U. JANSONS

FINAL REPORT

CONTRACT NO. H0122104

to

UNITED STATES BUREAU OF MINES

DESIGN OF A BENEFICIATION SYSTEM FOR EVALUATION AND RECOVERY OF GOLD AND ACCESSORY MINERALS FROM ALASKAN BEACH DEPOSITS

from

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Final Report
Contract No. H0122104

"Design of a Beneficiation System for Evaluation and Recovery of Gold and Accessory Minerals from Alaskan Beach Deposits"

INTRODUCTION

On May 16, 1974, a semi-final report was submitted on this contract in conjunction with five copies of a Master of Science thesis which completed participation of a graduate student on the project.

At this time it was requested that submission of the final report be delayed for the following reasons:

1. It was considered desirable that the froth flotation process used for the recovery of alluvial gold be compared with a centrifugal amalgamation process in regards to accuracy of evaluation and efficiency of recovery.

   This latter study was made possible under a separate contract with Consolidated Purchasing and Designing Inc. to study the operating parameters and recovery efficiency of 6" and 12" centrifugal amalgamation bowls.

2. Modification No. 1 of Contract No. H0122104 involved Bureau of Mines participation in National Science Foundation Grant No. GF38893 which is a cooperative research agreement with the Republic of China on the subject of "Evaluation and Recovery of Detrital Heavy Minerals."

   It was also considered desirable to include the interim report on the 1974 trip to Taiwan in this final report.

This document on Contract No. H0122104 will therefore consist of information primarily concerned with the amalgamation unit as a means of evaluation and recovery of gold from beach sand deposits, and a report on the current status of the cooperative research program with the Republic of China.

CENTRIFUGAL AMALGAMATION UNIT

Introduction

A research contract with Consolidated Purchasing and Designing Inc. involved an investigation into the potential of using a centrifugal amalgamator (Gilkey Bowl) as a method of recovering fine gold from alluvial material. This study was two-fold in that the use of 6" and 12" bowls were investigated at the University of Alaska with graduate student support, and an experimental 36" bowl was placed on an operating dredge at El Bagre, Colombia, South America.

Operating parameters of rim speed, pulp density and pulp flow as they effect mercury retention and gold recovery for the 6" and 12" bowls were studied in a closed circuit system primarily as the thesis subject for the graduate student. This thesis entitled "Parameter Impact on the Performance Ability of the Centrifugal Forced Amalgamator" is included in this report in Appendix A.

After the necessary parameters were established to assure a minimum of mercury loss and a maximum recovery efficiency in the closed circuit system, open circuit tests were conducted in a single stage operation.

Material used for these tests consisted basically of two types: Typical placer gravel material obtained from the Nome operation of U.V. Industries, and heavier beach sand concentrates obtained from the Cape Yakataga and Nome areas.

The placer gravel was screened at minus 1/8 inch to meet the normal feed size criteria for bowl operation, but unlike the sands encountered at El Bagre, an unusual amount of slime material was encountered. On the other hand, the beach sands were void of material finer than 150 mesh.

For the open circuit testing the unit was arranged so that the dry solids were fed by the vibrating feeder to a mixing trough where 9 GPM of water was added to wet down the solids and wash the material into the bowl feed inlet. The balance of the water for pulp density and volume specifications was obtained by recycling the water from hopper No. 1 and by-passing the desired flow rate through the bowl. This was made possible by inserting a stand pipe in the hopper so that recycled water was obtained from a zone in which the solids had settled. This arrangement is shown in Figure 1.

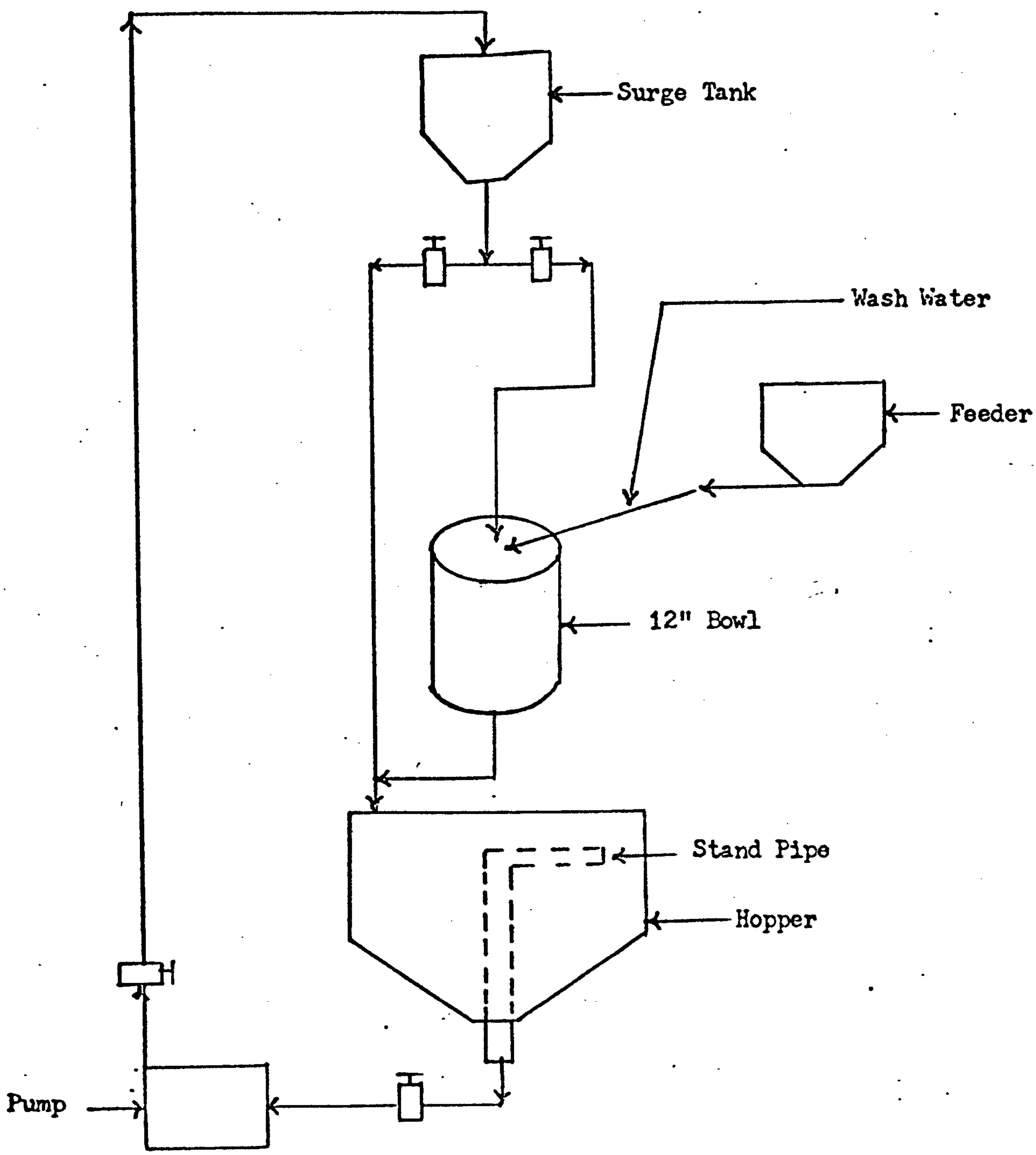


FIGURE 1

OPEN CIRCUIT SYSTEM

Dry Creek Placer Gravel

This material, with a specific gravity of 2.65, was screened at 1/8 inch and the minus fraction split into two portions for separate runs through the bowl. The operating conditions and the results of these tests are shown in Table No. 1.

A representative sample was obtained from each portion in an attempt to arrive at a head analysis. Due to the improbability of obtaining a representative portion of the liberated gold it was decided to use the flotation process, analize the concentrate and calculate a head value. This procedure was complicated by the fact that the minus 1/8 inch material contained considerable slimes. Each sample to be floated was de-slimed four times allowing a 5 minute settling time prior to removing the slime material. Using this procedure the slime material in each sample amounted to 21% of the flotation feed.

To arrive at a calculated head value the concentrate and slime fractions were analized separately, and a weighted value used to arrive at the final head analysis. The results of these analyses are as follows:

Sample	Value Floated Sample oz./T	Value Slimes oz./T	Value Head Sample oz./T
Dry Creek No. 1	0.013	0.005	0.011
Dry Creek No. 2	0.007	0.005	0.007

When comparing the analysis of the flotation results with the results obtained by the bowl, as shown in Table 1, it is evident that the bowl is more effective then froth flotation in recovering gold from material containing a high percentage of slimes. With sizes ranging from minus 1/8" to slime material the efficiency can probably be increased by further classification of the bowl feed.

Yakataga Beach Concentrates

The natural beach concentrate obtained from the Cape Yakataga area contains gold that is decidedly flat and flakey in character. Because of the shape, it has a natural bouancy which can be enhanced in the froth flotation process to ensure practically 100% recovery. On the other hand this shape factor would be detrimental to the amalgamation process in that gold particles would tend to float out in the pulp

TABLE 1

Operating Data for Open-Circuit Tests

Unit	Nome Dry Creek #1	Nome Dry Creek #2	Yakataga Beach Conc. #1	Yakataga Beach Conc. #2	Nome Beach Conc.
Specific Gravity of Sample	2.65	2.65	3.57	3.57	3.57
Weight of Sample	86.0 lb.	91¼ lb.	107.0 lb.	121¼ lb.	46.0 lb.
Pulp Density of Feed	30%	30%	20%	20%	20%
Volume of Pulp	30 GPM	30 GPM	30 GPM	30 GPM	30 GPM
Rim Speed of Bowl	1326'/min.	1326'/min.	1326'/min.	1326'/min.	1326'/min.
Mercury Volume	300 cc.	300 cc.	300 cc.	300 cc.	300 cc.
Mercury Loss	0.0	0.0	0.0	0.0	0.0
Gold Recovered	33.0 mg.	12.0 mg.	74.25 mg.	56.51 mg.	78.70 mg.
Gold Recovered ounces/Ton	0.025	0.008	0.045	0.030	0.110
Gold In Tailings	Trace	Trace	24.75 mg.	47.18 mg.	27.22 mg.
Tailing Value ounces/Ton	Trace	Trace	0.015	0.024	0.038
Recovery In Bowl	-	-	75.0%	54.5%	74.3%

without contact to the mercury surface. The average gold size distribution in the Yakataga sand is shown in Table 2.

This sand concentrate is also characterized by a fairly high specific gravity (3.57) and a size distribution concentrated in the 28/65 mesh fractions with a minimum of minus 150 mesh material. An average screen analysis of this material is shown in Table 3.

A head sample was split from this material and the bulk sample split into two portions for individual runs through the 12" bowl. The head sample was split into eight portions to arrive at an average head analysis.

Recognizing the ease of gold floatability, each analytical sample was floated to obtain a gold concentrate, the concentrate analyzed and knowing the concentration ratio, the results calculated to arrive at a head analysis. These data are shown in Table 4, where an average head value of 0.54 oz./Ton is indicated.

Both Yakataga samples were processed through the 12" bowl with the operating parameters and results as shown in Table 1. Representative samples were obtained from the tailings of each bowl run and from the sand fraction remaining in the bowl after shutdown. These samples were again floated to recover gold lost during bowl operation. From this data and the recovery obtained in the bowl operation, head analyses were calculated for each bowl run. These values calculated at 0.060 ounces/Ton for run #1 and 0.054 ounces per ton for run #2 which are in good agreement with the average floated value as shown in Table 4.

As shown in Table 1, gold recoveries of 75.0% and 54.5% were obtained in run 1 and 2 respectively. A flowsheet for this test procedure is shown as Figure 2.

Nome Beach Concentrate

A sample of natural concentrate from the Nome beach was processed in a similar manner as the Yakataga samples. The operating parameters and results obtained from this test are also shown in Table 1. These data indicate a 74.3% recovery of gold in the 12" bowl which is comparable to Yakataga No. 1 sample.

TABLE 2

Gold Size Distribution, Yakataga

Tyler Mesh		Analysis	Percent	Cumulative Distribution	
Passed	Retained	ounces/ton	of size	Retained	Passed
-	10	Nil	0.00	-	-
10	14	Nil	0.00	-	-
14	20	Nil	0.00	-	-
20	28	0.070	3.11	3.11	100.00
28	35	0.030	6.46	9.57	96.89
35	48	0.020	5.87	15.44	90.43
48	65	0.070	15.82	31.26	84.56
65	100	0.150	11.74	43.00	68.74
100	150	1.230	14.69	57.69	57.00
150	200	25.950	18.50	76.19	42.31
200	270	225.450	20.09	96.28	23.81
270	400	22.190	1.98	98.26	3.72
400	Pan	6.520	1.74	100.00	1.74
Total or average		0.112	100.00		

TABLE 3

Screen Analysis, Yakataga

Tyler Mesh		Weight	Weight	Cumulative Weight %	
Passed	Retained	Grams	Percent	Retained	Passed
-	10	52	0.24	0.24	100.00
10	14	122	0.55	0.79	99.76
14	20	331	1.50	2.29	99.21
20	28	1,100	4.99	7.28	97.71
28	35	5,327	24.15	31.43	92.72
35	48	7,271	32.97	64.40	68.57
48	65	5,593	25.36	89.76	35.60
65	100	1,936	8.78	98.54	10.24
100	150	296	1.34	99.88	1.46
150	200	18	0.08	99.96	0.12
200	270	1	0.00	99.96	0.04
270	400	2	0.01	99.97	0.04
400	-	6	0.03	100.00	0.03
TOTALS		22,055	100.00		

TABLE 4

Head Analysis, Yakataga Sand

Sample No.	Sample Wt. Grams	Concentrate Wt. Grams	Ratio of Concentration	Conc. Value oz./ton	Calculated Head oz./ton
1	821	0.613	1339	123.23	0.092
2	871	0.855	1019	54.92	0.054
3	844	0.648	1302	62.42	0.048
4	863	1.699	508	26.44	0.052
5	877	0.645	1360	56.52	0.042
6	854	2.266	377	17.50	0.046
7	835	2.106	396	18.97	0.048
8	842	1.005	798	41.19	0.052
Average	851	1.230	692	37.13	0.054

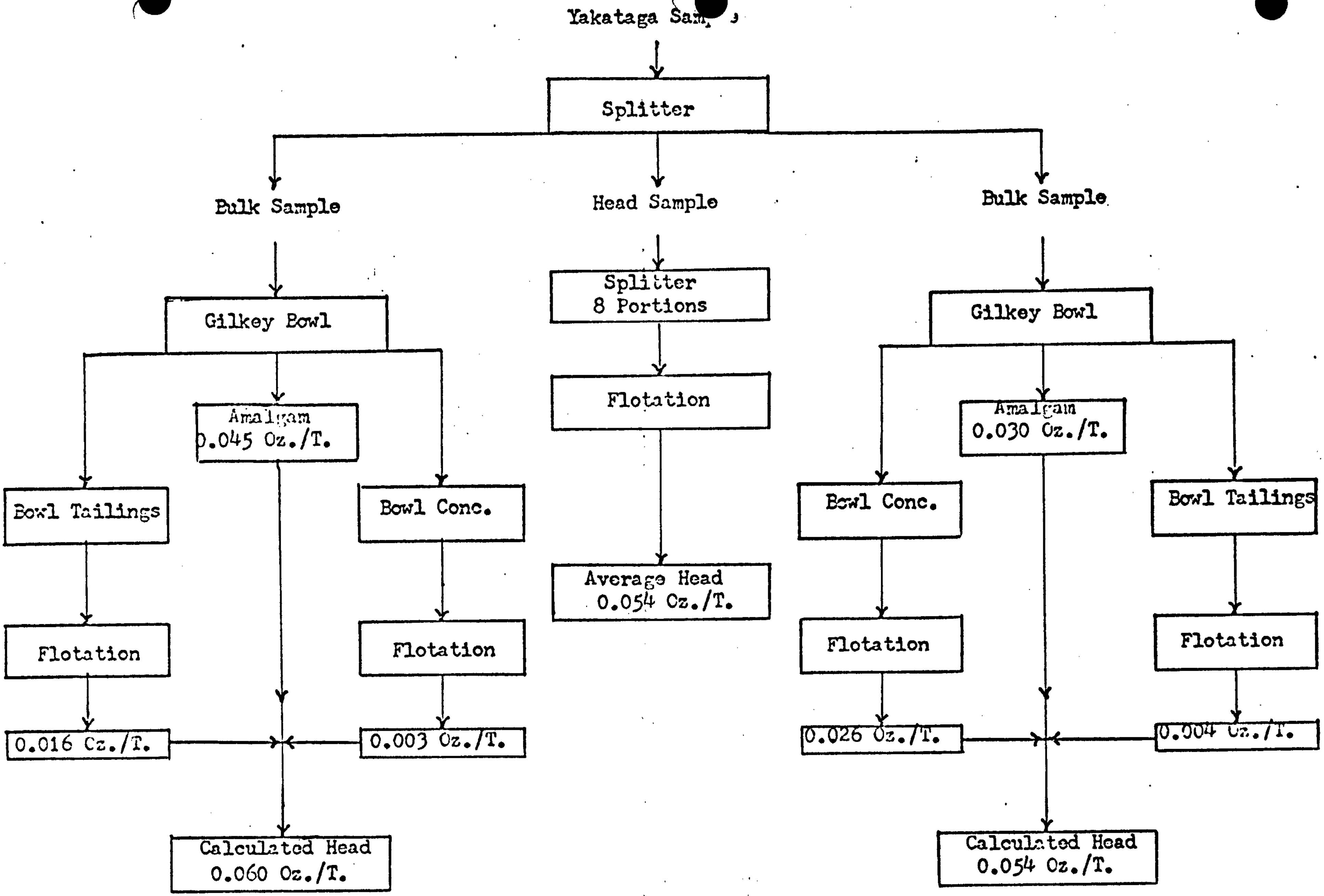


FIGURE 2
TESTING & ANALYSIS FLOWSHEET

Conclusions and Recommendations

It is evident that the bowl is superior to flotation in recovering gold from material that contains a large percentage of slimes. The slime material evidentally is helpful to the stratification process, but in all probability the process would also benefit by further classification of the bowl feed to remove the coarser mesh sizes.

Processing of heavier concentrates that are void of slimes with gold that has a tendency to float would appear off-hand to stretch the capabilities of the bowl. However, if it is recognized that the gold must be wetted to reach the mercury surface, the obvious answer is to provide proper pretreatment prior to feeding the bowl. This can be accomplished by premixing with the possible addition of a wetting agent.

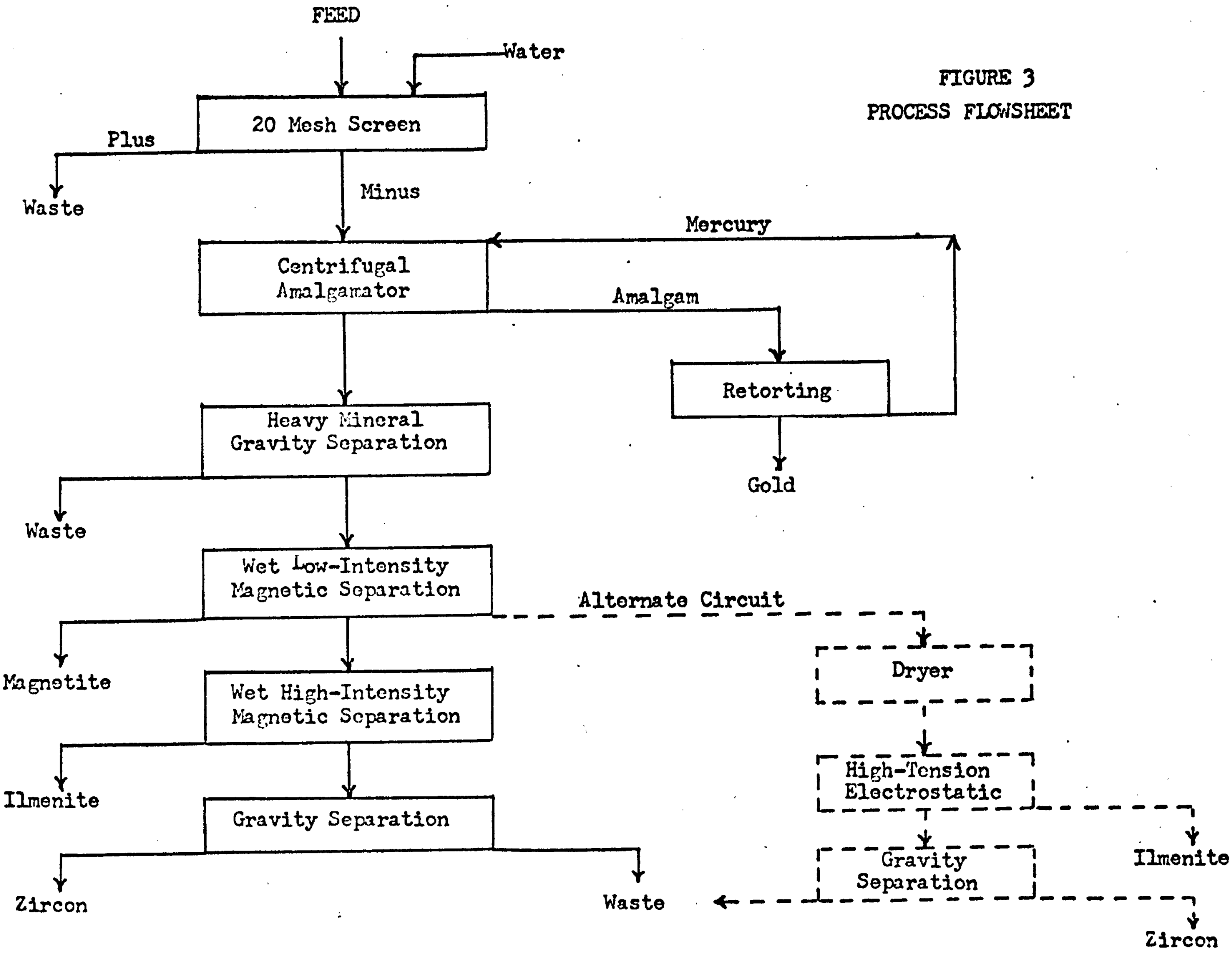
Recognizing the natural floatability of gold in beach sand deposits, and the simplicity of the amalgamation unit in comparison to froth flotation for gold recovery in an initial stage prior to heavy mineral concentration, the flowsheet shown in Figure 3 is suggested as a means of accomplishing this.

