the rather long 30 minute interval between samples, but this effect would largely average out over the 3 and 4 day sample periods.

The hand samples, however, were subject to considerable errors since the high velocity of the tail sluice streams caused spillage from the sample scoop. The extent of this error is difficult to estimate, for there were no exact comparisons between the hand samples and identical automatic samples. However related hand samples, e.g. from both tail sluices on a dredge, were consistent with each other, which suggests that the errors involved were constant so that the samples were useful for comparative purposes. Also, in test period 11 - 12 on Dredge # 11, the hand samples were of the same order as the automatic sample of the # 1 long sluice which forms the greater part of the tail sluice sands, which indicates that the percentage error in the hand samples was not too great.

Assaying: As mentioned above, the fire assays were subject to a small error due to the limit of sensitivity of the assay method, which was 0.00005 ounce per ton. The amalgamation assays however, were subject to considerable inaccuracy, so that, when it became evident that the gold content of the pyrite was small and unimportant, and that the gold was largely free in the tailings, there was no purpose in continuing the expensive amalgamation assays. The free gold in the sands was very fine and would not amalgamate readily under the assay conditions. An extension of the amalgamation time to 3 hours to approach complete amalgamation of the free gold was not successful due to flouring of the mercury.

RECOVERY OF GOLD IN DREDGE SAND TAILINGS

The investigation has been notable in showing that all the dredges tested are operating reasonably efficiently in recovering the gold being dug. While there is a small quantity of fine gold in the tailings, there is insufficient to make any form of tailings retreatment profitable, and the pyritic content of the sands has been shown to be of no economic importance.

Any form of tailings retreatment for gold recovery would involve some form of preliminary concentrating plant at each dredge, which would require one man for its operation. The concentrate would then have to be transported to a central plant for extraction of the gold, either by amalgamation or more probably by cyanidation, which would also require at least one man for operation.

The approximate value of the tailings, from the dredges sampled, may be computed as follows for an operating season, using 1957 yardage figures and the tailings assays obtained this season.

Dredge	Yardage 1957 Season	Tons of Sands	Assay cent/ton	Total Value
4	1,744,843	1,308,632	1.4	$ 18,320
6	1,139,803	854,852	2.3	19,661
8	947,278	781,504	2.3	17,975
9	546,864	451,163	3.2	14,437
10	887,980	790,162	5.6	44,754
11	860,721	645,541	3.1	20,012
				$ 135,159

Assuming a concentration ratio of 50 : 1 at the dredges, which ratio is probably higher than could be achieved while maintaining good recovery, the cost of transporting the concentrates to Bear Creek can be determined at a minimum rate of 15 cent per ton-mile. To this add the labour cost - 21 men (7 operations, 3 shifts) for a 7 month operating season at $1.33 per hour, unskilled rate.

Dredge	Tons of Concentrate	Miles to Plant	Ton-Miles	Total Cost
4	26,173	11.0	287,903	$ 44,685
6	17,097	44.1	753,978	113,097
8	15,650	34.6	541,490	81,244
9	9,025	32	288,800	43,320
10	15,984	36	575,424	86,314
11	12,911	5.1	65,846	9,877
	Labour 105,840 man hours at $1.33			140,767
				$ 539,284

To these costs must be added Company overhead, power and maintenance costs, and the amortization of the plant required at each dredge and at Bear Creek. Thus it is readily apparent that even if the assays had to be increased by 100 per-cent to allow for statistical sampling errors, recovery of gold losses in the tailings would not be profitable. Nor would it be profitable in the event that the price of gold trebled in the next few years.

SUMMARY AND CONCLUSIONS

(1) Dredge # 11 has ample capacity for recovery of plus 100 mesh gold, but finer gold than this is not effectively recovered due to the velocity of water that must be maintained on the sluices.

(2) Gold recovery increases with the gold content of the gravel, provided the riffle system is not overloaded, but the content of the gravel has no definite effect on the tailings losses.

(3) An increase in riffle capacity would not be expected to increase the recovery of fine gold appreciably.

(4) The average gold content of the samples taken from the starboard long sluices # 1, 2, and 3 on Dredge # 11, was 2.1, 5.1, and 10.0 cent per ton respectively, but the latter two figures are considered to be high due to abnormal operating conditions.

(5) A composite tail sluice sand assaying 4.5 cent per ton and the consequent percentage recovery of gold at 95.6 percent have been calculated for Dredge # 11, to represent the average performance in a cut across Hunker Valley. The figure of 4.5 cent per ton is considered to be higher than normal for the Dredge, because of abnormal conditions during part of the sampling programme. An estimated figure of 3.1 cent per ton should be closer to the average performance. This figure indicates a recovery of 97.0 percent for the Dredge, which is closer to the average recovery of 97.9 percent determined accurately on the short sluices # 1, 2 and 3, and long sluice # 1.

(6) The performance of Dredges # 4, 6, 8, 9 and 10 has been estimated from the assays of hand samples of the tailings and the gold recovered during the sampling period:

Dredge	Tailings Assay (Cent per ton)	Estimated Recovery (Percent)
4	1.4	91.8
6	2.3	94.6
8	2.3	98.1
9	3.2	96.0
10	5.6	91.1

(7) The distribution of the gold recovered on the larger dredges between port and starboard sluices was found to be very unequal; the ratio being as high as 1:4 on Dredge # 8. A method for correcting this unequality is proposed, whereby the overflow level of the distributor box is built up on the overloaded side to divert more of the flow of sand and gold to the lean side.

(8) The pyritic content of the sands was found to be quite low, and the gold content of samples of pyrite from Bonanza, Hunker and Lower Sulphur Creeks varied around 70 cent per ton. Thus pyrite is not an economic source for additional gold recovery.

(9) On the basis of the low gold content of the sand tailings from the dredges studied, and from a consideration of operating labour costs, the cost of transportation of concentrates to a central plant at Bear Creek, overhead, power and maintenance costs, and the amortization of the capital investment for plant, the retreatment of these tailings for additional gold recovery cannot be considered an economic proposition.

RECOMMENDATIONS

(1) No further consideration should be given by the Company to the possibility of gold losses in tailings being economically recoverable.

(2) The proposed modifications to distributor boxes, in order to more equally distribute the gold recovered on the dredges, should be carried out before the next operating season.

H. R. Rawling

27th November, 1958

REFERENCES

(1) J.W. Scott - Report on Gold Room Operations, December, 1941.

(2) J.F. Sealey - Memorandum to McLeod White, March, 1948.

(3) Haughen Metallurgical Processes Ltd. - Sands from #8 Dredge, March 1952.

(4) J.F.N. Miller - Investigation of Tailings Loss on Dredge # 8, August, 1955.

(5) J.C. Nixon - Report of a visit to Yukon Consolidated Gold Corporation, Limited, June 1957.

(6) P. Gy - Sampling Nomogram, Minerais et Metaux, Paris.