In general, the 12" bowl worked well with both types of material and suffered no mercury losses. It appears that efficient operation will hinge to a great extent on being able to supply a non-fluctuating feed to the unit as far as pulp density and volume are concerned. A classified feed will probably also be beneficial.

The investigation into the use of the centrifugal amalgamator, sponsored by Consolidated Purchasing and Designing Inc., will be continued with studies to determine the effect of classified feeds to the bowl in an open circuit system. These studies will also include the use of wetting agents and proper premixing to increase the efficiency of gold recovery from beach sand deposits. This system will also incorporate the use of jigs as a recovery unit for the classifier oversize. A contemplated flowsheet for this arrangement when dealing with typical placer material is shown in Figure 4.

Studies will also be conducted on changes in the physical configuration of the bowl such as effects of changes in the mercury column height and pulp feed mode.

FIGURE 3
PROCESS FLOWSHEET



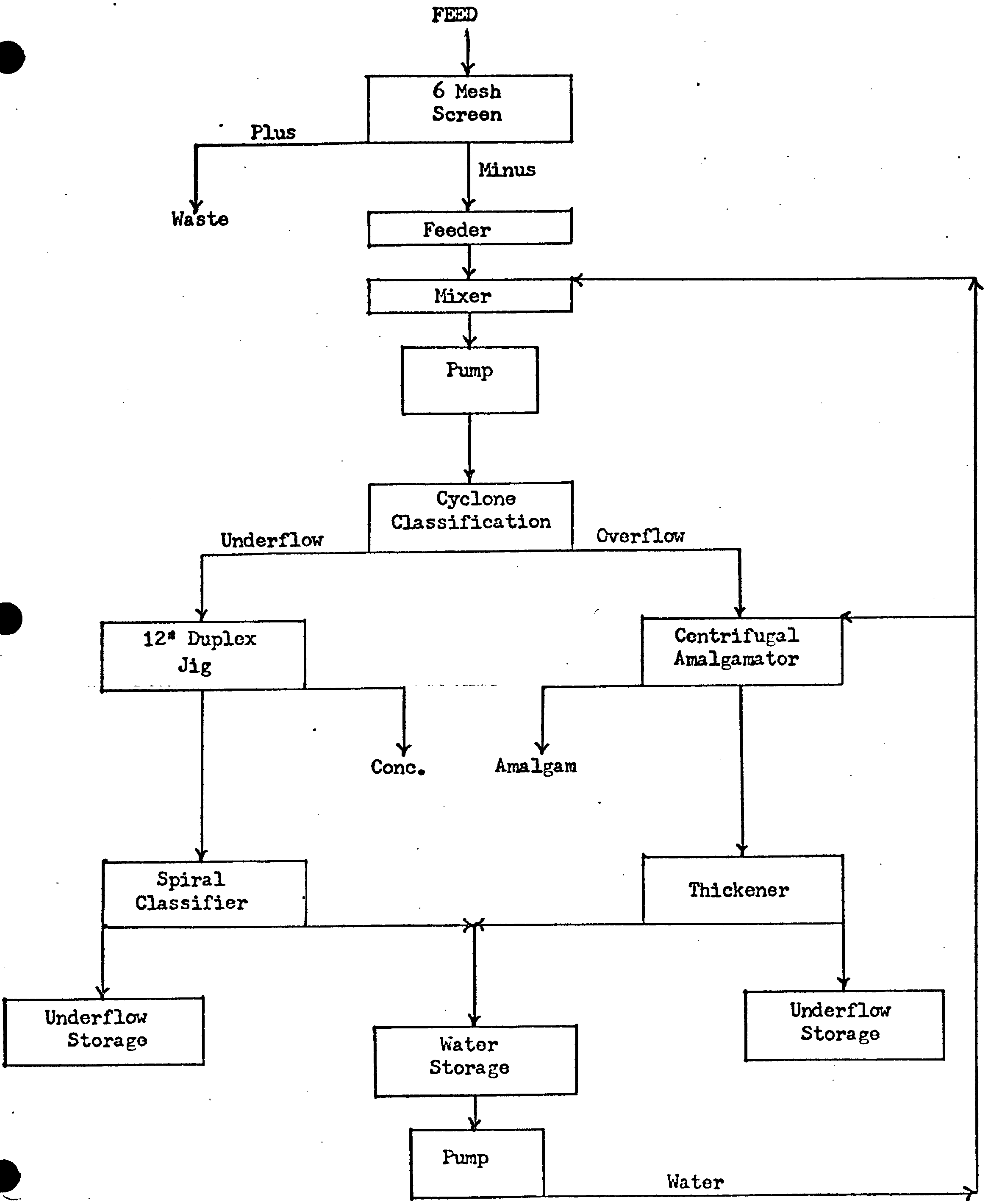


FIGURE 4
TESTING FLOWSHEET

COOPERATIVE RESEARCH PROGRAM

Introduction

As Modification No. 1 of Contract No. H0122104, the United States Bureau of Mines has participated as a sponsor with the Mineral Industry Research Laboratory of the University of Alaska in an international cooperative research project. This project as agreed upon between the National Science Council of the Republic of China and the National Science Foundation of the United States is concerned primarily with the Evaluation and Recovery of Detrital Heavy Minerals.

The writer, as American Principal Investigator, spent a month in Taiwan during the summers of 1973 and 1974. During those periods he collaborated with Dr. Liu Hok-Shing and Mr. Lee Ven-Chung of the Mining Research and Service Organization of that country on heavy mineral investigations. A report on the 1973 trip has been previously submitted to the Bureau of Mines, and the interim report summarizing the activities of 1973 and 1974 has been submitted to the Office of International Programs of the National Science Foundation. A copy of this interim report is included as Appendix B of this report.

Current Status

By agreement between the National Science Council of the Republic of China and the National Science Foundation of the United States, this program has been extended until November 1977. Proposal No. U4063 to the Chief, Office of University Relations of the Bureau of Mines has also requested continued sponsoring support from the Bureau of Mines on this project. This also has been approved effective with the coming fiscal year.

As indicated in the joint proposal, the writer will spend a month in Taiwan this summer in further investigations on heavy minerals and other projects of mineral beneficiation being studied by the Mining Research and Service Organization.

Of special interest will be collaboration with Dr. Liu Hok-Shing on a heavy mineral recovery plant currently under construction at Tainan to process monazite, zircon, ilmenite and magnetite from expected mining operations on the off-shore bars along the west coast of Taiwan.

As the continuation of the Bureau of Mines participation in this cooperative program is part of Contract No. H0122104, this contract should remain open although this constitutes a final report as far as the Alaskan Beach Sand studies are concerned.

APPENDIX A

PARAMETER IMPACT ON THE PERFORMANCE ABILITY OF THE CENTRIFUGAL FORCED AMALGAMATOR

A

THESIS

Presented to the Faculty of the
University of Alaska in partial fulfillment
of the Requirements
for the Degree of

MASTER OF SCIENCE

By
William James Anderson, B.S.

Fairbanks, Alaska

May 1975

ABSTRACT

The performance abilities under varying sets of feed and operating conditions, of a 6" diameter and a 12" diameter Gilkey Centrifugal Amalgamator, were investigated in this study.

The findings of this study show that the effect on the amalgamator's performance of input parameters, especially of pulp density of feed and rotation rate of the amalgamation bowl, is critical. Relatively small parameter variations can mean the difference between mercury loss or mercury retention modes and gold recovery efficiencies ranging from near 100% to near 15%.

Evident was a response difference to parameter variations between the two amalgamators caused by an interior configuration difference in the two amalgamation bowls.

Practical operating ranges were found to include lower rotational speeds and pulp densities for the 6" bowl, and a range of rotational speeds at higher pulp densities for the 12" bowl.

ACKNOWLEDGEMENTS

The completion of an investigation of this type depends on the indispensable cooperation and assistance of various people. I would therefore like to express my sincere appreciation especially to Consolidated Purchasing and Designing Incorporated for their project funding and technical assistance.

I am grateful to Dr. Donald J. Cook where his assistance in the capacity as my academic advisor, chairman of my advisory committee, instructor, project overseer, and friend, added encouragement and greatly contributed to my work.

The support of my advisory committee, Dr. Donald J. Cook, Dr. James B. Tiedemann; and Dr. Ernest N. Wolff is gratefully acknowledged.

And thanks to William McClintock II for his technical assistance in data gathering and keeping the machinery working.

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CHAPTER 1

INTRODUCTION

1.1 Description Of The Testing Program

This project investigated a particular type of centrifugal amalgamator, the Gilkey Bowl, in an effort to prove or disprove its ability as an efficient and practical device for the recovery of finely divided particles of placer gold. To this end a testing program was conducted and experimental evidence was obtained concerning the amalgamator's performance on parameter-controlled feed material as delivered from a custom built pilot plant.

The testing program's objective was to discover the centrifugal amalgamator's optimum potential through feed and operating parameter control, rather than by mechanical alteration of the existing device. Optimum potential for the centrifugal amalgamator was considered to be the simultaneous achievement of maximum continuous feed capacity, minimum mercury loss from the bowl, and maximum per cent gold recovery, from the processing of various typical auriferous alluvial material.

1.2 Program Motivation

This project was undertaken in part because of the fact that there is a need to provide a practical and more efficient recovery system to process particular types of gold bearing alluvial deposits. Such deposits cause existing processing systems to suffer poor gold recoveries due to the inherent obstinate nature of the physical condition of the gold particles contained therein.

It is generally conceded that, through many years of use and under widely varied conditions, the traditional gold recovery devices such as sluice boxes, jigs, cyclones, shaking tables, and amalgamation plates have been nearly developed to their limits of practical efficiency. Moreover there exist vast quantities of technically economic gold placer ground in which such conditions as the shape factor or particle size of the gold put the high gold recovery capabilities of even the best of engineered recovery systems beyond the realm of the practical. This aspect therefore suggested that to solve the dilemma the more rewarding course was to try to investigate and perfect a new or different recovery system rather than to try to increase the recovery efficiency of existing traditional machinery. The centrifugal amalgamator was thought to offer such an opportunity.

1.3 Previous Works

The concept of the centrifugal amalgamator is not new. A literature search revealed the fact that the centrifugal amalgamation process had been conceived before the turn of the century. Many years passed however before mechanical inventiveness, evidenced by the production of various mechanical designs of the centrifugal amalgamator, caught up with the theoretical stage.

Unit descriptions of some of the more noteworthy centrifugal amalgamators such as the Bazin Machine, the Taber Amalgamator, the Thayer Machine, (3)* and the Lorentsen Machine, (1) revealed that every machine type was based around the fundamental process of achieving gold recovery through the amalgamation of gold particles into a mercury layer which was held against the inner wall of a rotating bowl container.

The most interesting aspect of the literature search showed that great disagreement exists as to the performance of centrifugal amalgamators. For example, D. Campbell Mackenzie (5) in 1937 gave data which showed that the gold recovery efficiency of the Lorentsen Machine was consistently in the 93 to 99.9 per cent range. Arthur Taggart (8) however in his 1945 Handbook Of Mineral Dressing devoted insignificant attention to centrifugal amalgamators, noting primarily that they were not used with much success. This conflict has never to this writer's knowledge been satisfactorily resolved.

It is most significant to note that the literature search disclosed evidence which pointed out that attempts to improve the centrifugal amalgamator's performance had constantly focused on mechanical alteration of existing devices, rather than on

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*Numbers in parentheses refer to numbered references in the bibliography.

feed and operating parameter research and control. This can be seen by the typical developmental evolution sequence of amalgamators. For example the Bazin Machine was a crude first attempt, then to improve performance the Taber Machine utilized a different feed arrangement. The Thayer Machine later added an electrolysis process, and the Lorentsen Machine still later tried to apply differential acceleration forces between the feed and the mercury. In addition there was noted a marked lack of available data concerning systematic operating parameter guidelines for any machine. This strengthened the previous evidence, and therefore added encouragement to this project insofar as it was felt that parameter control was essential to the device's success.

1.4 Theory Of Application

The centrifugal amalgamator is a device designed to recover liberated gold particles from a continuous-feed sand-type slurry material, through the forced amalgamation of the gold particles into a captive mercury layer. Forced amalgamation is achieved by increasing the sedimentation effects on the gold particle's movement toward the mercury layer by artificially increasing the acceleration applied to those gold particles (6)[‡].

[‡]Stokes and Newton (7) put forth theoretical formulations showing the relationship between the settling velocity of particles in liquid, a representation of sedimentation properties, as being directly proportional to gravitational acceleration, and the difference between the particle density and the effective

The effective acceleration increase is obtained by applying the magnitude of the centripetal acceleration, $\frac{(\text{velocity})^2}{\text{radius}}$, from a rotating bowl container, in place of the earth's weaker gravitational acceleration (2).

The centrifugal amalgamator's theorized virtue is based on the fact that as gold particles are subjected to an increased gravitational force, the resulting increased sedimentation effect on the gold particles allows more gold-mercury contact and hence better gold recovery.

The process of differentially increasing the sedimentation effects on gold particles in a gangue mixture by applying a constant value gravitational force increase to all particles present may at first seem ineffective. The success of this endeavor depends on the unique condition of the gold particles. Coarse or non-flat gold particles do not merit consideration here because the density difference between them and the gangue particles is pronounced enough that the natural force of their mass times gravitational acceleration results in such a settling velocity as to effect a good gravity concentration. Very flat gold particles

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liquid density. It is also inversely proportional to the absolute viscosity of the liquid. Furthermore they maintained that the particle's size and shape is most important, where larger particles and spherical particles settle most rapidly while smaller particles and tabular particles settle least rapidly.

however might possess such a large surface area-to-mass ratio that frictional retardation resulting from contact of the total particle surface area with liquid or other suspended particles, compounded by possible problems such as incomplete wetting of the total particle surface area, and other buoyant phenomena, might be sufficient to successfully counteract the once effective gravitational settling impetus. If a large gravitational acceleration increase is applied to such suspended particles it would therefore seem reasonable that a gold particle settling mode can once more be achieved by that process which increases the magnitude of gravitational settling while not simultaneously increasing the retarding effects. It is hoped that the centrifugal amalgamator will accomplish this process.

1.5 Unit Description

This section is a description of the physical configuration and mode of operation of the particular centrifugal amalgamator around which this project is based. This device is a design variation known commercially as the Gilkey Bowl.

The heart of the device consists of an amalgamation bowl, symmetrical about the vertical axis around which the bowl rotates. Since the bowl itself is the vehicle for the fundamental recovery process, an in depth look at the bowl is in order at this point. (Test results of two model sizes of the Gilkey Bowl appear in this report, therefore the following unit descriptions treat each model size individually where necessary.)

The amalgamating bowl is in essence a symmetrical, open-mouth-upward, bowl-shaped, steel container. (See figures 1 and 2) It is dynamically balanced around a drive spindle secured into an acceptor sleeve which resides in the bottom center of the bowl. The bowl's inner peripheral wall is of a right cylindrical configuration, and has in it a recessed area which forms a right cylindrical channel.

In practice a volume of mercury, less than the volume of the cylindrical channel, is placed into the bowl. The rotation effects of the bowl pack and hold the mercury in the recessed channel in the bowl wall, thereby forming a right cylindrical sleeve of mercury around the bowl's inner perimeter. Feed slurry is fed into the rotating bowl under a small gravity head via a feed delivery pipe. (See figure 3) This pipe extends into the bowl from the top, and extends over the spindle sleeve. The pipe terminates just above the bowl floor but extends below the lower horizon of the recessed channel limit. Upon entering the bowl volume at the bottom of the bowl, the feed particles in the slurry accelerate toward the inner perimeter of the bowl. To prevent mercury flouring and the resulting mercury loss from the bowl caused by feed particles impinging directly on the mercury, there is provided a metal striking surface situated just below the mercury sleeve. It is of such a configuration that it directs the pulp flow more or less smoothly upward over the mercury surface and out of the bowl.

Gold recovery occurs in the bowl during the amalgamation

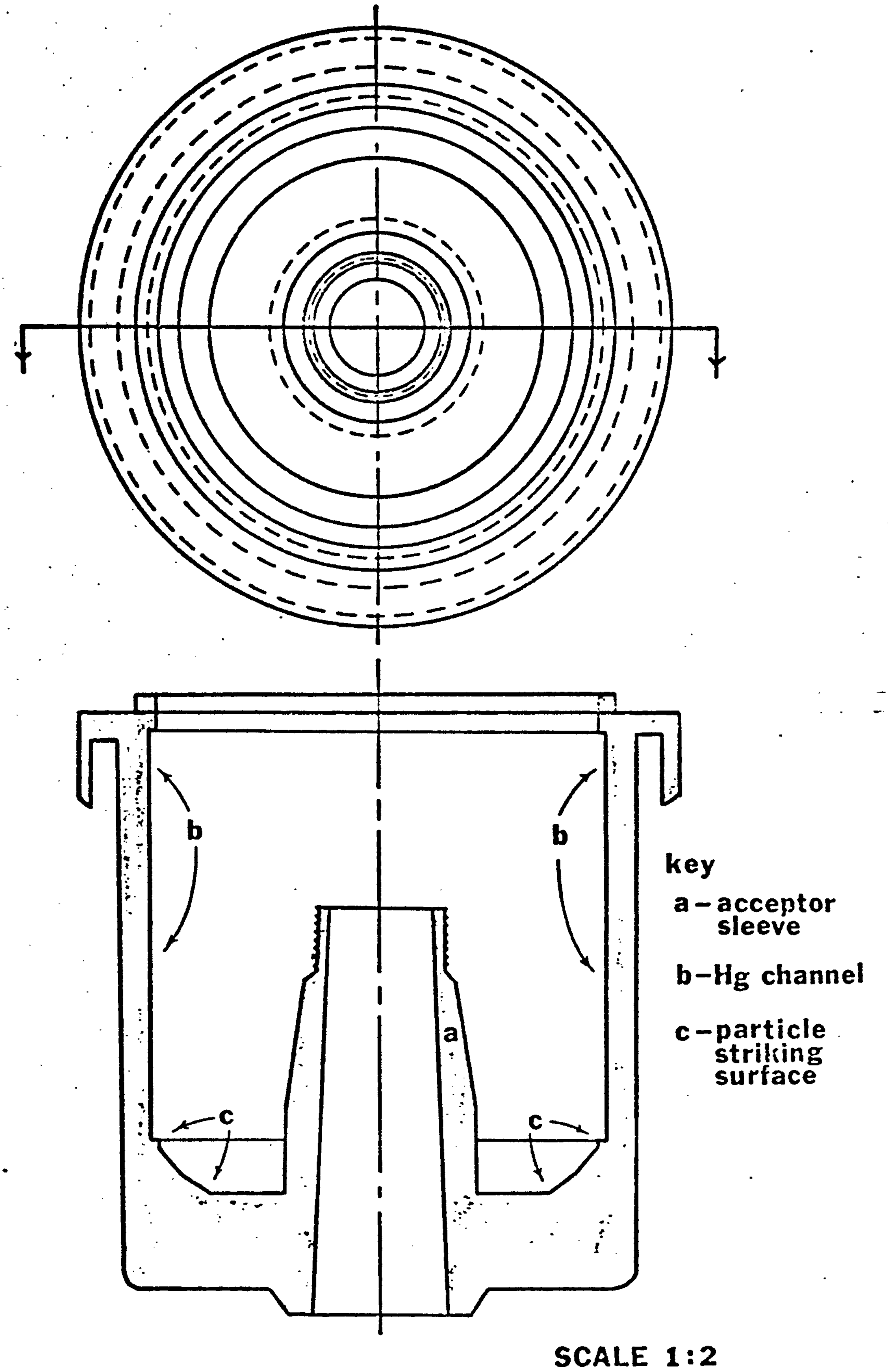


Figure 1. Plan and Section of 6" Gilkey Bowl

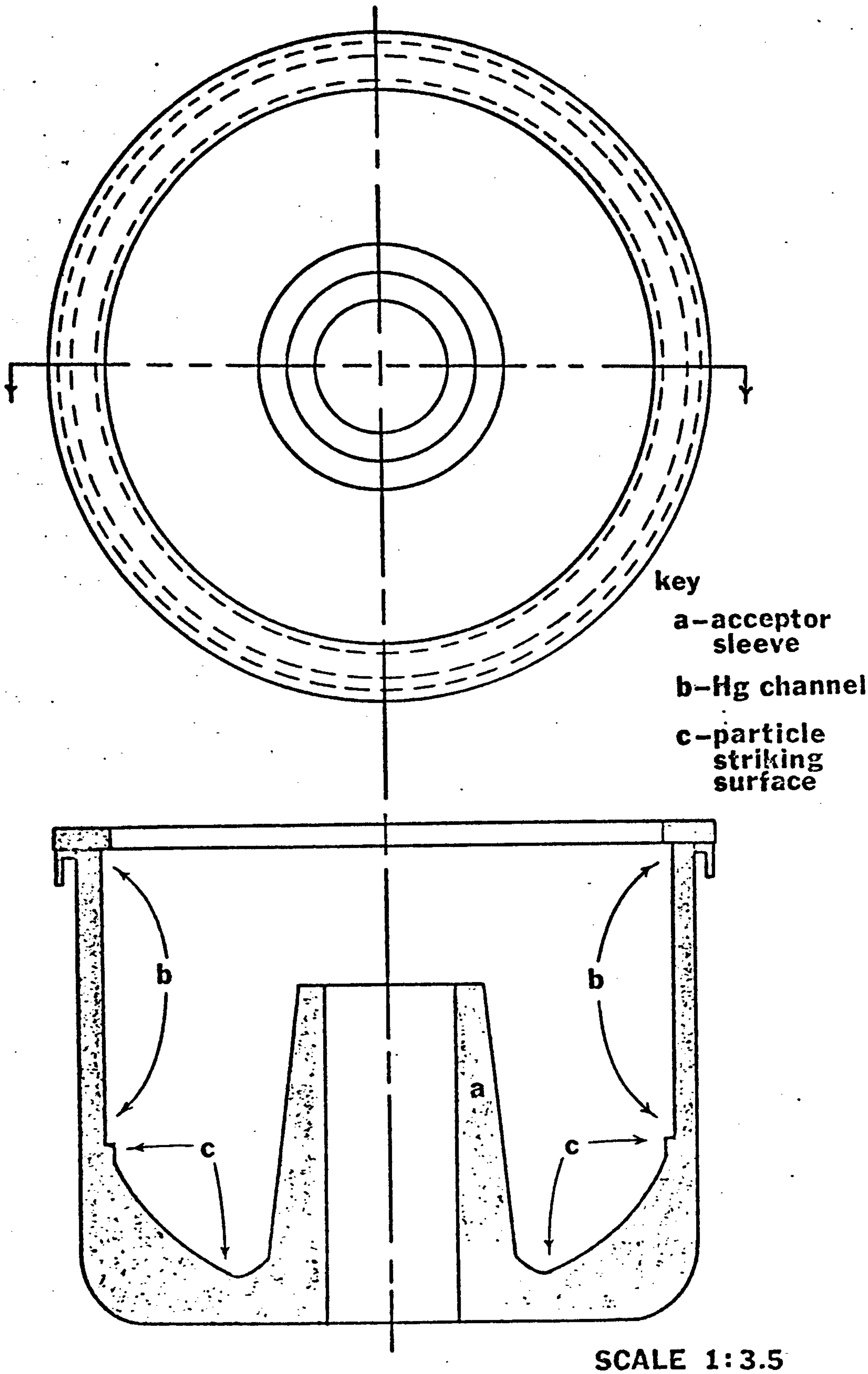


Figure 2. Plan and Section of 12" Gilkey Bowl

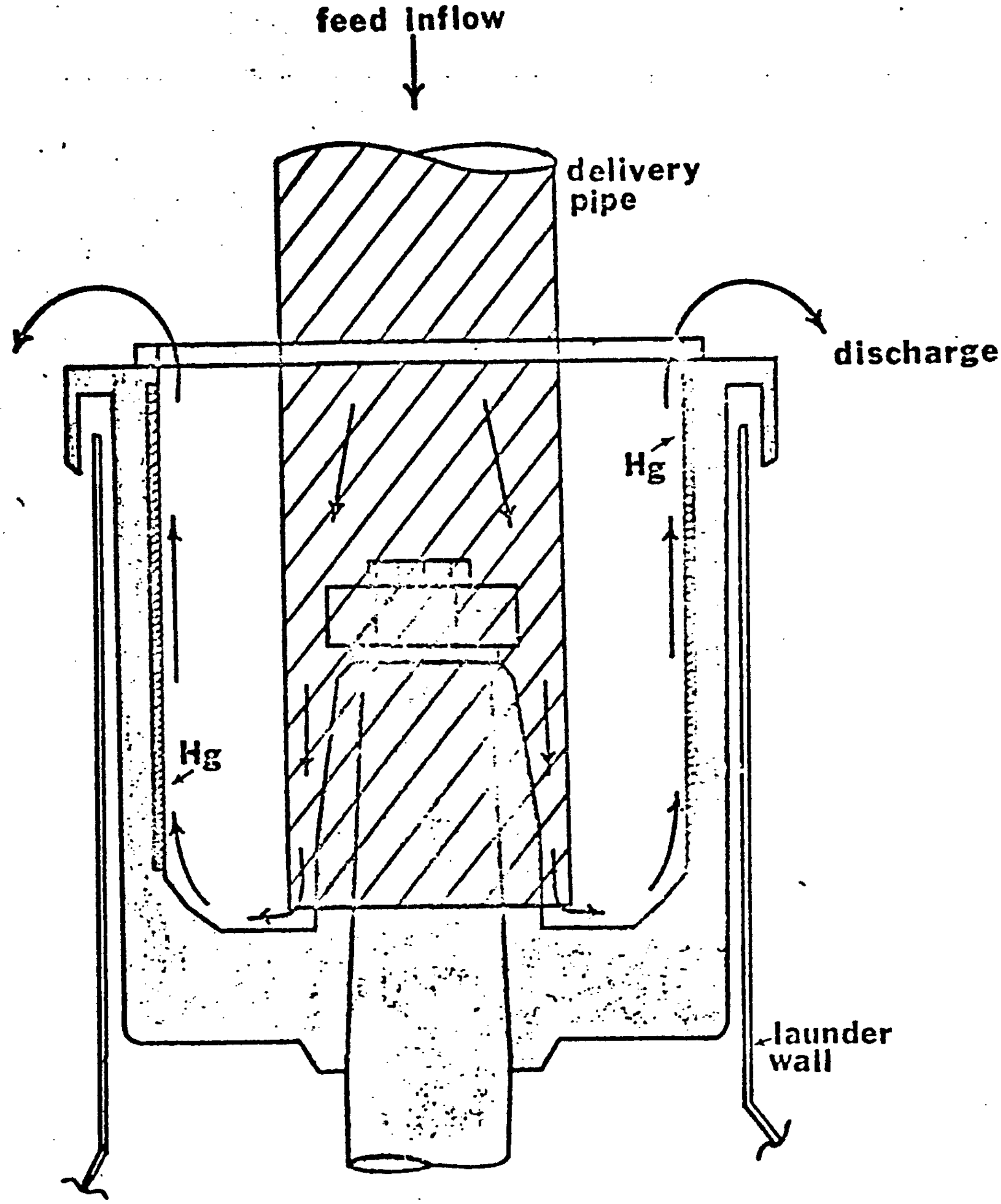
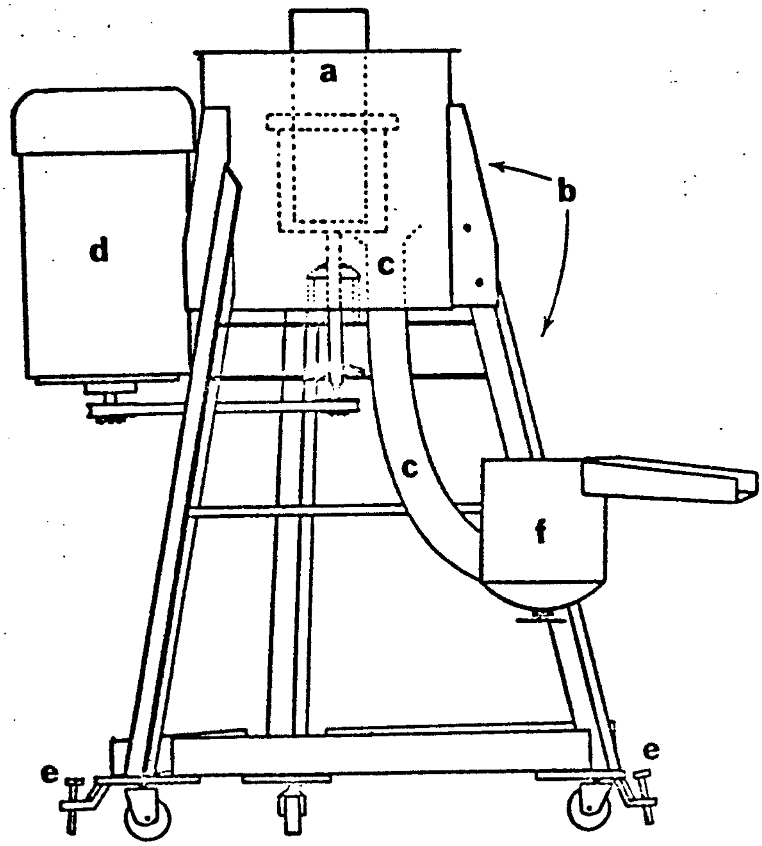


Figure 3. Feed Flow Diagram in Gilkey Amalgamator

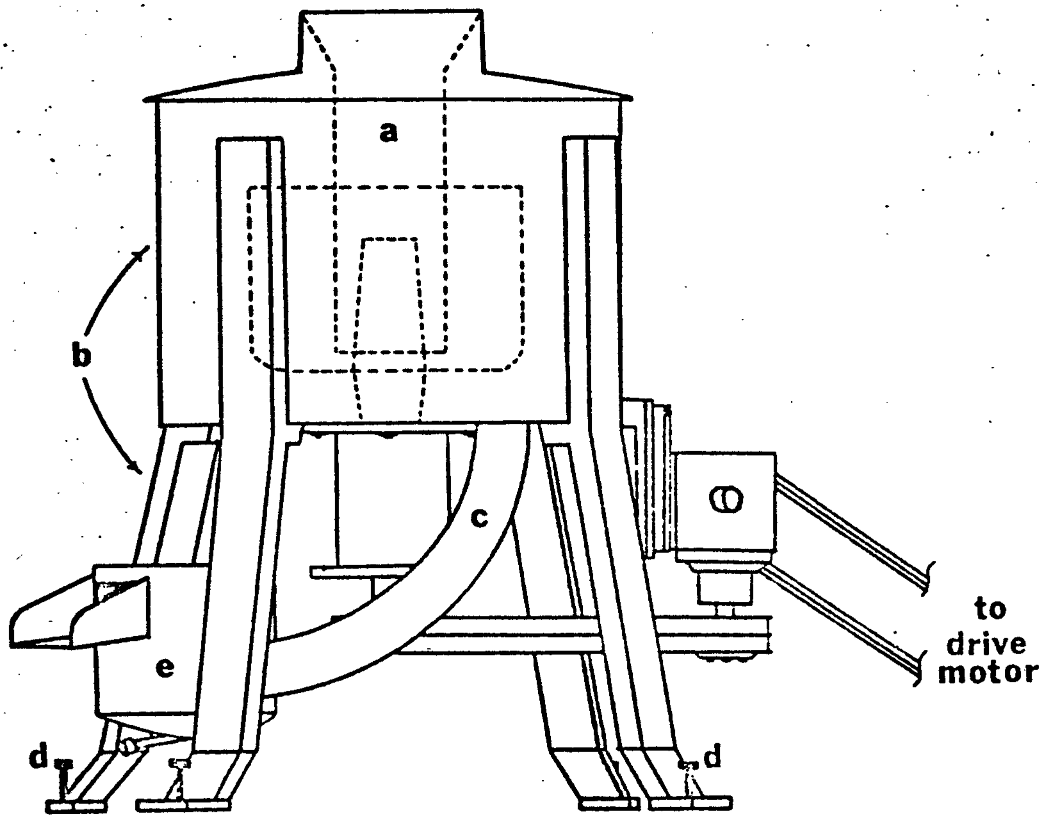
process when the gold particles contact the surface of the mercury sleeve as a result of the positive gold stratification phenomena previously discussed under Theory Of Application.

The remainder of the Gilkey apparatus in effect consists of an assembly designed to serve the bowl in its operation. (See figures 4 and 5) It consists of a feed launder to conduct the feed into the bowl, the support chassis for the bowl, a discharge launder in the chassis to conduct the processed material away from the bowl, a variable speed drive motor used in the experimental work, leveling bolts to maintain the bowl in a plumb attitude, and a mercury trap to recover mercury accidently lost from the bowl.



key a – feed launder
b – support chassis
c – discharge launder
d – drive motor
e – level bolt
f – mercury trap

Figure 4. Gilkey Assembly for 6" Bowl



key a – feed launder
b – support chassis
c – discharge launder
d – level bolt
e – mercury trap

Figure 5. Gilkey Assembly for 12" Bowl

CHAPTER 2

THE TESTING PROGRAM

2.1 Introduction

This segment is a description of the testing program. Included is a general description of the program with respect to basic objectives and methods of approach. In addition there is a brief discussion concerning the basic theory of the testing procedure with the intent of illustrating the parameter significance as may effect test results. This is followed by a description of the scope of this project.

This chapter also contains a description of the testing pilot plant and contains an explanation of the plant's flowchart, essential equipment specifications, and an explanation of the intended mode of operation of the plant.

Specific analytical methods for the various series of tests are found in APPENDIX A in the back of this thesis.

2.2 General Description Of The Testing Program

The testing program was designed to evaluate the centrifugal amalgamator's operation performance under conditions which would result in maximum gold recovery efficiency without mercury loss. The program was therefore divided into two basic phases. The first phase involved testing to discover the parameter range in which the bowl could operate without mercury loss. The second phase involved testing to discover the amalgamator's gold recovery ability while operating in the safe mercury loss range of parameters as determined in phase one.

2.3 Theory Of Testing Procedure

To carry out the above objective, it was deemed necessary to identify all of the possible parameters which might have an effect on the performance characteristics of the amalgamator, and also to discover the desirable operation range of those pertinent parameters.

To satisfy the first of these requirements it was determined that, based on previous experiences with analogous slurry handling and mineral processing devices, the parameters compiled in Table 1 might significantly affect the centrifugal amalgamator's performance.

To satisfy the second requirement, as concerns discovering the effect and safe operating range of the various parameters, a testing plant was constructed of such a nature that control of all important parameters could be maintained. In this way series of tests were conducted in which all but a single parameter per suite of tests were held constant. The single parameter was varied through a practical range during mercury loss or gold recovery testing, and the parameter's effect on the amalgamator's performance was duly noted for the various points in the range. In an attempt to ultimately have every parameter in an optimum range,

Table 1

Possible Effect of Various Feed and Operating Parameters on the Performance Characteristics of the Centrifugal Amalgamator

Significant Feed or Operating Parameter	Amalgamator's Performance: Mercury Loss from the Bowl	Amalgamator's Performance: Per Cent Gold Recovery
feed input:		
Feed type	X	X
Particle size and sizing characteristics	X	X
Specific gravity feed solids	X	X
Pulp density feed slurry	X	X
Volumetric flowrate	X	X
Hg sickening agents in feed	X	X
mechanical input:		
Operation duration	X	-
Rotation rate of bowl	X	X
Volume Hg charge in bowl	X	X
Startup procedure	X	X
Stopping procedure	X	X
Level attitude of bowl	X	X
gold input:		
Size of gold	-	X
Shape of gold	-	X
Degree of liberation	-	X
Surface coatings on gold	-	X

(X Denotes Possible Effect)

each new parameter test series would have the previously tested parameters placed in their experimentally determined optimum range. The one parameter at a time testing procedure was adopted for this project because the impact evaluation of any single parameter on the performance of the amalgamator is relatively easy and is more accurate than dealing with the disguising effects of impacts from several simultaneously changing parameters.

2.4 Scope Of Project

The possible influencing parameters seen in Table 1 are capable of varying through a very large range. Limitations of time and resources available to this project dictated that parameter investigations be confined to some limits. Those limits used were chosen for their estimated ability to meet both the practical limitation criteria and to most closely approximate the real conditions found in the field.

The scope of this project was therefore limited to the stipulations in Table 2 as concerns mercury loss tests and per cent gold recovery tests on the Gilkey centrifugal amalgamators.

More comprehensive testing increments for existing tests, or other types of tests than so listed, and routine mechanical alteration investigations of the amalgamators is beyond the scope of this investigation at the present time.

Table 2

Scope Limitations of Test Parameters

PARAMETER:	STIPULATIONS:
Feed Type	Limited to alluvial sand-water slurries.
Solid Particle Size	Screened to minus 8 mesh Tyler, size distribution held constant. (Screen analysis in APPENDIX A)
Specific Gravity Feed	Held near the 2.67 range. (Standard for silica sands)
Pulp Density Feed	Variable, 10%, 20%, and 30% solids by weight.
Feed Flowrate	Variable, 6 and 9 G.P.M. of pulp for 6" bowl, 10, 20, 30, and 40 G.P.M. of pulp for 12" bowl.
Gold Size	Variable, 20/28 mesh, 28/35 mesh, and 35/48 mesh.
Gold Nature	Decidedly flat and flaky placer particles clean and liberated.
Operation Duration	Standard one half and one hour durations, with several multiple hour investigations.
Horizontal Attitude Of Bowl	Plumb level, with several special out of level tests.
Mercury Charge In Bowl	Variable, from 64 to 92% of the bowl's channel volume.
Rotation Rate Of Bowls	Variable, from 500 to 2100 ft/min periphery speed, in 100 FPM increments, for the 6" bowl. For the 12" bowl, rotation increments generating centripetal acceleration forces equivalent and corresponding to the forces exerted from the rotation range of the 6" bowl.

2.5 Description Of Test Plant

(Refer to the flowchart in Figure 6, and to the equipment specifications in Table 3 for this discussion. Note that a reference number exists on the flowchart for each device to provide the reader with an easy correlation between the pictorial representations and the written descriptions of the various devices.)

A) Mechanical Nature

Essentially the testing pilot plant consists of; The source tank, item 1, which delivers its contents via gravity underflow to item 3, an electrically driven centrifugal slurry pump. The pump, to achieve the subsequent desired gravity flow, elevates the material into item 4, an elevated surge tank. The feed gravity underflows the surge tank and enters a splitting junction, item 5, composed of a simple pinch valve in a pipe tee. From this primary splitter the desired amount of feed material flows by gravity through steel pipe to item 6, a secondary splitting junction. This secondary splitter is designed to deliver controlled volume feed to either or simultaneously to each of two centrifugal amalgamators. The feed at the splitting junction may either gravity flow into a 6" diameter centrifugal amalgamator, item 7, and then through a small vertical current classifier mercury trap, item 8, or alternately into a 12" diameter centrifugal amalgamator, item 9, and then through its corresponding vertical current mercury trap, item 10. From either mercury trap the slurry flows into a common launder, and passes over a silvered amalgamation plate, item 11, which is situated at the lower end of the launder. From the launder the pulp gravity flows back into the source tank thereby completing its closed circuit path. The excess feed split at item 5 returns by gravity directly to the source tank.

To facilitate material handling, alternate flow paths were provided in the plant by a plumbing system containing pinch valves at strategic points. (The alternate flow paths are depicted in Figure 6 as dashed lines). The most significant alternate flow provides storage for the material from the source tank. The storage tank is item 2. There is also a dry feed storage tank and vibrating feeder combination, item 12, which can deposit feed material into a water mixing launder leading directly into the amalgamators.

B) Mode Of Operation

The plant is designed to continuously re-cycle that feed slurry which was initially placed into the source tank. In this way a stable and constant pulp density slurry feed can be continuously fed to the amalgamators thereby simulating the feeding process of some typical mining machine such as a dredge washing plant. Pulp density variations can be easily achieved in the circulating load by adding or removing water or solids at the source tank during operation. Flow rates of slurry delivered to the amalgamators are controlled by adjusting the splitter valves and/or adjusting the slurry pump's delivery rate. Uniformity of flow is facilitated by the surge tank. Rotation rates of the bowls are controlled by variable speed drive motors.

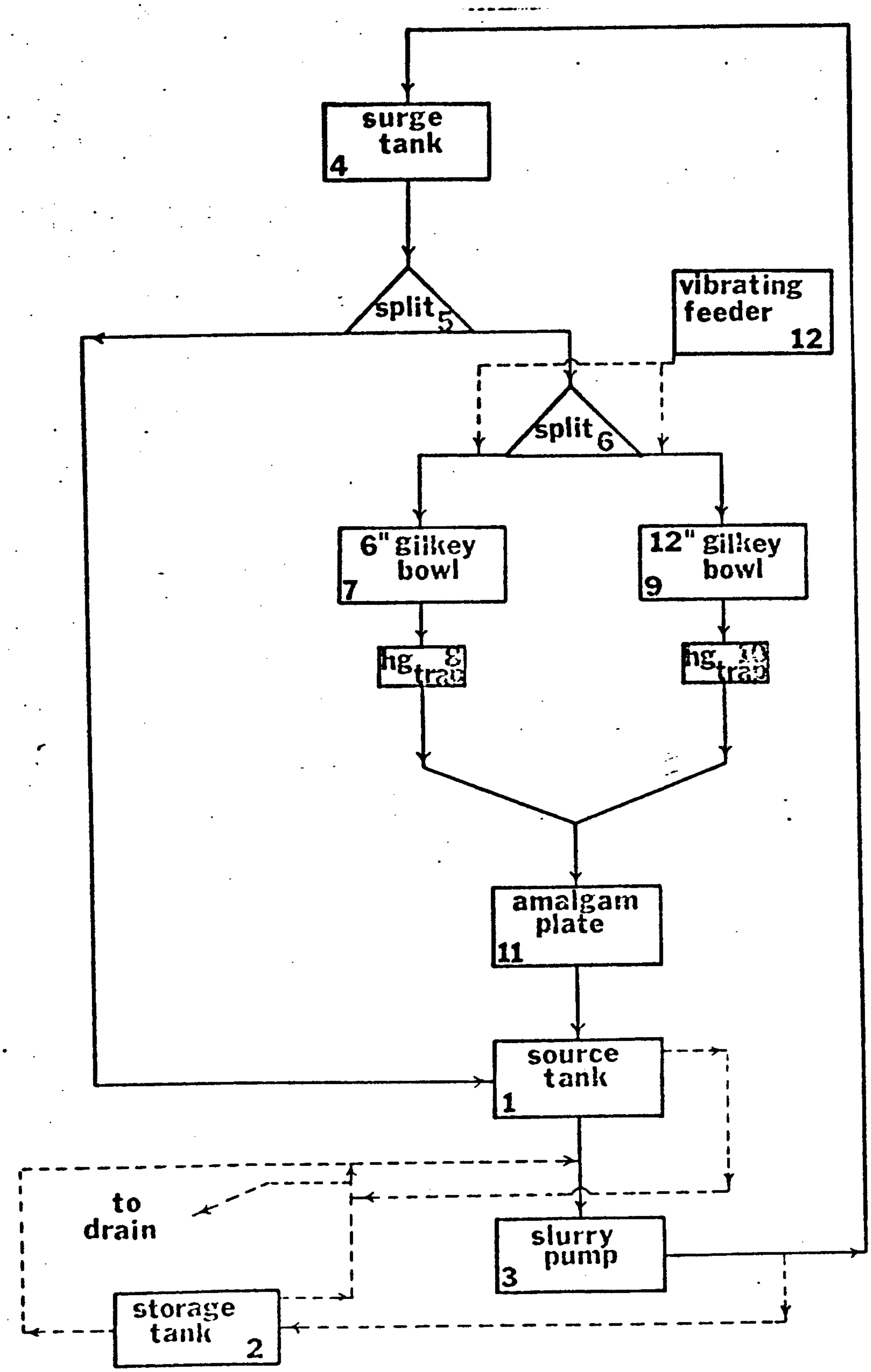


Figure 6. Test Plant Flowchart

Table 3
Equipment Specifications

Item Reference No.	Equipment Name and Size	Number Required	Hp.	Mfg.	Miscellaneous
1	Source tank 80 gallon capacity	1	-	Custom	
2	Storage tank 100 gallon capacity	1	-	Custom	
3	3"x 2" centrifugal slurry pump with electric motor drive	1		Linatex pump	Model M left hand pump/w.model D gland
		1	10	Appleton motor	220/440 V 60 Hz. 3 phase frame 215T
4	Surge tank 20 gallon capacity	1	-	Custom	
5	Primary split valve junction	1	-	Custom	Pinch valve in a pipe tee
6	Secondary split valve junction	1	-	Custom	Two pinch valves in a pipe tee
7	Gilkey Centrifugal Amalgamator, 6" bowl	1	-	Gilkey	
7-a	Drive motor for 6" Gilkey Bowl	1	¼	Century Electric	Browning brand with Browning rheostat drive control

Table 3 (Continued)

Item Reference No.	Equipment Name and Size	Number Required	Hp.	Mfg.	Miscellaneous
8	Vertical current mercury trap	1	-	Gilkey	Part of 6" amalgamator's chassis
9	Gilkey Centrifugal Amalgamator, 12" bowl	1	-	Gilkey	
9-a	Drive motor for 12" Gilkey Bowl	1	3	Reliance	3 phase 1725 rpm 220/440 V 60 Hz. w. Reeves moto drive
10	Vertical current mercury trap	1	-	Gilkey	Part of 12" amalgamator's chassis
11	Amalgamation plate 6"X 12"	1	-	Custom	Silver coated copper plate
12	Vibrating feeder with feed bin	1		Syntron	Model F-152 115 V 2.5 A 60 cycle

For gold recovery testing, gold particles are manually fed into the feed slurry just ahead of the amalgamators. The gold particles are meant to make only one pass through the amalgamators per test. The possibility of recirculating any unrecovered gold particles with the circulating load of slurry is minimized by the effect of the gold stopping nature of the mercury traps and amalgamating plate. The gold recovery test period is held to a minimum duration to further reduce the gold recirculation possibility. In addition, the circulating slurry load is purged of gold particles between tests periodically.

Special tests on particular gold bearing sands can be made with this plant where the dry material feeds from the vibrating feeder, enters a water mixing launder, makes one pass only through an amalgamator, and comes to rest at the bottom of the source tank. The source tank is rigged in this case to act in the capacity of a thickener where the solids are allowed to settle out of suspension, and the clear water is decanted and pumped back to be mixed with more dry feed to repeat the cycle. This arrangement constitutes an open-circuit system.

CHAPTER 3

TEST RESULTS

3.1 Introduction

This chapter is a presentation and discussion of the data obtained from the testing program, and is divided into four sections. The first section contains results which were derived from the testing of the 6" diameter amalgamator. This is subdivided into two parts. Part A presents data and discussion concerning the ability of the 6" bowl to retain mercury when influenced by the parameters of rotation rate of bowl, pulp density of feed, and volumetric flowrate of feed. Part B presents data and discussion reflecting the ability of the 6" bowl to recover gold when influenced primarily by the rotation speed, pulp density, flowrate, and size of gold in feed.

The second section contains test results which were derived from the testing of the 12" amalgamator. As in the previous case, this section is subdivided into two parts, A and B, dealing respectively with the mercury retention and gold recovery abilities of the bowl when influenced by the parameters of rotation rate, pulp density, feed flowrate, and size of gold.

The third section contains a series of miscellaneous test results, and significant observations made during the testing program. Included in the miscellaneous tests are; mercury retention for the 6" bowl during out of level operation, mercury retention for the 12" bowl during startup and stopping sequences, and gold recovery for the 12" bowl from the processing of gold bearing beach sands from Nome Alaska and Yakataga Alaska.

The fourth section is a discussion of data reliability.

3.2 Tests On 6" Amalgamator

A) Mercury Retention

A series of 60 tests were conducted on the 6" bowl to determine the bowl's mercury retention characteristics as influenced by pulp density, rotation rate, and volumetric pulp flow rate. Pulp density ranges were varied at 10, 20, and 30 per cent solids by weight. Pulp flows were varied at 3.5, 6, and 9 gallons per minute. Rotation rate of the bowl, measured as rim speed in feet per minute, varied from 900 to 2400 feet per minute. Non changing parameters common to every test in this series were; one half operation duration, 95 cubic centimeters of mercury charged into the bowl, (66% of the channel volume), and sized silica sand feed. (See APPENDIX A for further information)

Data from this series of tests are presented in Table 4.

Tests having pulp flows of 3.5 gallons per minute at 20 and 30 per cent pulp densities were not able to be made as these conditions would cause a clog of material in a splitter valve in the feeding arrangement to the 6" bowl thereby creating an uneven feed. Additionally, 9 gallons per minute pulp flow was the maximum practical limit for this bowl, as faster flowrates would cause overflow spillages in the discharge launder at the mercury

Table 4

Test Data and Results on Mercury Retention Characteristics for 6" Bowl

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. Loss %
1	900	10.3	3.5	0.00
2	1100	8.8	3.5	0.00
3	1300	11.4	3.5	0.00
4	1500	11.2	3.5	0.00
5	1700	10.0	3.5	0.00
6	1900	11.2	3.5	0.00
7	2100	11.1	3.5	0.00
8	2400	9.0	3.5	0.53
9	900	10.1	6	0.00
10	1100	10.1	6	0.00
11	1300	9.5	6	0.00
12	1500	10.2	6	0.00
13	1700	9.8	6	0.00
14	1900	9.8	6	0.00
15	2100	10.6	6	0.00
16	2400	10.0	6	0.00
17	900	8.9	9	0.00
18	1000	9.1	9	0.00
19	1100	10.1	9	0.00
20	1200	11.8	9	0.00
21	1300	10.5	9	0.00
22	1400	10.1	9	0.00
23	1500	8.6	9	0.00
24	1600	10.9	9	0.00
25	1700	9.7	9	0.00

Table 4 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg Loss %
26	1800	10.4	9	0.00
27	1900	10.4	9	0.00
28	2000	11.1	9	0.00
29	2100	10.5	9	0.00
30	2200	11.0	9	0.00
31	2300	12.1	9	0.00
32	2400	9.3	9	0.00
33	900	22.6	6	0.00
34	1100	21.8	6	0.00
35	1300	22.6	6	0.00
36	1500	18.8	6	0.00
37	1700	19.0	6	0.00
38	1900	23.5	6	0.00
39	2100	19.8	6	0.53
40	2400	20.2	6	0.53
41	900	20.6	9	0.00
42	1100	20.4	9	0.00
43	1300	20.7	9	0.00
44	1500	22.1	9	0.00
45	1700	20.0	9	0.00
46	1900	19.2	9	0.00
47	2100	20.7	9	0.00
48	2400	20.6	9	0.00
49	900	34.2	6	1.05
50	1100	33.0	6	1.05

Table 4 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. Loss %
51	1500	28.7	6	1.05
52	1500	29.4	6	5.79*
53	900	31.8	9	0.00
54	1100	31.5	9	0.00
55	1300	31.5	9	0.00
56	1500	29.3	9	0.00
57	1700	32.1	9	0.53
58	1900	32.3	9	0.53
59	2100	31.6	9	1.05
60	2400	30.4	9	1.68

* Two hour test

trap. Spillages at this point were not able to be contained easily.

Discussion Of Results

Figure 7 is a graphical representation of the data from Table 4. Rim speed is plotted as a function of pulp density for 3.5, 6, and 9 gallon per minute flowrates. Solid graph lines indicate actual data, dashed lines indicate the probable projections for unmeasurable points. From Figure 7 it can be seen that any speed under the respective pulp flow graph line would be a safe speed to operate at the various pulp densities. Speeds above those pulp flow lines, for the particular corresponding pulp densities, would result in a mercury loss.

The test results indicate that as the flow rate decreases, the speed rate must correspondingly decrease in order to prevent mercury loss. In addition, as the pulp density increases with the same pulp flow rate, the speed rate must decrease in order to prevent mercury loss.

Test number 51, with an operation duration of one half hour, lost 1.05% of the input mercury. Test number 52 was a duplication of test 51, except that the operation duration was increased four times to two hours. This resulted in a mercury loss of 5.79% of the input volume. This indicates that when conditions are present for a mercury loss, the amount of loss will be a function of time.

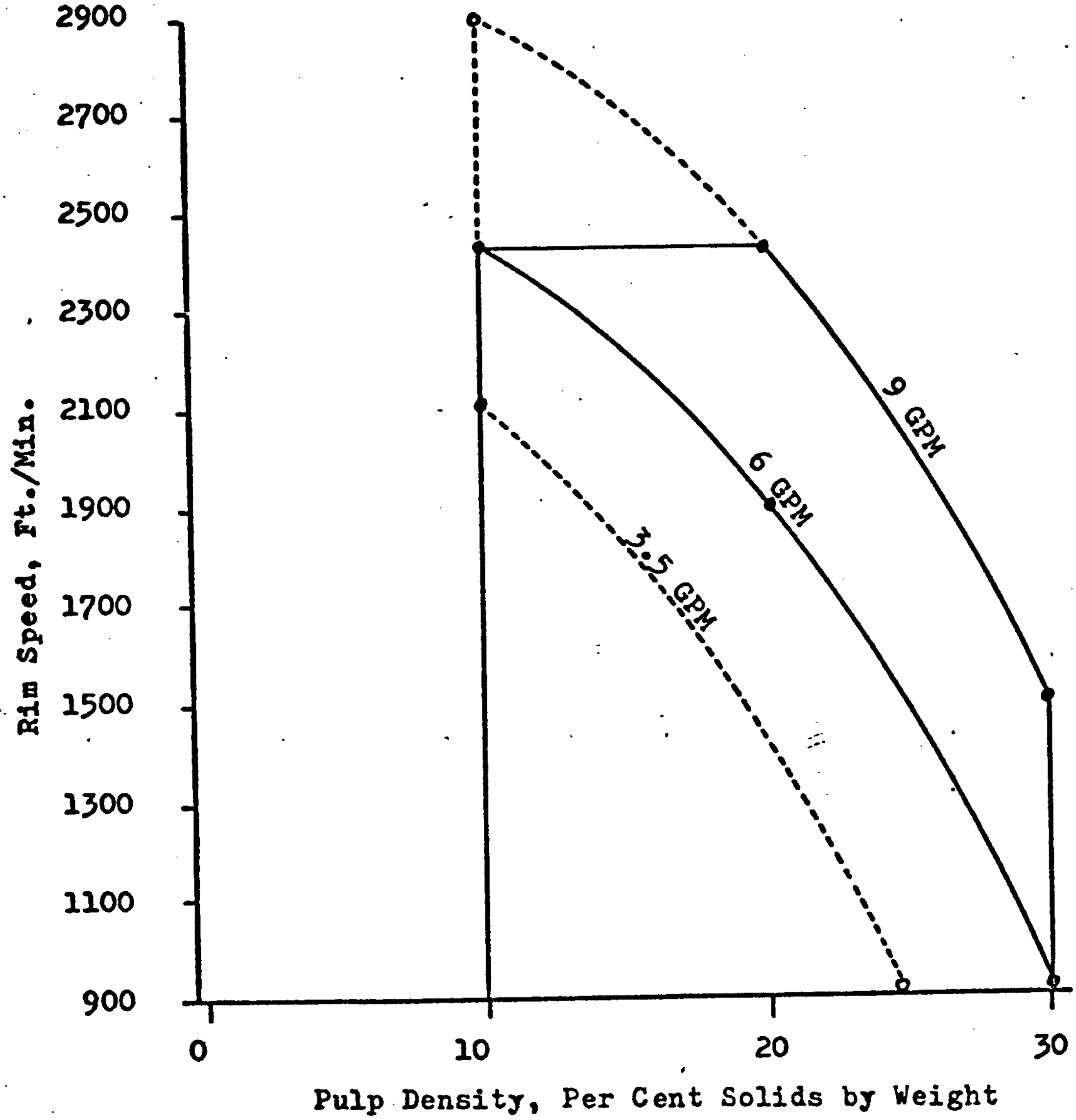


Figure 7. Operating Range of Rim Speeds for Mercury Retention, 6" Bowl

B) Gold Recovery

With parameters having been established which would assure no mercury losses from operations within their limits, gold recovery testing was conducted on the 6" bowl. A series of 84 tests were carried out. The data from this series of tests are presented as Table 5. Essentially this series investigated the effect that the parameters of pulp density, rotation rate, volumetric flow rate, and size of gold had on per cent gold recovery. Pulp densities were tested at 10 and 20 per cent solids by weight, flow rates were tested at 6 and 9 gallons of pulp per minute, and rotation rates varied from 500 to 2100 feet per minute rim speed in 200 foot per minute increments. These speed rates generate forces ranging from 10 to 166 g's. Gold particle sizes varied from 20/28 mesh, 28/35 mesh, and 35/48 mesh batches. All gold was solid, clean, and of a decidedly flat nature.

At the start of this series of tests it was desired to determine the effect of input mercury volume on per cent gold recovery. Accordingly, the first five tests were conducted using 10 per cent pulp density, 9 gallon per minute flow, and 1500 feet per minute rim speed, at mercury volumes of 90, 100, 110, 120, and 130 cubic centimeters. The per cent gold recovery is shown in Table 5, and is plotted in Figure 8. A duplication of the first five tests was carried out in tests 6 to 10 and was also plotted in Figure 8. An examination of these data reveals that gold recovery efficiency is influenced by mercury volume, and that the

Table 5

Test Data and Results on Per Cent Gold Recovery Characteristics for 6 Inch Bowl

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
(20/28 Mesh Gold Particles)								
1	1500	9.2	9	90	0.1151	40	50.2	47.5
2	1500	10.8	9	100	0.1206	40	51.9	50.0
3	1500	10.7	9	110	0.0977	40	62.1	60.0
4	1500	9.9	9	120	0.0974	40	31.2	35.0
5	1500	8.7	9	130	0.1011	40	31.4	35.0
6	1500	10.6	9	90	0.0930	40	52.1	45.0
7	1500	12.1	9	100	0.0871	40	48.1	50.0
8	1500	10.7	9	110	0.1071	40	67.8	72.5
9	1500	10.5	9	120	0.1017	40	45.9	52.5
10	1500	8.8	9	130	0.1112	40	32.7	32.5
11	500	11.0	6	110	0.0788	40	88.6	87.5
12	700	11.2	6	110	0.0890	40	88.8	87.5
13	900	12.0	6	110	0.0929	40	71.2	72.5
14	1100	9.6	6	110	0.0917	40	78.4	82.5
15	1300	10.6	6	110	0.0915	40	62.1	57.5
16	1500	10.0	6	110	0.0980	40	62.9	67.5
17	1700	10.6	6	110	0.1107	40	62.7	62.5
18	1900	11.6	6	110	0.1043	40	45.3	45.0
19	2100	12.0	6	110	0.1009	40	40.8	47.5

Table 5 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
20	700	20.8	6	110	0.0865	40	93.1	92.5
21	900	22.3	6	110	0.1134	40	41.2	42.5
22	1100	19.0	6	110	0.1004	40	58.4	62.5
23	1300	22.5	6	110	0.1125	40	33.8	40.0
24	1500	23.0	6	110	0.1007	40	34.5	32.5
25	1700	23.0	6	110	0.0926	40	38.4	30.0
26	700	11.4	9	110	0.1003	40	97.6	100.0
27	900	10.2	9	110	0.0909	40	113.0	112.5
28	1100	11.5	9	110	0.1143	40	77.2	80.0
29	1300	13.1	9	110	0.1026	40	58.4	52.5
30	1500	9.4	9	110	0.0816	40	44.2	45.0
31	1700	10.4	9	110	0.1061	40	40.3	40.0
32	1900	9.5	9	110	0.1097	40	55.8	57.5
33	2100	9.6	9	110	0.0962	40	41.1	40.0
34	700	20.7	9	110	0.0923	40	103.4	107.5
35	900	19.9	9	110	0.0996	40	25.8	35.8
36	1100	21.6	9	110	0.0887	40	16.5	22.5
37	1300	18.7	9	110	0.1061	40	24.1	30.0
38	1500	19.3	9	110	0.1106	40	25.4	30.0

Table 5 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
39	1700	20.7	9	110	0.0957	40	48.7	52.5
40	1900	22.8	9	110	0.0956	40	53.0	55.0
41	2100	24.0	9	110	0.1102	40	47.3	60.0
(28/35 Mesh Gold Particles)								
42	500	10.6	6	110	0.0355	40	62.4	65.0
43	700	11.3	6	110	0.0368	40	85.3	82.5
44	900	11.6	6	110	0.0370	40	76.9	75.0
45	1300	11.5	6	110	0.0342	40	48.9	42.5
46	1700	11.9	6	110	0.0331	40	81.5	80.0
47	700	23.0	6	110	0.0304	40	107.4	107.5
48	900	22.0	6	110	0.0317	40	70.1	70.0
49	1300	20.7	6	110	0.0266	40	17.1	25.0
50	1700	19.4	6	110	0.0373	40	31.7	30.0
51	700	10.7	9	110	0.0320	40	80.7	62.5
52	900	10.1	9	110	0.0337	40	85.3	82.5
53	1300	12.9	9	110	0.0353	40	67.3	57.5
54	1700	11.8	9	110	0.0355	40	73.0	60.0
55	700	19.3	9	110	0.0345	40	56.4	70.0
56	900	19.0	9	110	0.0371	40	75.2	97.5

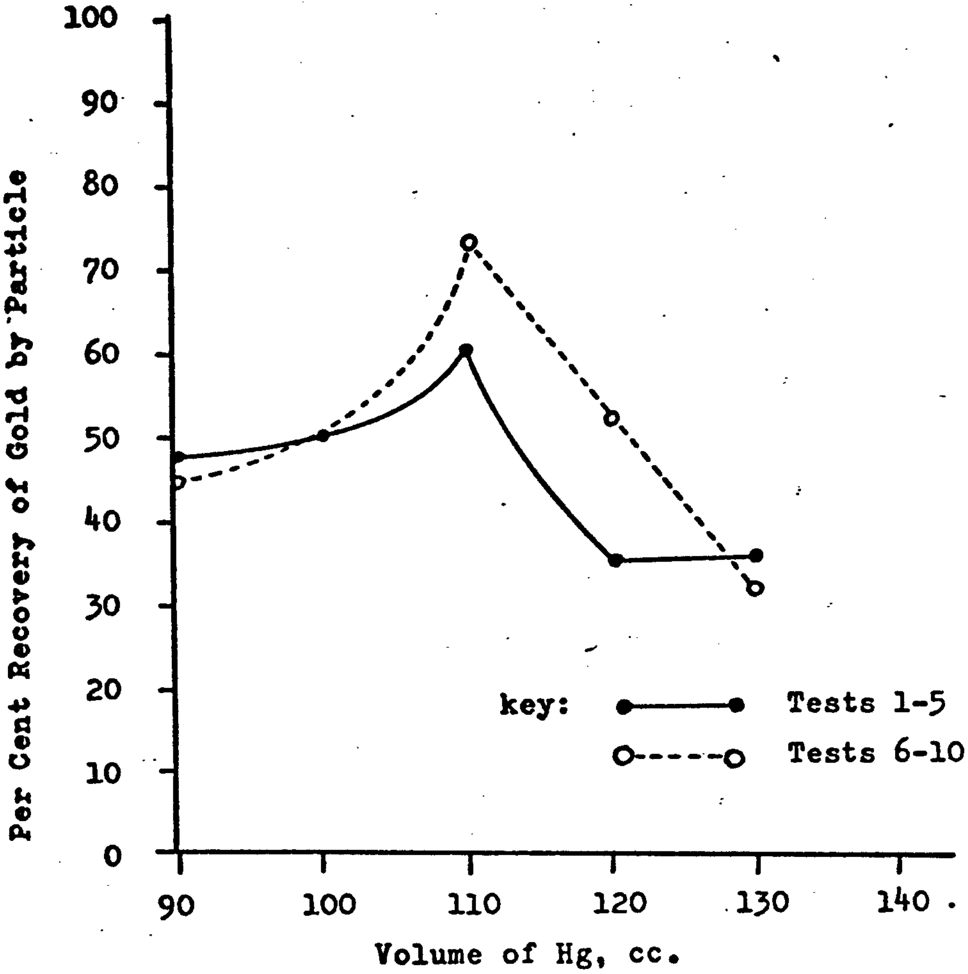
Table 5 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
57	1300	21.8	9	110	0.0342	40	20.5	22.5
58	1700	21.1	9	110	0.0279	40	13.1	15.0
(35/48 Mesh Gold Particles)								
59	500	9.0	6	110	0.0182	40	44.0	90.0
60	700	9.0	6	110	0.0164	40	-**	80.0
61	900	8.0	6	110	0.0167	40	-**	82.5
62	1100	12.0	6	110	0.0178	40	-**	40.0
63	1300	12.0	6	110	0.0166	40	-**	52.5
64	1500	11.8	6	110	0.0162	40	-**	60.0
65	1700	10.5	6	110	0.0168	40	-**	37.5
66	700	20.5	6	110	0.0180	40	70.4	87.5
67	900	20.5	6	110	0.0200	40	81.7	85.0
68	1100	23.0	6	110	0.0204	40	56.6	56.9
69	1300	21.6	6	110	0.0176	40	30.9	40.0
70	1500	23.5	6	110	0.0165	40	53.3	60.0
71	1700	23.6	6	110	0.0157	40	34.9	47.5
72	1900	21.7	6	110	0.0175	40	47.3	52.5
73	700	9.9	9	110	0.0176	40	-**	72.5
74	900	11.2	9	110	0.0159	40	-**	67.5

Table 5 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
75	1100	11.2	9	110	0.0173	40	-**	80.0
76	1300	10.2	9	110	0.0188	40	-**	35.0
77	1500	10.3	9	110	0.0171	40	-**	65.0
78	1700	10.3	9	110	0.0153	40	-**	50.0
79	700	19.8	9	110	0.0220	40	100.0	77.5
80	900	20.7	9	110	0.0254	40	76.0	117.5
81	1100	22.4	9	110	0.0220	40	27.7	30.0
82	1300	20.5	9	110	0.0205	40	57.6	70.0
83	1500	20.0	9	110	0.0254	40	15.4	25.0
84	1700	21.4	9	110	0.0220	40	27.7	57.5

-** % Gold Recovery by Particles is more accurate due to control difficulty of dissolved gold in weighout. % Recovery by Weight was deleted in cases of apparent low accuracy.



Constant Parameters: Rim Speed, 1500 Ft./Min.
9 G.P.M. Pulp Flow
10% Pulp Density by Wt.

Figure 8. Effect of Mercury Volume on Gold Recovery

efficiency falls off significantly above the 75% volume. Accordingly, all subsequent tests in this gold recovery series were conducted with 110 cubic centimeters or 76% of the total mercury volume.

Other conditions common to all of these tests were; 4 to 5 minute operation duration to minimize gold recirculation, and sized silica sand material for solid feed. (See APPENDIX A for more information) In each test, 40 particles of gold were selected, weighed, and salted into the clean feed slurry in order to determine per cent gold recovery by both particle count and weight recovery.

Discussion Of Results

The test results from Table 5 are graphically depicted in Figures 9 to 17.

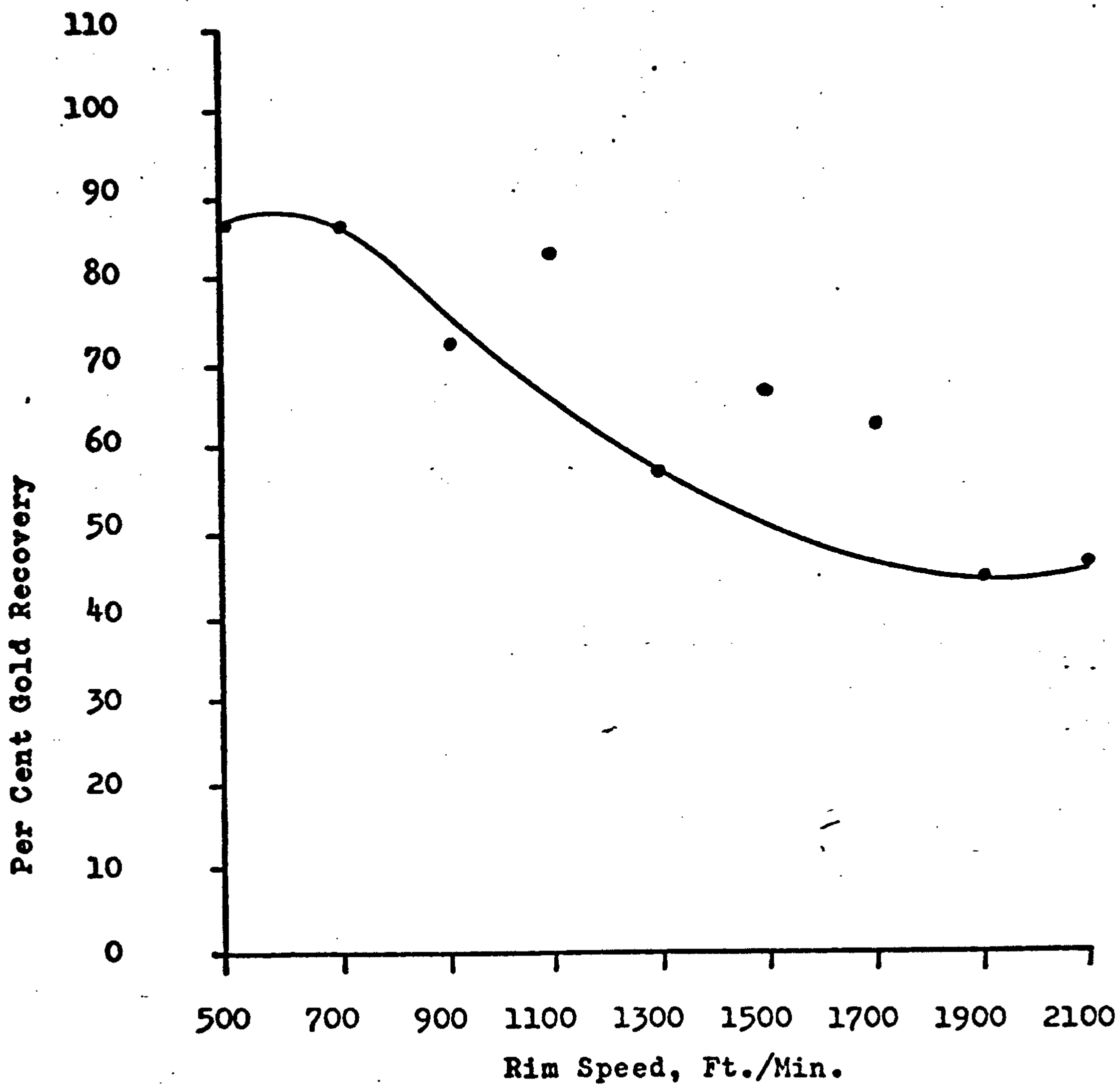
A representation of tests 11 to 19 are found in Figures 9 and 10 which show gold recovery by particle and weight respectively. This group of tests show the effect of rim speed on gold recovery using 6 gallons per minute capacity, at 10 per cent pulp density, and with 20/28 mesh gold particles. It can be seen from Figures 9 and 10 that gold recovery falls off as rim speed increases. The data indicates that an operating range of less than 500 feet per minute rim speed would be desirable. The lower data points on these plots were graphed with the rationalization that some recirculation of gold particles in the closed circuit setup was responsible for the higher recovery results.

Tests 20 to 25 are shown graphically in Figures 11 and 12. Here the effects of rim speed on gold recovery is examined using 6 gallon per minute capacity, at 20 per cent pulp density, and with 20/28 mesh gold particles. These plots similarly show gold recovery falling off with increasing rim speed. Here however the fall off is more pronounced. The significant indication is the fact that rim speeds in excess of 700 feet per minute seem impractical.

Tests 26 to 33, conducted at 9 gallons per minute, 10 per cent pulp density, and with 20/28 mesh gold particles, are shown in Figures 13 and 14. Again the gold recovery efficiency can be seen to fall off sharply at rim speeds above 700 feet per minute. It is assumed that the high readings at 900 and 1900 feet per minute are again due to recovery of gold particles in the recirculating pulp.

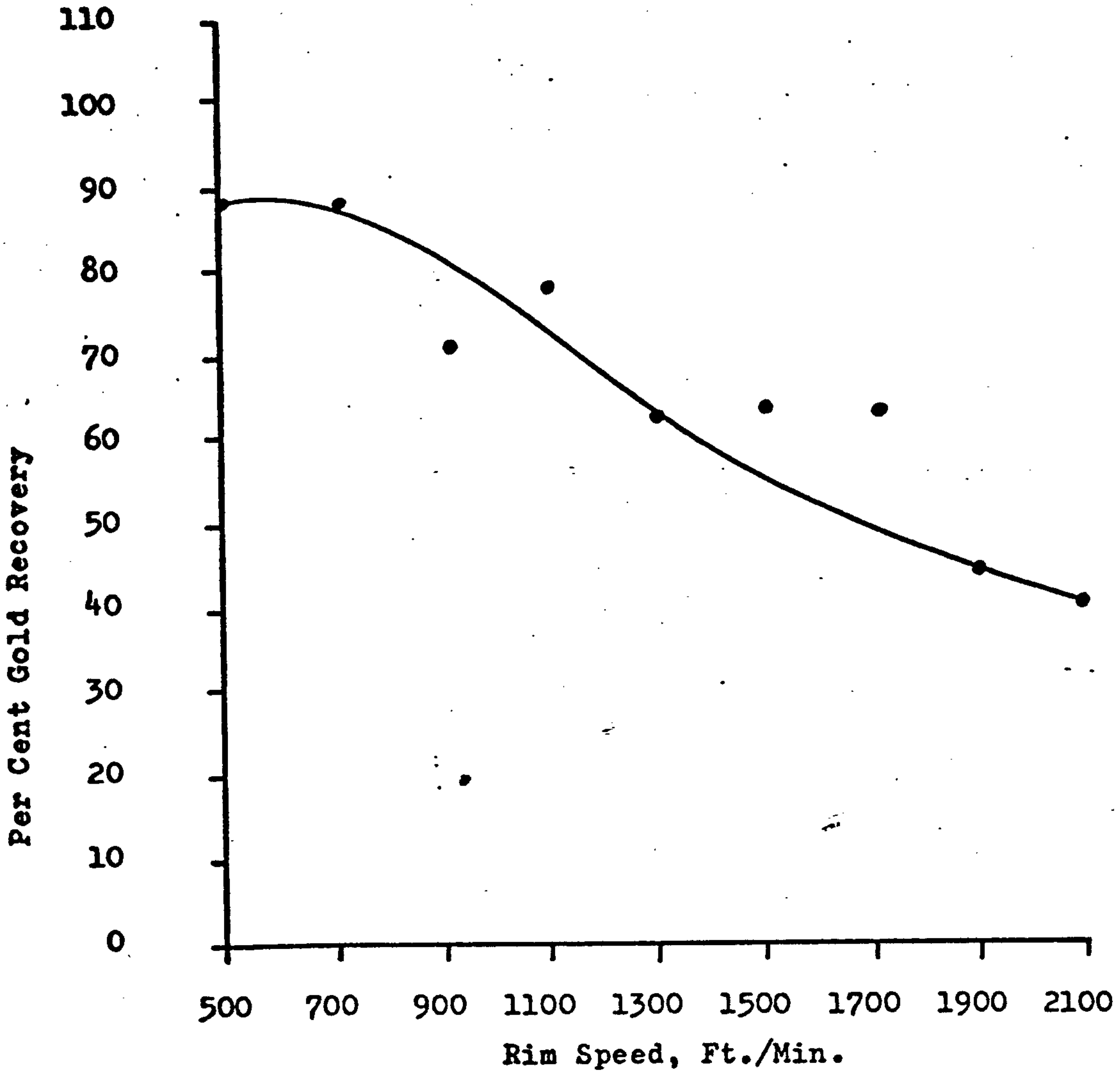
Tests 34 to 41 are represented in Figures 15 and 16. This group of tests was carried out with 9 gallon per minute flow, 20 per cent pulp density, and with 20/28 mesh gold particles. The gold recovery efficiency drop can still be seen above rim speeds of 700 feet per minute. The uptrend of these plots at the higher rim speeds is thought to be a consequence of reclaiming lost gold particles.

The remaining tests 42 to 84 are essentially a repeat of the four series of tests which were just discussed except that, in this case, infed gold particle size was changed to examine the



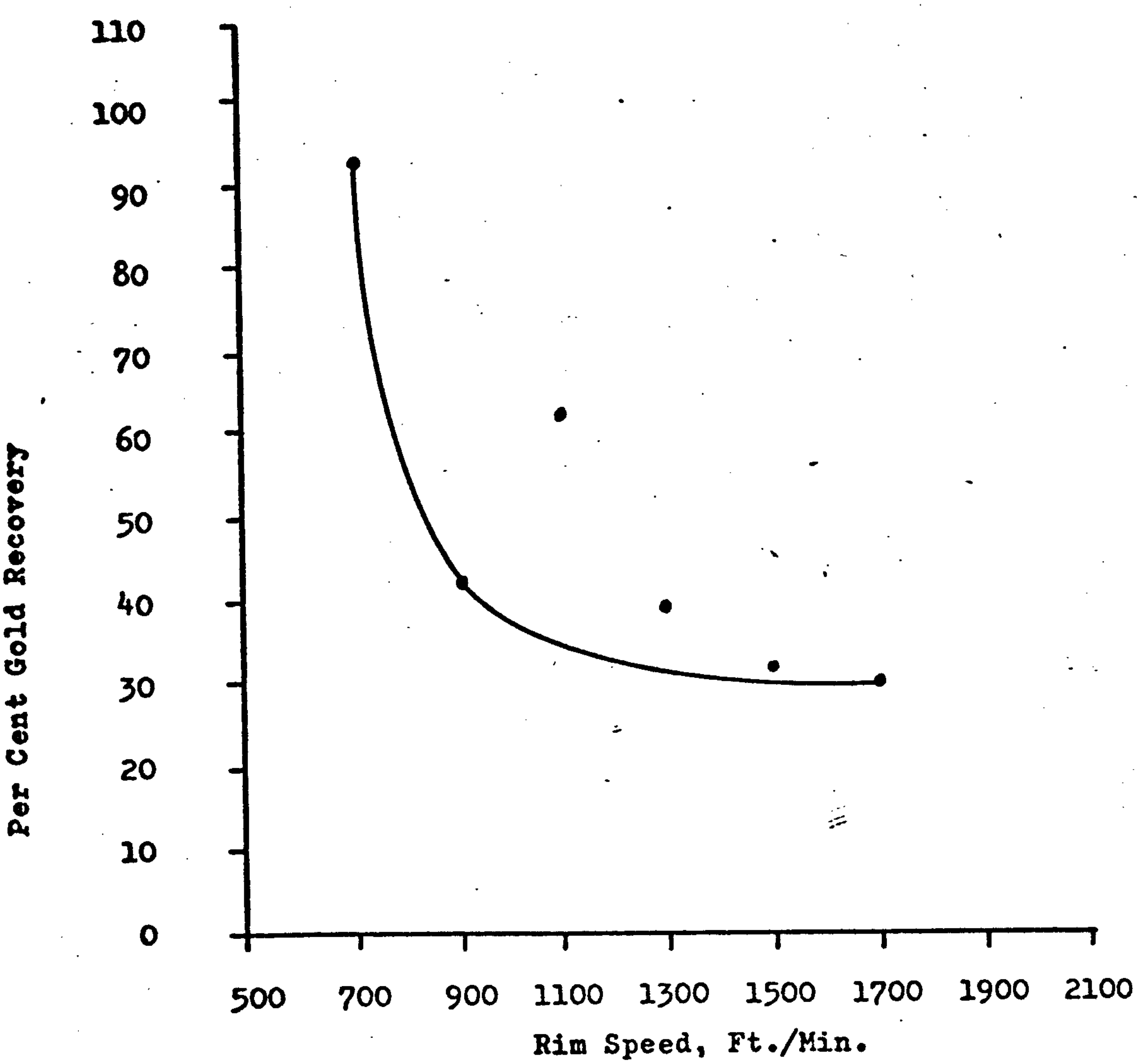
Conditions: Hg Volume 110 cc.
6 G.P.M. Pulp Flow
10% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 9. Gold Recovery by Particle



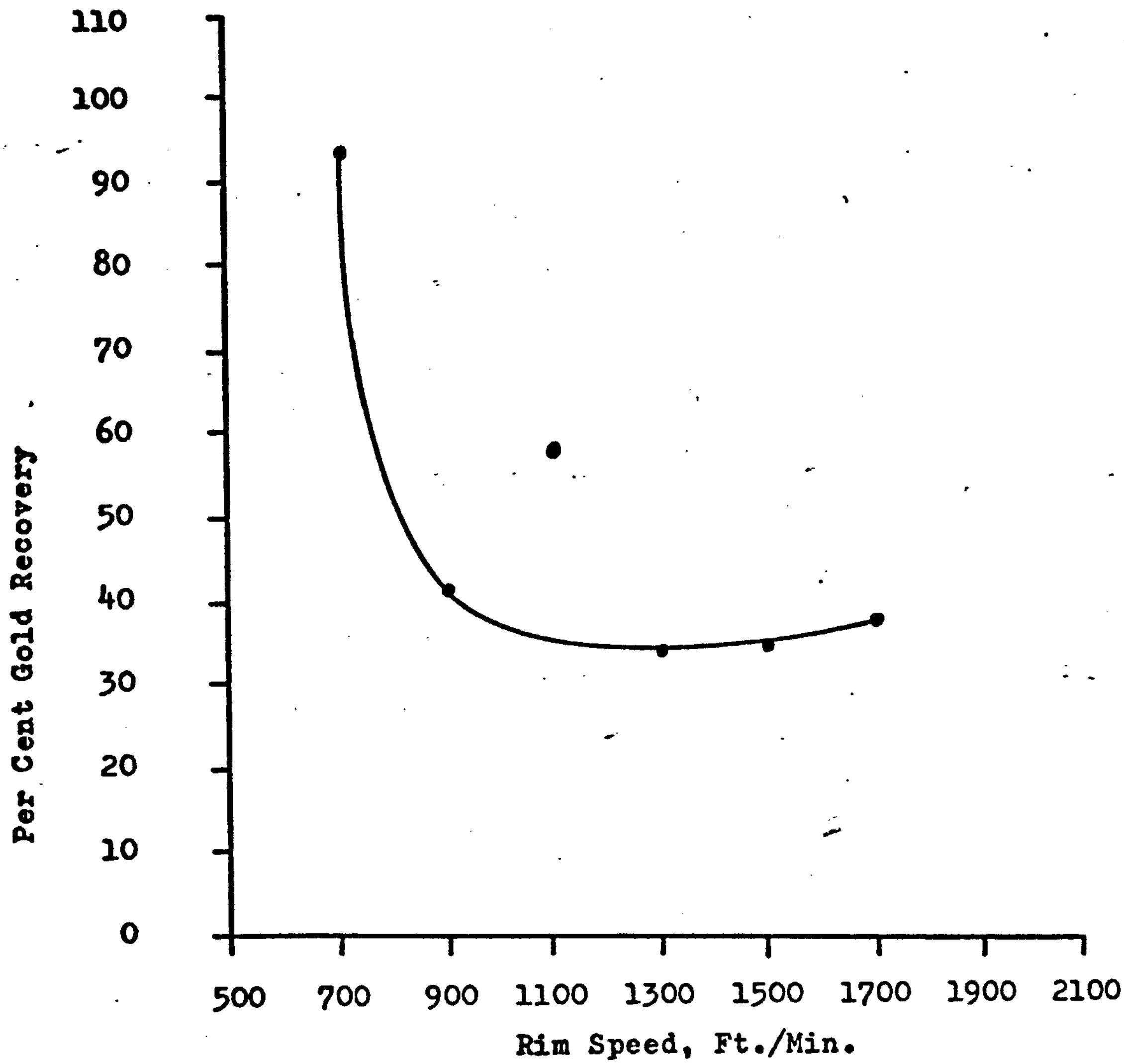
Conditions: Hg Volume 110 cc.
6 G.P.M. Pulp Flow
10% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 10. Gold Recovery by Weight



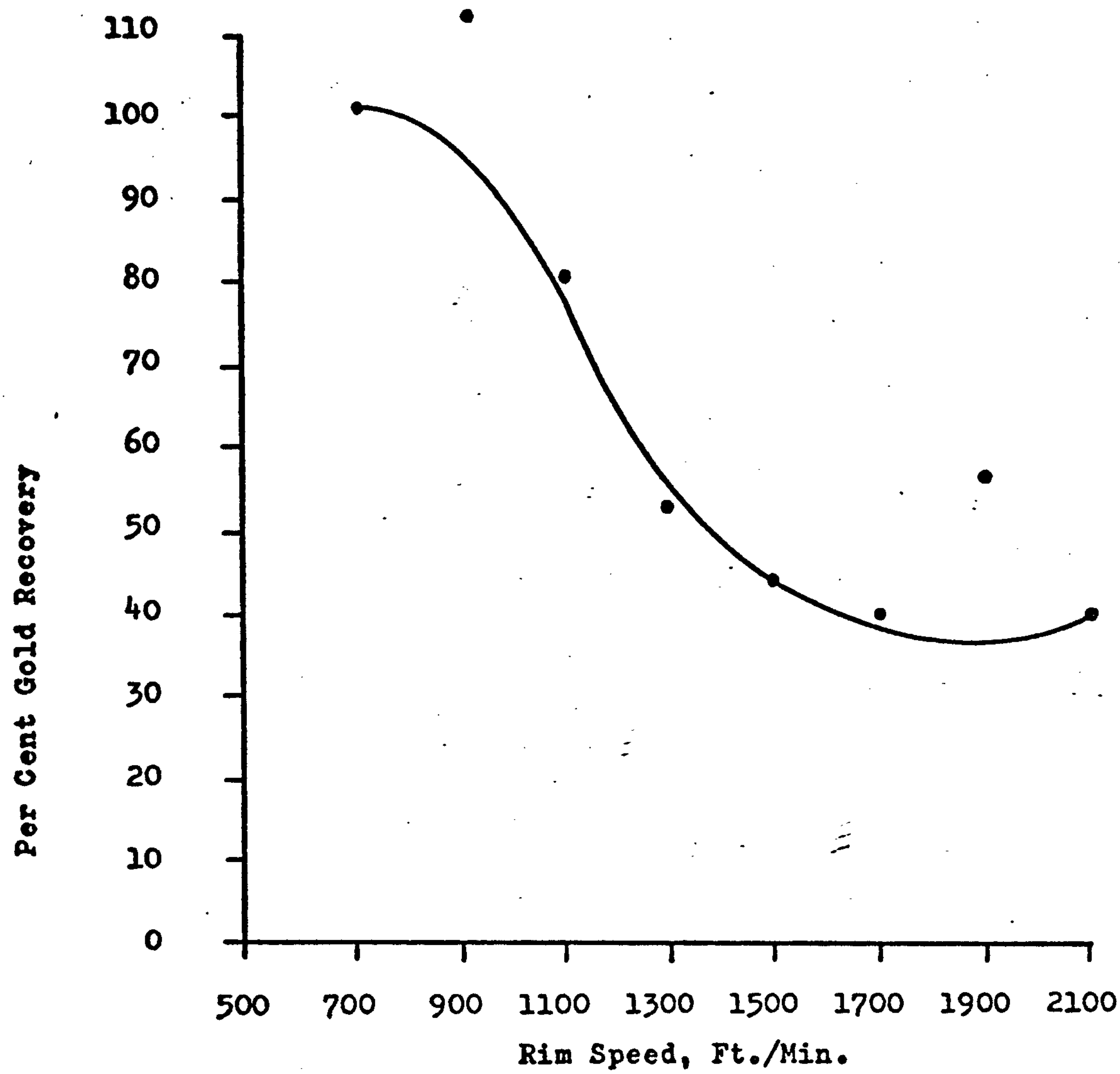
Conditions: Hg Volume 110 cc.
6 G.P.M. Pulp Flow
20% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 11. Gold Recovery by Particle



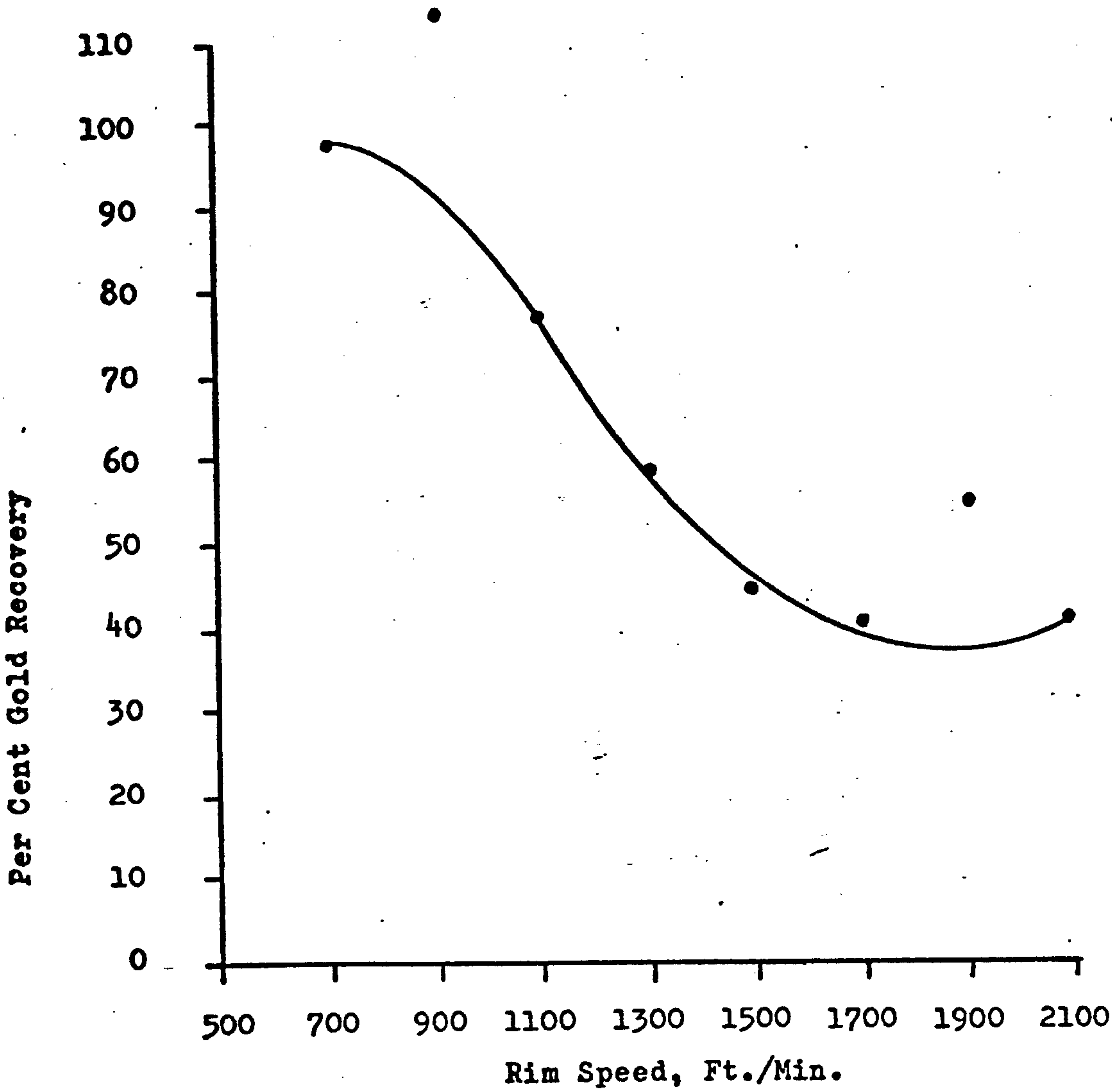
Conditions: Hg Volume 110 cc.
6 G.P.M. Pulp Flow
20% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 12. Gold Recovery by Weight



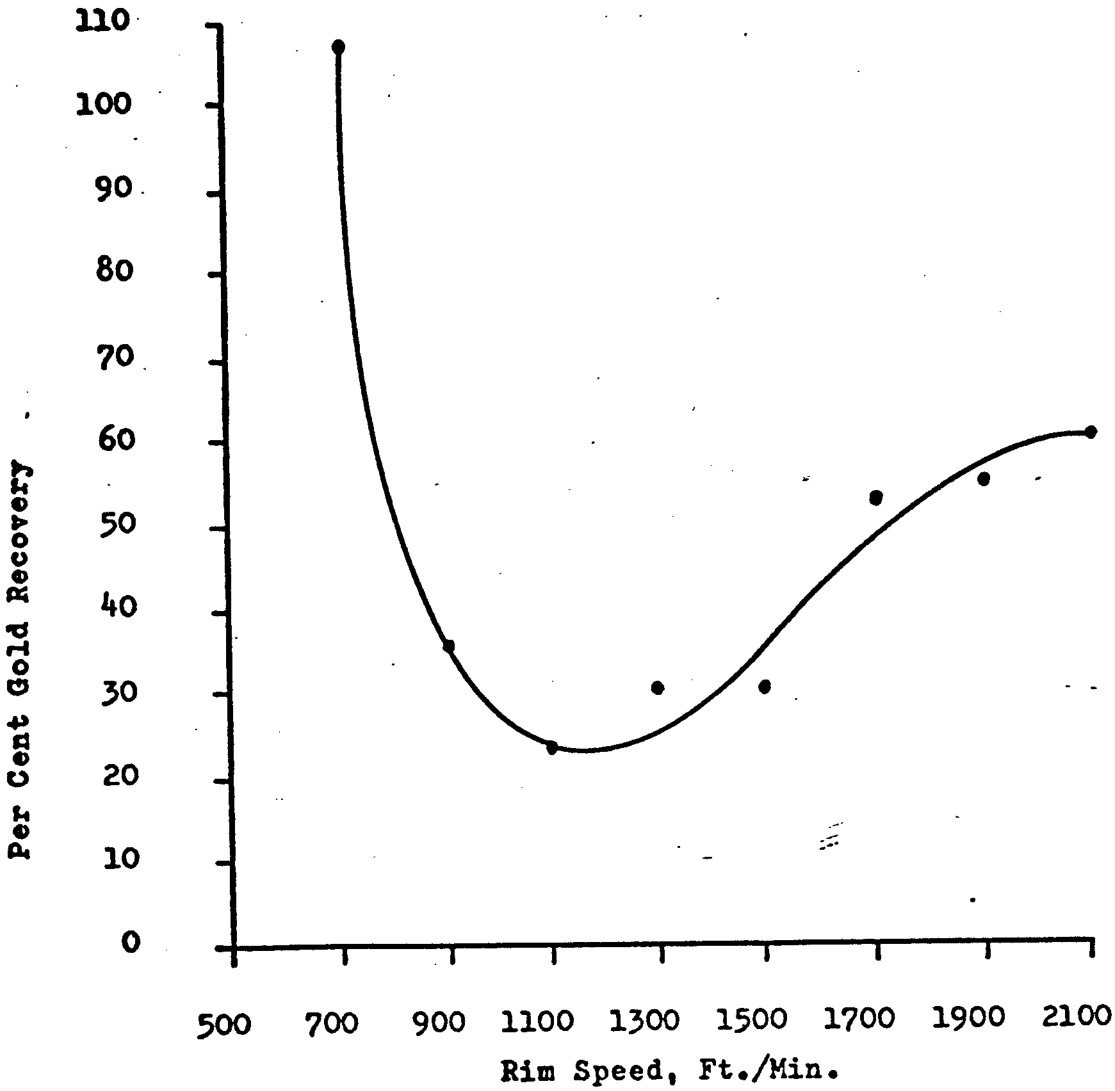
Conditions: Hg Volume 110 cc.
9 G.P.M. Pulp Flow
10% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 13. Gold Recovery by Particle



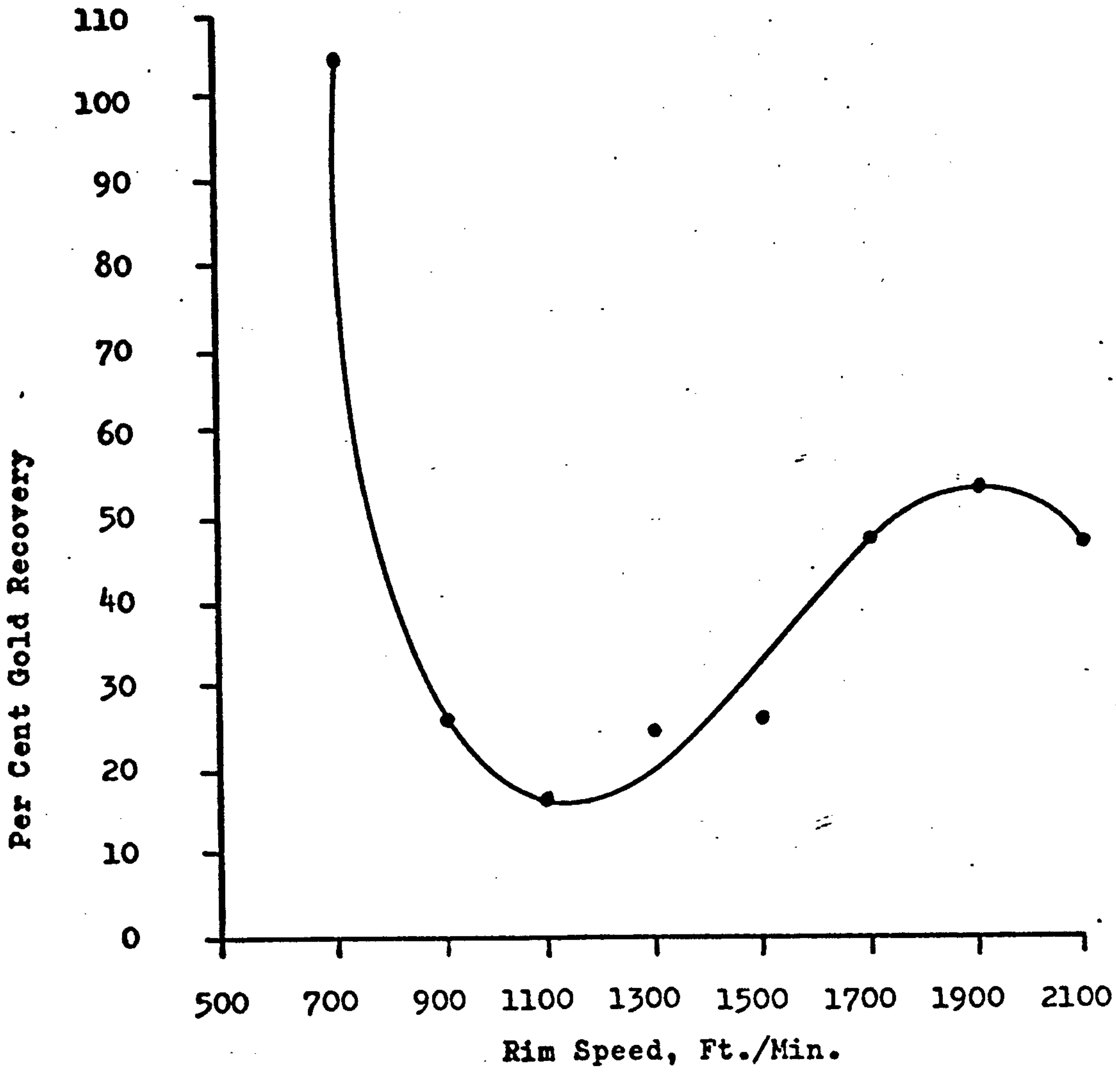
Conditions: Hg Volume 110 cc.
9 G.P.M. Pulp Flow
10% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 14. Gold Recovery by Weight



Conditions: Hg Volume 110 cc.
9 G.P.M. Pulp Flow
20% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 15. Gold Recovery by Particle



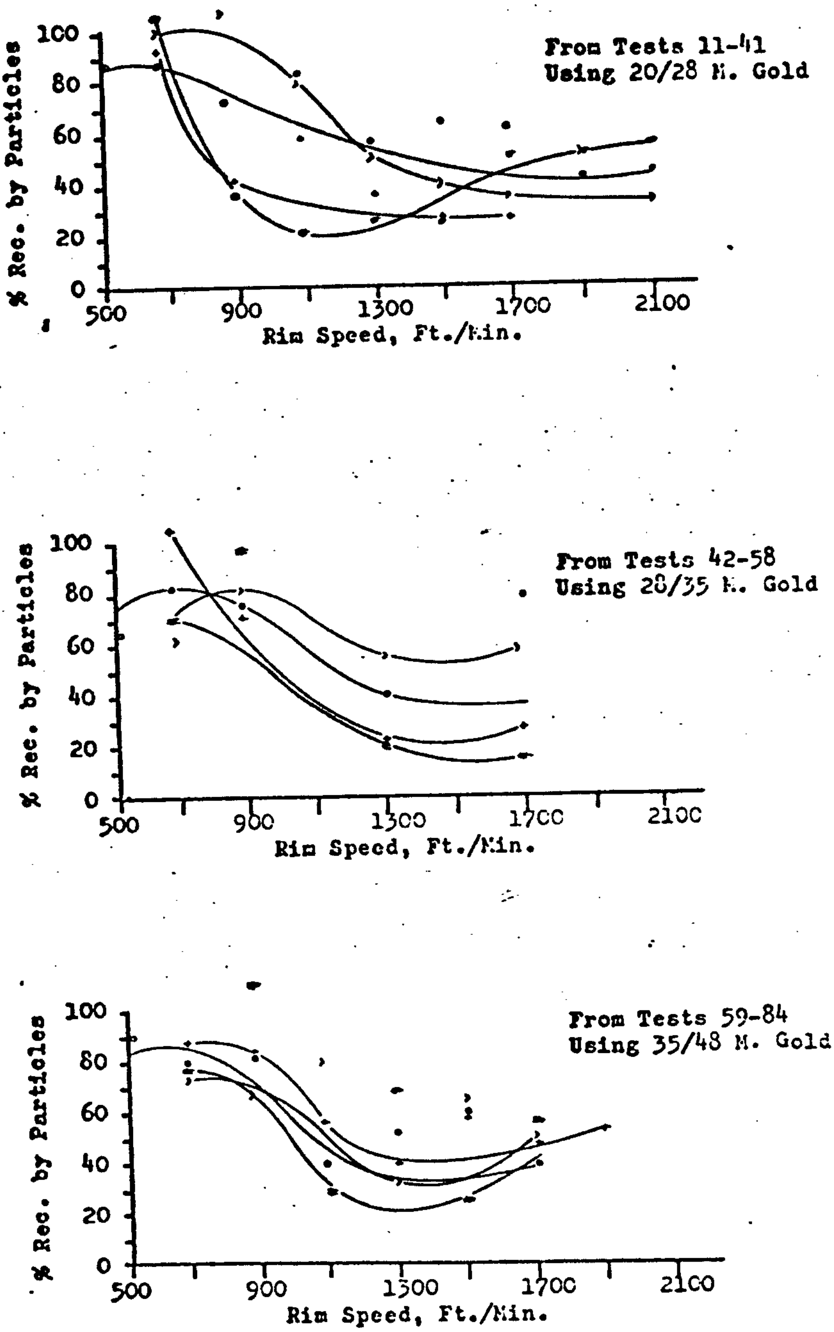
Conditions: Hg Volume 110 cc.
9 G.P.M. Pulp Flow
20% Pulp Density by Wt.
20/28 Mesh Gold Particles

Figure 16. Gold Recovery by Weight

effect that this had on per cent gold recovery. Tests 42 to 58 used 28/35 mesh gold. Tests 59 to 84 used 35/48 mesh gold.

The effect that gold particle size has on per cent gold recovery is illustrated in Figure 17. The four per cent gold recovery by particle plots from Figures 9, 11, 13, and 15 are plotted on a common co-ordinate system at the top of Figure 17. This represents the recovery trend using 20/28 mesh gold. The center co-ordinate system in Figure 17 contains the graphical representations from tests 42 to 58 concerning per cent gold recovery by particles for each combination of parameter group tests contained therein. This represents data from 28/35 mesh gold. The bottom co-ordinate system in Figure 17 contains the graphical representations from tests 59 to 84 for each of the parameter combination groups contained in that series. This represents data using 35/48 mesh gold particles, and again the data points were obtained from the recovery by particle column in Table 5. In each of the three co-ordinate systems, graph lines are labeled "°", representing data using 6 gallon per minute flow at 10 per cent pulp density parameters, "+", using 6 gallon per minute flow at 20 per cent pulp density ranges, ">", using 9 gallon per minute flow at 10 per cent pulp densities, and "#", for 9 gallon per minute flow at 20 per cent pulp density ranges.

This rather compact method of illustration points out two significant facts. The first is that the gold particle sizes used here do not significantly effect the gold recovery trends set by the parameters of rim speed, capacity, and pulp density.



Legend: • 6 GPH, 10% P.D. > 9 GPH, 10% P.D.
+ 6 GPM, 20% P.D. # 9 GPM, 20% P.D.

Figure 17. Common Plots of Curve Families to Illustrate Gold Size Influence on Per Cent Gold Recovery

The second fact is that the trend similarities of all three plot families in Figure 17 suggest a fair degree of reproducability in the testing.

3.3 Tests On 12" Amalgamator

A) Mercury Retention

The mercury retention testing procedure, as applied to the 6" bowl, was similarly applied to the 12" bowl in that the effects of pulp density, rim speed, and volumetric flowrate on the bowl's mercury loss characteristics were again investigated. However some of the input parameter magnitudes were enlarged to accommodate the expected capacity of this larger device. This test series was therefore conducted with the following input parameters; pulp density varied at 10, 20, and 30 per cent solids by weight, flow varied at 20, 30, and 40 gallons of pulp per minute, and rim speed varied from 1500 to 2200 feet per minute in 100 feet per minute increments. Other parameters constant in all tests for this series were; 250 cubic centimeter charge of mercury in the 400 cubic centimeter channel volume of the 12" bowl, one hour operation duration per test, and sized silica sand material for solid component of feed slurry. (See APPENDIX A for more information).

Using this set of input parameters 68 mercury retention tests, as represented in Table 6, were conducted. In addition, three tests were conducted to note the effect on mercury reten-

Table 6

Test Data and Results on Mercury Retention Characteristics for 12" Bowl

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. Loss %
1	1500	9.4	20	0.00
2	1600	10.3	20	0.00
3	1700	10.3	20	0.00
4	1800	9.5	20	0.00
5	1900	9.5	20	0.80
6	1500	13.8	30	0.00
7	1600	12.6	30	0.00
8	1700	12.7	30	0.00
9	1800	12.4	30	0.20
10	1900	12.4	30	0.00
11	2000	10.0	30	0.00
12	2100	10.4	30	2.40
13	2100	9.8	30	0.80
14	2200	11.2	30	0.10
15	1500	13.2	40	0.00
16	1600	12.5	40	0.00
17	1700	9.5	40	0.00
18	1800	9.8	40	0.00
19	1900	11.7	40	0.80
20	2100	10.6	40	0.10
21	1500	18.2	20	0.00
22	1600	22.2	20	0.00
23	1700	19.2	20	0.00
24	1800	22.5	20	0.00
25	1900	21.6	20	0.00
26	2000	20.5	20	0.00

Table 6 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. Loss %
27	2100	22.9	20	0.00
28	2200	20.2	20	0.00
29	1500	19.0	30	0.00
30	1600	20.2	30	0.00
31	1700	20.5	30	0.00
32	1800	19.5	30	0.00
33	1900	20.2	30	0.00
34	2000	19.7	30	0.00
35	2100	18.6	30	0.00
36	2200	20.3	30	0.00
37	1500	19.3	40	0.00
38	1600	18.9	40	0.00
39	1700	21.4	40	0.00
40	1800	20.8	40	0.00
41	1900	19.9	40	0.00
42	2000	22.1	40	0.00
43	2100	20.3	40	0.00
44	2200	19.7	40	0.00
45	1500	31.4	20	0.20
46	1600	31.3	20	0.00
47	1700	30.2	20	0.00
48	1800	30.5	20	0.00
49	1900	32.9	20	0.00
50	2000	31.3	20	0.00
51	2100	32.0	20	0.00
52	2200	29.9	20	0.00
53	1500	31.3	30	0.20
54	1600	30.2	30	0.00

Table 6 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. Loss %
55	1700	28.3	30	0.00
56	1800	28.2	30	0.00
57	1900	32.3	30	0.00
58	2000	29.7	30	0.00
59	2100	29.3	30	0.00
60	2200	29.0	30	0.40
61	1500	33.6	40	0.00
62	1600	28.8	40	0.00
63	1700	31.6	40	0.00
64	1800	27.8	40	0.00
65	1900	30.0	40	0.00
66	2000	30.8	40	0.00
67	2100	29.2	40	0.00
68	2200	29.4	40	0.20
69	1800	30.2	30	0.30*
70	1800	19.8	30	0.00*
71	1800	10.7	30	0.00*
72	2100	10.3	30	1.80**
73	1150	30.4	30	0.00

* From an initial 350 cc Hg. charge in bowl.

** From an operation duration of two hours.

tion of an increased volume of mercury in the bowl's channel. Accordingly tests 69, 70, and 71 were run using pulp densities of 30, 20, and 10 per cent solids by weight respectively, and commonly, 1800 feet per minute rim speed, 30 gallon per minute pulp flow, and a 350 out of 400 possible cubic centimeter charge of mercury.

To spot check the effect that the time parameter has in a test experiencing mercury loss, test 72 was conducted as a repeat of tests 12 and 13 except that the operation duration was doubled to two hours. These additional tests also appear in Table 6.

Discussion Of Results

Figure 18 is a graphical representation of the first 68 test results from Table 6. Rim speed for the 12" bowl is plotted as a function of pulp density for 20, 30, and 40 gallon per minute pulp flow rates. Similar to Figure 7 concerning mercury retention for the 6" bowl, rim speeds under the pulp flow lines would be a safe operation zone with this particular combination of input parameters. Rim speeds above the pulp flow graph lines would result in a mercury loss.

From Figure 18 it is evident that low pulp densities in the 10 per cent range cause mercury losses unless rim speeds are decreased to compensate. Higher pulp densities in the 20 and 30 per cent ranges give a more stable operation mode and allow higher rim speeds to be used. This is the opposite of the situation for the 6" amalgamator.

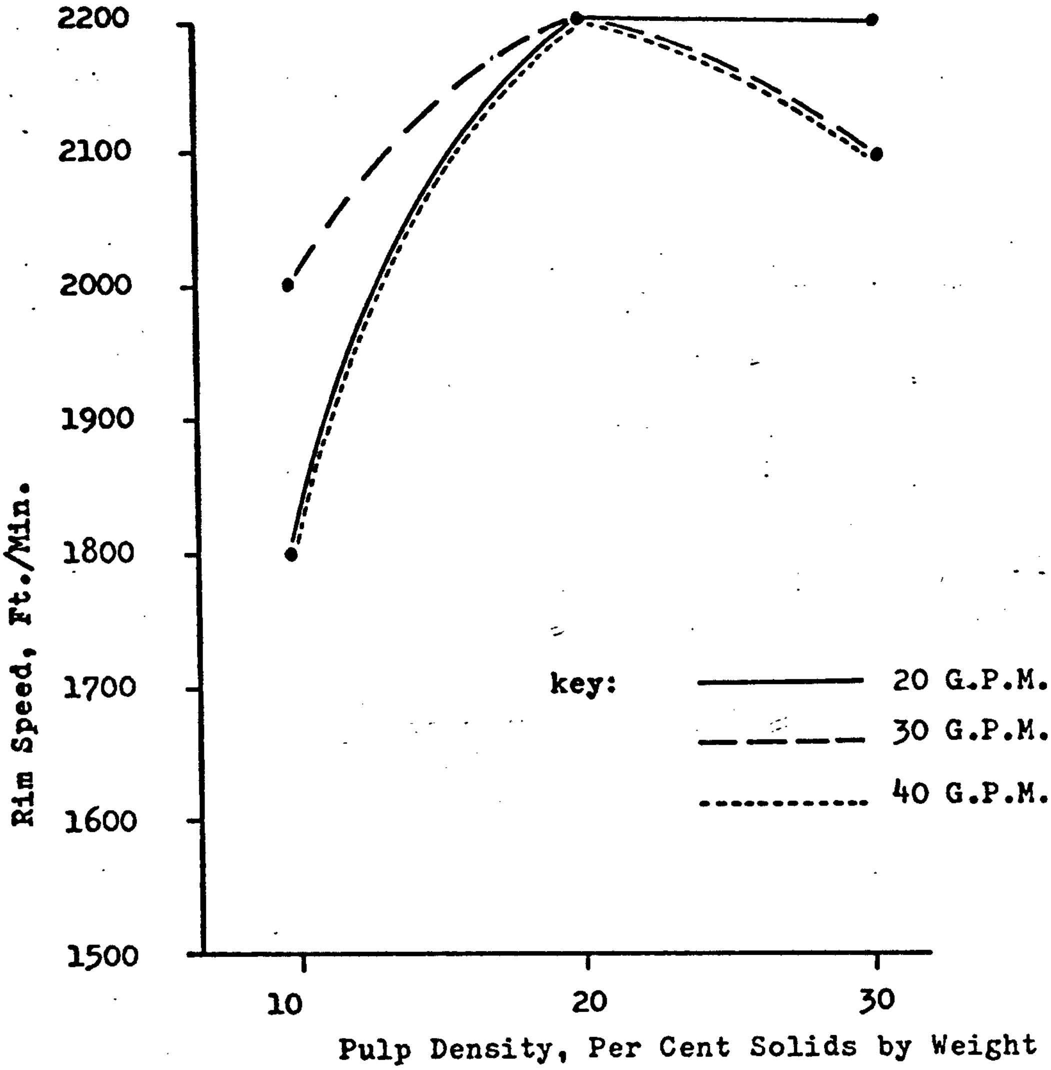


Figure 18. Operating Range of Rim Speeds for Mercury Retention, 12" Bowl

The mercury loss at the 10 per cent pulp density range is believed to be a consequence of solid particle impingement on the mercury sleeve in the bowl during operation. Observations at cleanups during the testing phase revealed that the lost mercury was in an extremely floured condition, thereby supporting the impingement theory. Also encouraging this idea is the observation that higher pulp densities provide a protective coating, of a viscous- like slurry flow, along the inner face of the mercury sleeve. Particles accelerating outward from the feed launder are thought to impinge on the slurry coating and slow down, saving the mercury sleeve from being struck and floured. The slurry coating is observed to be greately diminished in thickness during operations with 10 per cent pulp densities, and it is supposed that impingement occurs at this time.

Tests 69, 70, and 71, run to spot check the effect of increased mercury volume on the 12" bowl's mercury retention characteristics, showed no losses for 69 or 70 and 0.3 per cent loss for test 71. Although three tests are not comprehensive enough to be conclusive, this data indicates that functional ranges are probably similar in nature for larger mercury charges as for the 250 cubic centimeter charge from which the retention curves were developed.

One test was run to spot check the time factor effect on mercury losses. Run 72 was a duplication of runs 12 and 13, except that in this case the operation duration was doubled to two hours. Conditions were 2100 feet per minute rim speed, 250 cubic centimeter charge of mercury, 10 per cent pulp density, and 30 gallon per minute pulp flow rate. Tests 12 and 13 lost 2.40 and 0.80 per cent of the input mercury respectively. Run 72 lost 1.80 per cent of the input mercury. The conclusion drawn from these tests is that flouring losses do not seem to be constant enough over a short time period to accurately see the influence of time on the amount of mercury loss. One can merely see whether the parameters are in a mercury loss or a mercury retention operational mode, and assume that mercury losses will be time functional in some manner.

Test 73 was a check to see if mercury loss would occur at the lowest rim speed attainable with the presently existing drive linkage on the drive motor to the 12" bowl. This is in anticipation of rim speed limits as concerns upcoming gold recovery testing. Accordingly the test was run at 1150 feet per minute rim speed, 30 per cent pulp density, 250 cubic centimeter charge of mercury, and 30 gallon per minute pulp flow. No mercury loss was noted.

B) Gold Recovery

This testing phase for the 12" amalgamator was basically a repeat of the procedure used on the 6" bowl where parameters of rim speed, pulp density, capacity, and input gold sizes were investigated as effect gold recovery. This test series was designed to be a spot check of the trends set forth from the testing of the 6" bowl, and therefore series comprehensiveness has been cut down slightly. The input parameters from the 6" bowl's tests have been scaled up to accommodate the increased capacity of the 12" bowl. This series is composed of 33 tests as represented in Table 7. Input parameters investigated were; pulp densities of 10 and 30 per cent solids by weight, and volumetric flowrates of 10, 30, and 40 gallons of pulp per minute. Rim speeds of 1327, 1622, 1917, and 2212 feet per minute were investigated which correspond respectively to the forces, of the magnitude $\frac{(\text{velocity})^2}{\text{radius of bowl}}$, generated by the 6" bowl from rotations of 900, 1100, 1300, and 1500 feet per minute. These rates generate 31, 46, 64, and 86 g's respectively at the mercury surface. Rim speeds corresponding to 500 and 700 feet per minute in the 6" bowl could not be generated in the 12" amalgamator due to the nature of the large bowl's motor drive arrangement. The input charge of mercury in the 12" bowl was similar to the charge used in the small bowl in that 75 per cent of the channel capacity, or 300 cubic centimeters, was used. The effect on gold recovery of gold particle size is spot checked by conducting the first 21 tests with 20/28 mesh gold, and the last 12 tests with 35/48 mesh gold particles.

The previous mercury retention tests related to this series demonstrated that mercury losses were gradual in nature due to a slight flouring of the mercury. Therefore since the test duration for this series was about five minutes per test, only very small amounts of mercury could be lost during testing when loss conditions were present. Accordingly some gold recovery tests

Table 7 (Continued)

Test No.	Rim Speed Ft./Min.	Pulp Density Wt. % Solids	Capacity G.P.M. Pulp	Hg. c.c.	Gold In Grams	Gold In Particles	% Gold Rec. Grams	% Gold Rec. Particles
18	1917	33.0	30	300	0.0824	40	109.8	120.0
19	2212	28.7	30	300	0.1192	40	104.7	112.5
20	1327	32.8	40	300	0.0838	40	88.0	85.0
21	2212	30.9	40	300	0.1112	40	104.5	110.0
(35/48 Mesh Gold Particles)								
22	1327	12.9	10	300	0.0212	40	78.4	85.0
23	2212	8.0	10	300	0.0191	40	80.1	92.5
24	1327	11.2	30	300	0.0181	40	56.9	72.5
25	2212	11.3	30	300	0.0199	40	106.9	105.0
26	1327	12.5	40	300	0.0195	40	78.9	82.5
27	2212	13.0	40	300	0.0190	40	84.9	85.0
28	1327	28.6	10	300	0.0166	40	106.6	92.5
29	2212	27.5	10	300	0.0167	40	104.5	102.5
30	1327	30.1	30	300	0.0172	40	99.2	105.0
31	2212	29.1	30	300	0.0183	40	98.3	100.0
32	1327	29.0	40	300	0.0198	40	106.3	100.0
33	2212	28.3	40	300	0.0183	40	104.5	97.5

were run in the mercury loss range, as poor gold recovery would not in this case be a consequence of mercury loss, and also because it was desired to discover the gold recovery abilities of the bowl in part of the mercury loss range.

Discussion Of Results

The data from Table 7 indicates that gold recovery is uniformly high throughout this series of tests. Due to the fact that insufficient data points were taken to plot unquestionable curves, the illustration of this data can be seen in Figures 19 and 20 as a zone into which all of the data points could fit. Figure 19 shows per cent gold recovery against rim speed for the various combinations of pulp densities and flowrates used for 20/28 mesh gold size. Figure 20 relates the same information but for 35/48 mesh gold size. All points above 100 per cent gold recovery are assumed due to gold recirculation lost from previous testing. The low 56 per cent gold recovery by weight result of run 24 was observed to be due to a dissolving away of part of the gold from each gold particle during the handling process. Therefore the per cent recovery by particle result is the more accurate of the two evaluation methods in this case.

The indication of this data is that there is no great difference in gold recovery between points plotted from the various combinations of rim speeds, pulp densities, or volumetric flowrates. The data point zone in Figure 20, using 35/48 mesh gold, can be seen to have slightly lower per cent gold recovery limits

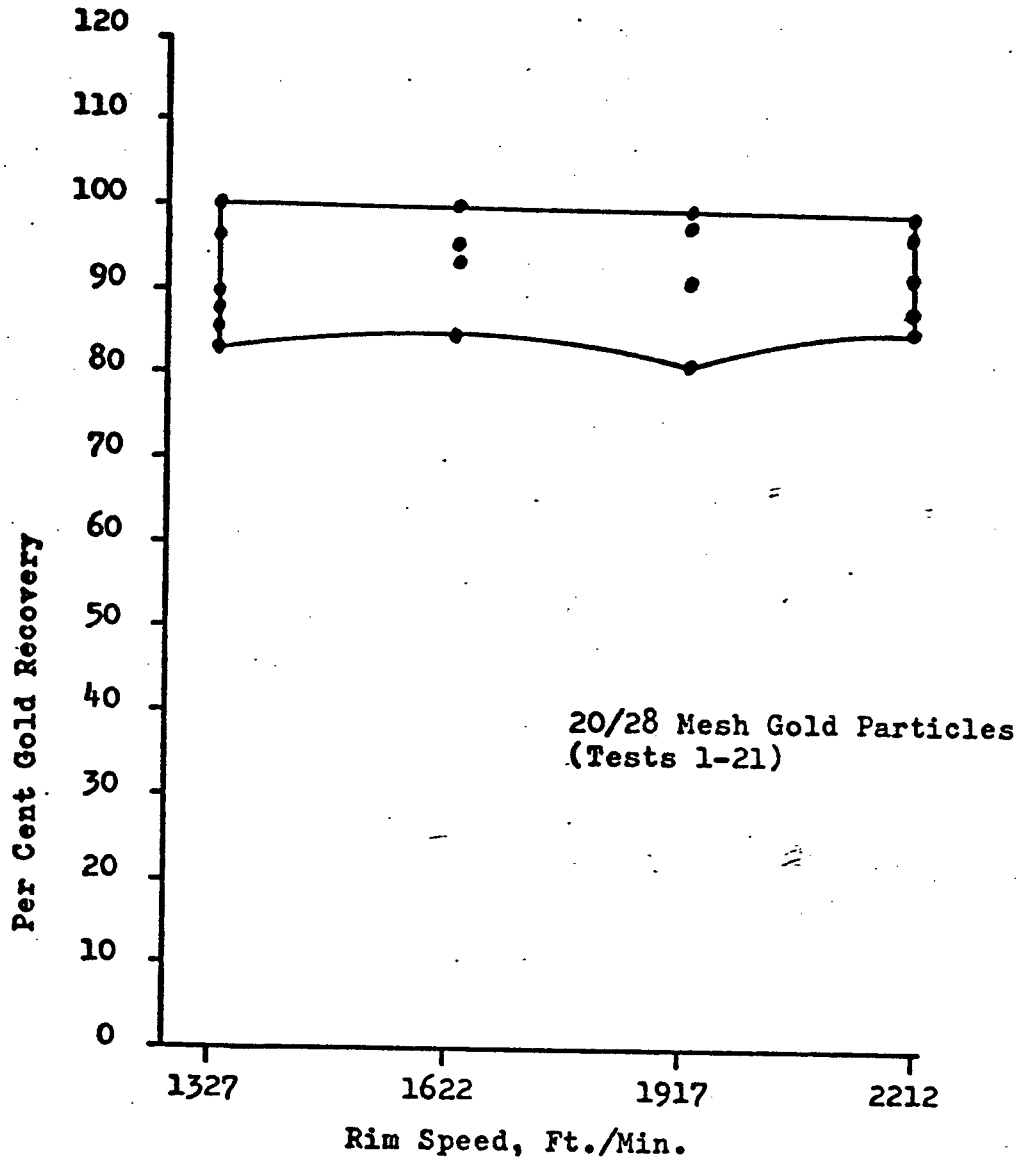


Figure 19. Gold Recovery by Weight and Particle

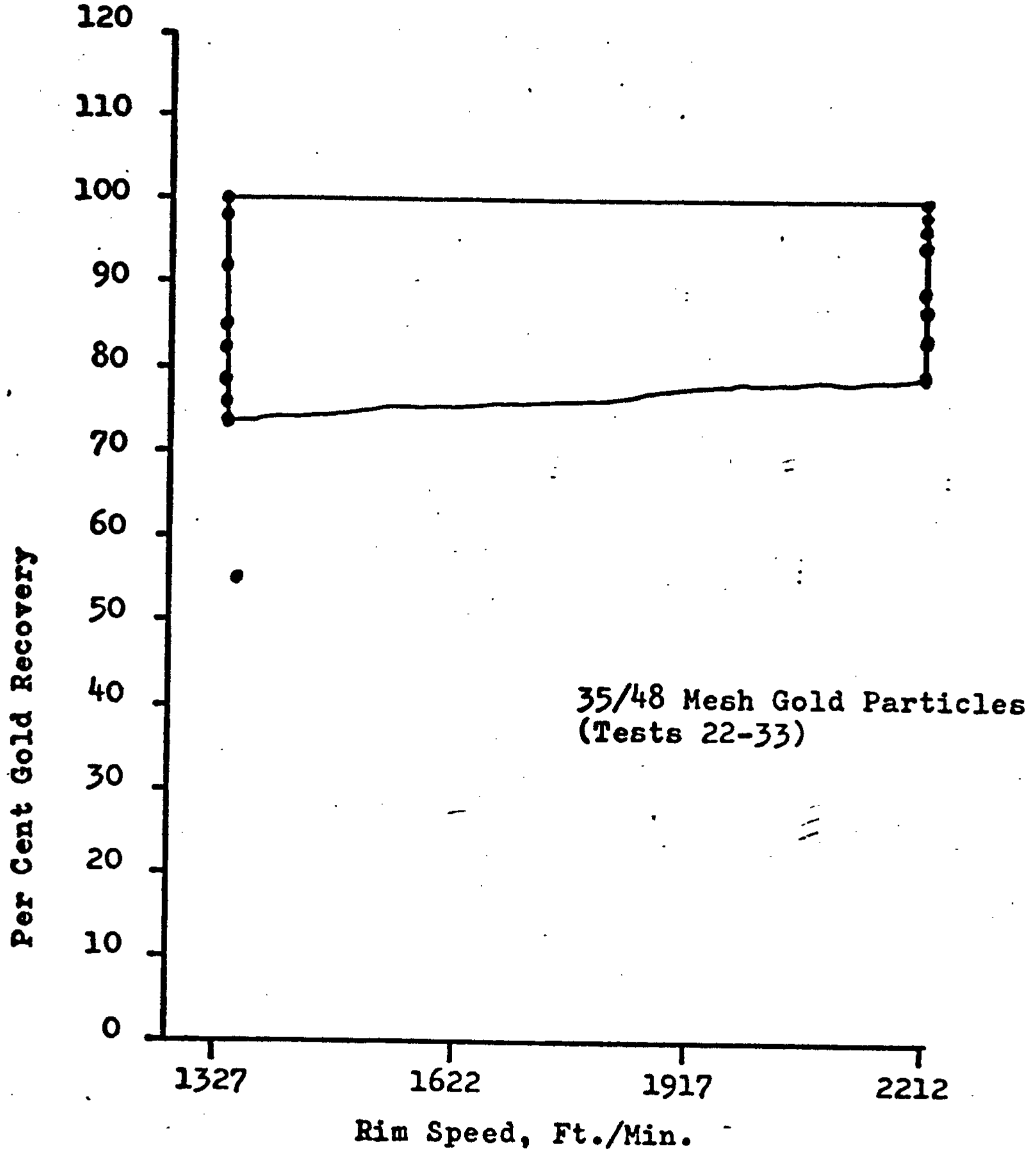


Figure 20. Gold Recovery by Weight and Particle

than does the recovery zone from Figure 19 using 20/28 mesh gold.

3.4 Miscellaneous Tests, And Significant Observations And Comments Derived From Testing

Part A of this section contains several miscellaneous test descriptions and results which do not fit into the previously considered series of tests. Their purpose is to provide answers in anticipation of various questions which might logically occur. This aspect therefore provided for a varied scope in the nature of these investigations. Included in this section are; out of level mercury retention tests for the 6" amalgamator, sequence of startup procedures on the 12" bowl as concerns mercury retention, and testing of the ability of the 12" bowl to recover gold from samples of Alaskan gold-bearing placer deposits which are comprised of finer gold particles and heavier specific gravity solids.

Part B of this section presents pertinent observations concerning the functional quirks of the centrifugal amalgamators which have been detected during the testing program. Of particular interest is a description of the former tendency of the mercury channel in the 6" bowl to propagate waves of mercury around the cylindrical mercury sleeve, resulting in violent vibrations and mercury spillages during operation. Accompanying this description is an explanation of the successful preventative measures taken to negate this phenomenon.

A) Miscellaneous Tests

Out of level tests.

Three tests were conducted on the 6" amalgamator to determine the effect on mercury retention of operating not at a true level attitude. Common input test parameters were; one half hour operation duration, 6 gallons per minute flow of a 20 per cent by weight pulp density slurry, 1500 feet per minute rim speed, 95 cubic centimeters charge of mercury into the bowl, and minus 8 mesh, sized silica sand for solid component of slurry. The out of level attitude of the amalgamator varied at 0.7, 1.3, and 1.9 degrees for this test series. No mercury losses were noted.

Startup sequence for mercury retention on 12" bowl

Four tests were conducted on the 12" bowl to determine if combinations of conditions or situations exist which would cause mercury losses from the bowl during the starting phases of operation.

For test 1, 250 cubic centimeters of mercury were placed in the otherwise empty bowl. No pulp or water flow entered the bowl. From a stationary mode the bowl was quickly brought to a rim speed of 1500 feet per minute, operated for 20 seconds, then brought to rest. Three such tests showed no mercury losses.

Test 2 was the same as test 1 except that about one half gallon of water was placed, along with the 250 cubic centimeters of mercury, into the stationary bowl. There was no flow through the bowl during operation, and three rotation tests showed no mercury losses.

Test 3 had the bowl rotating and containing dry mercury only. Feed slurry was then applied, at speed, to see if the feed slurry initially hitting the dry mercury sleeve would cause mercury loss. Conditions were 1500 feet per minute rim speed, 250 out of 400 possible cubic centimeters of mercury in the charge, feed slurry at 12 per cent pulp density solids by weight, and 40 gallons per minute of flow. The result was no mercury loss.

Test 4 was a duplicate of test 3 except that capacity was increased to 45 gallons per minute of pulp. The test had a one minute duration. There was no mercury loss observed. It was noted that the feed delivery pipe momentarily clogged just ahead of the amalgamator, when the test first started. This resulted in a solid slug of moist sand being injected, just ahead of the regular 12 per cent pulp density slurry, into the dry rotating bowl. It was noteworthy that this drastic feed situation did not cause a mercury loss when the sand slug struck the mercury sleeve.

Gold recovery 12" bowl, Alaska placer deposits

Gold recovery testing on the 12" amalgamator was conducted in duplicate on samples of Alaska gold placer sands taken from two locations in the state. The first sample was taken from auger cuttings from what was believed to be the bedrock fraction of an ancient beach placer at Dry Creek just inland from Nome Alaska. The alluvium material was of an extremely slimy nature where the bulk of the minus 200 mesh material forms a very sticky clay gumbo when wet. Specific gravity of the solids was measured as 2.65. The gold from this sample, as observed in a gold pan, is estimated to be around minus 65 mesh. (See APPENDIX A for further information about test procedures and sample description) Duplicate tests, delivering in a single pass 86 and 91¼ pounds respectively of sample through the 12" amalgamator, were run using 30 per cent pulp density by weight, 30 gallons per minute of pulp flow, 1326 feet per minute rim speed, and a 300 cubic centimeter charge of mercury in the bowl. Gold weights of 33 and 12 milligrams were recovered from the two tests respectively. Extended barrel amalgamation of a sample of the tailings showed only trace amounts of gold as being rejected.

The second placer sample was taken as a grab sample in a naturally concentrated zone of gold bearing beach sand at Cape Yakataga Alaska. This material is composed mainly of garnet sand, lacks slime-size material, and has a measured specific gravity of 3.57. The gold is very flat and finely sized, thereby making it very difficult to recover from the garnet sands by conventional methods. (See APPENDIX A for further information concerning sample characteristics and test procedures) Duplicate tests on the 12" bowl were similarly run with this material, delivering 107 and 121¼ pounds of sand respectively through the bowl on a single pass basis. Other test parameters were 20 per cent pulp density by weight of slurry flowing at 30 gallons per minute through the bowl. Rim speed of the bowl was 1326 feet per minute. Mercury charge in the bowl was again 300 cubic centimeters. The test results showed no mercury loss from the bowl. The tests recovered 74.25 and 56.51 milligrams respectively of gold from the beach sands from that single pass. Flotation analysis to obtain the head values of the sand indicated that there was an average 99.84 and 101.82 milligrams of gold respectively in those head samples. The per cent gold recovery based on those figures computes to be 74 and 55 per cent respectively. It was significant to note the amalgamator's satisfactory mercury retention performance with this heavy specific gravity sand feed.

B) Observations And Comments

These remarks reflect random but significant observations made throughout the testing program on the performance characteristics of the amalgamators.

Modification of 6" bowl's mercury channel

Before this testing program got underway, it was necessary to modify the 6" amalgamation bowl's mercury channel to improve its mercury retention capabilities. Initially the channel was configured so as to form a right cylindrical shell volume. It was observed, with the aid of a strobe light's ability to visually stop high speed cyclic motion, that there were actually waves of mercury generated and propagated around the perimeter of the cylindrical sleeve of mercury which occupied the bowl's channel. This happened at rotation rates starting at 950 revolutions per minute. The net result was an unbalancing effect on the bowl involving the uneven weight distribution of the zonal concentrations of the high density mercury in the waves. When this occurred, violent vibrations would wrack through the entire amalgamator chassis which would expel most of the mercury from the bowl's channel. Figure 21 shows the observed configuration of the propagating waves.

It was thought that some minor eccentricity in the otherwise carefully dynamically balanced bowl would initially cause a mercury wave to generate. Then it was supposed that if the wave was allowed to propagate unobstructed around the channel at a velocity which was some even multiple of the bowl's rim speed, there would occur times when the wave and the initiating eccentricity would become coincident. This would result in a continued resonant reenforcement of the wave which would magnify the problem to its final dramatic condition.

Based on the above theory, the remedy used was to weld three narrow wave-breaker strips of metal in the bowl's channel parallel to the axis of rotation and at 120° spacings. This in effect prevented the possibility of this type of resonant reenforcement by obstructing the mercury wave's travel path around the channel perimeter. The total lack of this problem after the wave-breaker strips were installed indicated that the remedy was sufficient. The 12" bowl's channel was filled with a one-layer woven wire mesh to deal with this problem. This would have the mercury filling the numerous voids provided by the mesh thereby

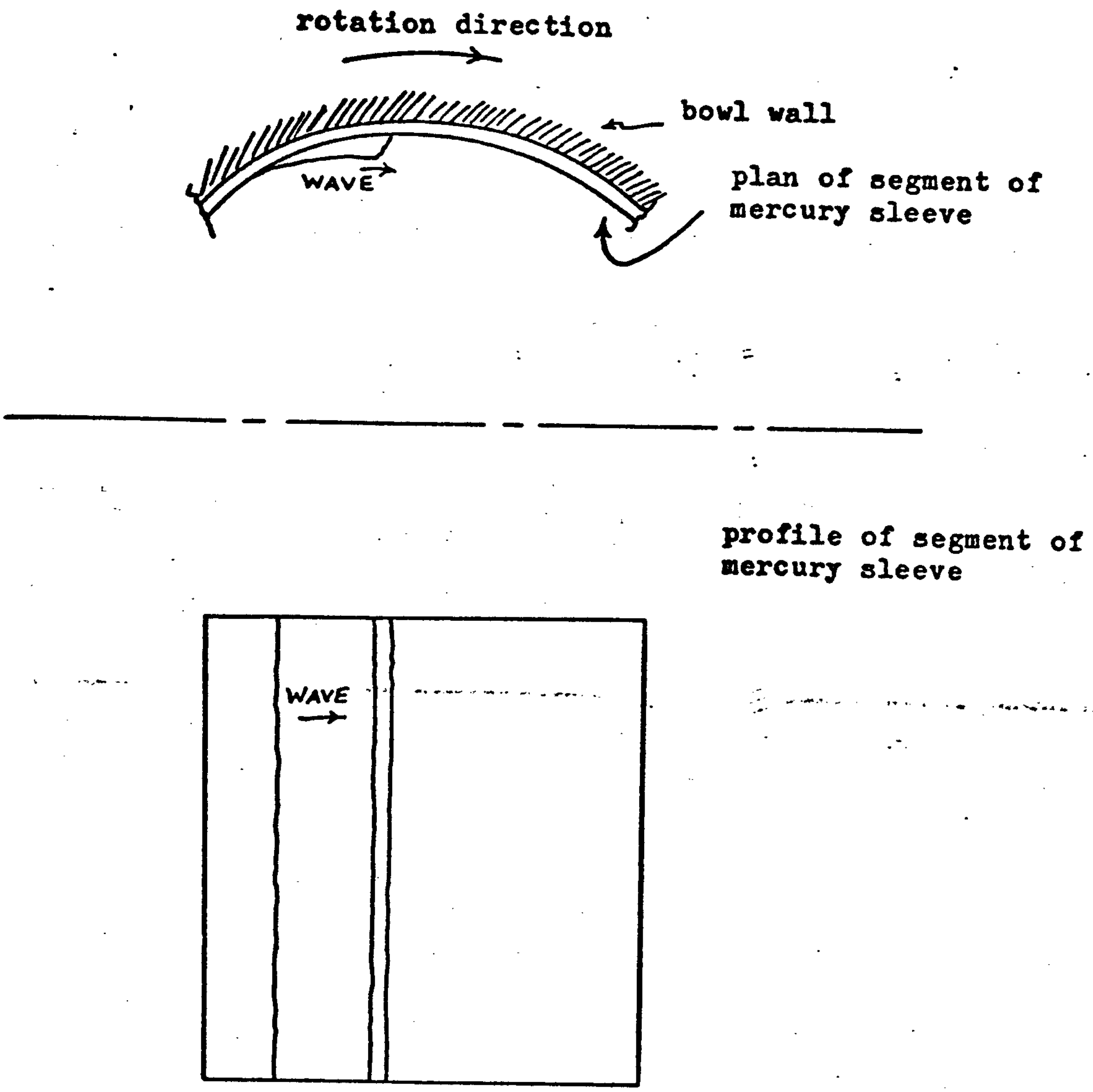


Figure 21. Wave Configuration in Mercury Sleeve

allowing no motion whatsoever in the mercury sleeve. This method also proved satisfactory.

Observations concerning mercury loss during shutdown

Periodically during the testing program the 12" amalgamator was operated in the standard manner, recieving the regular circulating pulp flow, but without its mercury charge. The intent was to purge the circulating slurry load of any lost and floured mercury. During the end of one such run using 1500 feet per minute rim speed, 30 gallons per minute pulp flow, and 10 per cent pulp density solids by weight, the pulp flow was shut off while the bowl continued to rotate at speed for about two minutes. The end result was that there was an extremely hard packed sand shell along the bowl channel. In addition four cubic centimeters of mercury were recovered from the bowl, but there was observed to be about one half cubic centimeter of mercury, in a floured condition, laying in the overflow launder. This suggests that the mercury can possibly escape the bowl when such a hard sand shell exists in the bowl. It is assumed that since the discharged mercury did not appear to be washed down the launder to any degree, the loss must have occurred in the last two minutes of operation during the period of no flow. Subsequent tests having a small amount of water flowing into the bowl up until the time that the bowl is stopped showed that this minimizes the hard sand shell and prevents this type of mercury loss.

Pulp spillages from feed flow into non-rotating bowls

It has been observed that at all times when there is a flow of water or pulp through the bowls, the bowls must be kept rotating to prevent overflow spillages. The spillages occur as a consequence of the clogging of the gap between the bottom of the feed launder and the bottom of the bowl volume, thereby causing the feed of water or pulp to back up into the feed launder and overflow at the nearest opening. The clogging occurs from either having the pulp solids or the bowl's mercury filling that gap. Bowl rotation prevents this, for the mercury fills the channel instead of the gap, and the feed particles are being continuously accelerated toward the channel and away from the gap due to the rotation's effects.

Capacity limitations for the amalgamation devices

It has been observed that the factor limiting the capacity of pulp flow through both bowls is the size of the discharge launder at the mercury traps. Flows above 9 gallons of pulp per minute for the 6" bowl, and above 45 gallons per minute for the 12" bowl would cause overflow spillages at those launders. There was nothing to indicate during the testing that the amalgamators could not otherwise handle somewhat larger capacities adequately with respect to processing the feed, having mercury retention, and simultaneously having good gold recovery.

3.5 Data Reliability

The intent of this project was to get a broad picture of the performance ability of the amalgamators. This widened the scope of the testing at the expense of the comprehensiveness which could have been obtained had investigations been confined to more narrow limits. With this approach and with time deadlines in mind, the majority of tests were run without precision check duplications. Test results were considered to be reasonably reliable, without precision check duplications, if those results fit closely into the trend set by the related results in the associated test series. In cases where the anomalous results did not fit into a series trend, or where there were not enough points to see a trend, then duplicate tests were run to check the validity of the data.

In consideration of the above procedure it is suggested that the absolute values of the test results, obtained from the use of particular parameters, not be held as sacred. For example tests yielding results of 2.6 per cent mercury loss or 98 per cent gold recovery should not be considered except as parameter combinations yielding a mercury loss mode or yielding a very high gold recovery mode. The values of the numbers are that they point out the relative effects on the amalgamator performance characteristics of changing input parameters. For the numbers to have validity in an absolute sense, it is suggested that multiple tests or long term tests be carried out under a particular set of conditions and then evaluated as results which will be expected to average that particular performance designation.

It is also suggested that considerations of pulp density values, from the input parameter group, be thought of in terms of merely being in the ranges of 10 per cent, 20 per cent, or 30 per cent since the tabulated values represent the average of several measurements taken over the test period.

CHAPTER 4

CONCLUSIONS AND RECOMMENDATIONS

The data presented in Table 4 and as plotted in Figure 7 clearly shows that rim speed correlated with pulp density and volumetric flow rate influences mercury retention in the 6" diameter amalgamator. Retention for given pulp densities and flow rates is adversely affected by increasing rim speeds. For given rim speeds retention is adversely affected by increasing pulp density, but is improved by increasing capacity. The data shows that indeed feasible operation ranges do exist for the 6" bowl, at the lower rim speeds, where total mercury retention can be expected.

The data in Table 5 and plotted in Figures 9 to 17 show that rim speed drastically affects gold recovery in the 6" amalgamator where gold recovery efficiency falls off with increasing rim speed. The lowest rim speeds account for the highest recovery rates. Fortunately this range is coincident with the mercury retention range. From this data it can also be seen that higher pulp densities for given flow rates require a drop in rim speed to gain maximum gold recovery.

The effect on gold recovery that gold size has, for the 20 to 48 mesh gold size range tested, is not significant.

Based on the preceeding information it is concluded that rim speeds on the 6" amalgamator, for the test conditions used, should not exceed 700 feet per minute. A speed of 700 feet per minute will provide the relative maximum gold recovery, and will provide a large operating range for mercury retention.

The data in Table 6 and plotted in Figure 18 shows that again rim speed, acting with capacity and pulp density, affects mercury retention in the 12" amalgamator. However the nature of the influence is different in this case than in the case of the smaller amalgamator. Pulp density is now also of a critical importance. Here a low pulp density of 10 per cent has a large adverse effect on mercury retention where rim speeds have to be reduced to retain the mercury. The large pulp density of 30 per cent shows this same trend but in a less drastic fashion, leaving the 20 per cent pulp density as the most stable range. Increasing rim speed again adversely affects mercury retention, where the lower speed ranges provide the safest operating zone. Mercury retention is not as drastically affected in this case by changing flow rate as in the case of the smaller amalgamator.

Tests relating the parameter influence on gold recovery efficiency for the 12" bowl, as presented in Table 7, showed the larger amalgamator to enjoy uniformly high gold recoveries from all of the various combinations of rim speeds, flow rates, pulp densities, and gold sizes. Therefore it is suggested that the operating ranges of the 12" bowl be confined only to limits protecting against mercury losses. This would include the pulp density range as being 20 per cent solids by weight.

Relative to each other, the larger amalgamator can be seen to produce generally a higher gold recovery efficiency than does the smaller bowl. This is thought to be a consequence of a more limited stratification opportunity in the small bowl, as compared to the larger bowl. Figures 1 and 2 reveal that the two bowls do not have the same configuration. It can be seen that there is a longer stratification zone, from the feed launder discharge point to the top of the mercury channel, in the 12" bowl. It is hereby noted therefore that rewarding potential exists in gold recovery investigations as a function of a changing configuration of the amalgamation bowl's stratification zone.

The uniformly high gold recovery ability of the 12" bowl compares favorably against the traditional gold recovery devices. It is anticipated that investigations concerning gold recovery from the processing of more closely sized materials will show still higher gold recovery efficiencies. It is therefore recommended that this investigation base be expanded to include studies showing the effects of size control of feed particles on the amalgamator's gold recovery ability.

It was encouraging to note the larger amalgamator's good performance on the heavy specific gravity Yakataga Beach garnet sands. The fact that good gold recovery was achieved with total mercury retention indicates that there exists a practical operating range for higher specific gravity materials. This suggests that the amalgamator might also be successfully applied to the task of recovering gold from heavy concentrate products derived from other recovery methods. Valuable information therefore would be obtained from the systematic investigation of gold recovery as a function of a changing specific gravity of the feed solids.

It would also be of interest to alter the discharge launders at the mercury traps on each amalgamator to accommodate larger flows. This in effect would remove the limiting restriction that stands in the way of determining the amalgamator's true capacity.

Final Conclusion

This final concluding summary is directed at the question, " Is the Gilkey Centrifugal Amalgamator capable of satisfactory production in the field? "

This thesis points out that; A) Parameter control is essential to the amalgamator's performance. B) The amalgamator is capable of high gold recovery when the input parameters are confined to the proper limits. However let it be noted that the order of magnitude of change between the parameters yielding good operation performances and those yielding poor operation results is in fact not large. Indeed it is difficult in large production sized operations to maintain as close a quality controlled feed slurry delivery as the Gilkey bowls demand. This infers the utilization of a sophisticated pretreatment feed system. Therefore the answer to the question in point is that satisfactory performances from the bowl will be obtained only with the simultaneous high quality performance of the related feed system.

APPENDIX A

Analytical Procedure

The following is a description of the methods and materials used for the various tests in this study.

General Procedure For Mercury Retention Testing

Preparation Of Feed Slurry:

A) Solids and water, to constitute the circulating slurry load, were placed into the source tank. Solids were composed of well washed silica sands having a specific gravity of 2.61. The screen analysis of the input sand was;

Tyler mesh	Weight per cent
6/14	11.5
14/28	36.0
28/48	40.0
48/100	10.0
100/200	2.0
-200	0.5
Total	100.0

B) Valves in the plant were arranged to shunt about 1/3 of the flow through the amalgamator, having the remainder of the circulating load by-pass directly from the primary splitter back to the source tank.

C) The bowl was then rotated at lower rim speeds of around 500 to 900 feet per minute.

D) The plant's pump was activated to maintain the pulp flow.

E) The pulp was allowed to circulate until equilibrium was achieved with respect to pulp density and flow rate.

F) Valves were adjusted to shunt desired flowrates through the rotating bowl. Flow rate was measured with a stopwatch and volume bucket at the amalgamator's discharge.

G) Pulp density by weight was measured from a sample of slurry which flowed from the discharge launder at the mercury trap. The measurement consisted of; 1) partially filling the volume bucket with the slurry, 2) weighing the contents of the bucket, 3) determining the specific gravity of the pulp by dividing the results of step 2 by the results of step 1 where the weight per volume was expressed in g./c.c., 4) calculating the pulp density by weight of the slurry according to:

$$P = \frac{Ss(Sp-1)}{Sp(Ss-1)} \times 100\%$$

where P= pulp density by weight
Ss= specific gravity solids
Sp= specific gravity pulp
1= specific gravity water

H) The contents of the sample bucket were returned to the source tank.

I) Solids were added or removed, and/or water was added or removed from the source tank, then the measurements of step G were repeated until the desired pulp density range was observed to be consistently present.

Testing Sequence

J) The pump was shut down, slurry flow ceased, and the bowl was stopped, cleaned out, and charged with the desired volume of mercury.

K) The bowl was then made to rotate around 700 to 900 feet per minute rim speed.

L) The pump was started, slurry flow recirculated, and a pulp density measurement made as per step G, to insure that the proper range was present.

M) Flow rate was rechecked with stopwatch and volume bucket from a sample taken from the discharge launder at the mercury trap.

N) Bowl speed was brought up to desired operation rate. Tachometer readings were made to check for proper drive shaft speeds.

O) The test operation was maintained for the specified duration. During the test interval at least 3 pulp density measurements as per step G, were taken on the circulating load. Measurements were taken at the start, midpoint, and end of the test period. If reasonably close they were averaged and recorded. If the P.D. deviation was 4 % or more, the entire test was voided.

Evaluation Sequence

P) The pump was shut down, the bowl was slowed down, pulp flow ceased, the bowl was stopped, and its contents were removed to a bucket.

Q) The retained mercury was separated from the sand particles with the use of a small elutriator.

R) Graduated cylinder comparisons of input mercury volume versus retained mercury volume showed mercury losses or retentions.

S) When necessary after mercury losses, the mercury trap on the amalgamator was cleaned as was also the silvered amalgamation plate in the launder leading to the source tank.

T) Periodically the entire pulp slurry was elutriated and run slowly over amalgamation plates to remove any floured mercury.

General Procedure For Gold Recovery Testing

A) Procedures similar to steps A-N in the mercury retention sequence were carried out to maintain and measure the desired input parameters.

B) Gold amounts of known weights, particle counts, and sizes were manually salted into the feed slurry just ahead of the entrance to the amalgamator. The salting was on a several particl[illegible] [illegible]t a time feed basis, where about 5 minutes were requir[illegible] [illegible]o salt the 40 particles.

C) Shutdown procedures, similar to steps P, Q, R, S, and T of the mercury retention testing series, were carried out.

D) Gold particles were separated from the volume of mercury by filtering through a chamois. The separation process was facilitated by the use of an amalgam plate and scraper.

E) Residual mercury was removed from the gold with nitric acid.

F) The gold particles were cleaned, dried, re-counted, and re-weighed. Recovery efficiencies were evaluated on the basis of input versus retained gold amounts.

<u>General Procedure For Processing Alaskan Placer Sands From Nome And Yakataga</u>

It was desired to process each sample through the 12 inch amalgamator on a one pass basis. Accordingly the material was fed in a dry state, from the vibrating feeder, into a water mixing launder which led into the amalgamator's feed opening. The specific methods are as follows:

Nome Sample

A) The sample was run through a roll crusher and vibrating screen to eliminate plus 6 mesh rock particles imbedded in dry clay balls.

B) The sample was split in half with a sample splitter to provide material for duplicate amalgamation test runs.

C) It was desired to process the material at 30 per cent pulp density, 30 gallons per minute flow, 1326 feet per minute rim speed and with a 300 cc mercury charge. The mercury charge was placed in the bowl and the rotation rate control was set to achieve the desired rim speed. To achieve the 30 per cent pulp density and 30 gallon per minute feed, it was calculated that for this 2.65 specific gravity feed material, solid-liquid ratios should be 699.6 grams per second of material mixed with

25.8 gallons per minute of water.

D) Water was placed in the source tank and circulated through the plant's system and through the 12 inch amalgamator at a rate of 16.8 gallons per minute, as measured with a volume bucket and stopwatch.

E) The vibrating feeder was set to feed the sample at the rate of 699.6 grams per second, as measured with a stopwatch and balance, into the water mixing launder.

F) The sample was washed down the launder leading into the amalgamator with nine gallons per minute of water from an independent source.

G) At the entrance to the bowl, the 16.8 gallons per minute circulating water flow and the nine gallons per minute slurry flow from the launder were merged.

H) Recirculation of the bowl's discharged particles with the 16.8 gallons per minute circulating water load was prevented by altering the source tank. The tank was made to act in the capacity of a thickener where a stand pipe was installed in the floor drain in the tank, making the water exit the tank in an overflow mode rather than by underflow. In this way the discharged particles from the bowl would flow into the source tank and then settle out of the circuit.

I) When the batch feed lot was completely fed out, the water flows were shut off, the bowl stopped, and the system was cleaned, mercury collected, and gold recovered in a similar fashion to the procedure used in the routine mercury loss and gold recovery test series.

J) The weight of gold recovered was based on the gold recovered as a filtered residue in the chamois plus the weight of gold that was estimated to be dissolved in the mercury, based on the analysis of a proportional part of that mercury.

K) Duplicate one hour barrel amalgamations of a representative sample from each tailing material were used to determine the amount of gold rejected. Again the total weight of rejected gold was computed as the weight of gold residue from the chamois filter plus the weight of gold estimated to be dissolved in the mercury.

Specific Gravity Test

The specific gravity of the Dry Creek sample was estimated by displacing a volume of water with a 500 gram sample of material. The displaced volume, 189 cubic centimeters, indicated that the specific gravity of the sample was about 2.65.

Screen Analysis Of Dry Creek Sample

Tyler mesh	Weight on mesh (grams)	Weight per cent
6	7	1.3
8	17	3.2
10	38	7.3
14	42	8.0
20	35	6.7
28	32	6.1
35	25	4.8
48	18	3.4
65	38	7.3
100	41	7.8
150	42	8.0
200	23	4.4
-200	166	31.7
Totals	524	100.0

Yakataga Sample

The processing of the Yakataga sample closely paralleled the methods used on the Dry Creek sample.

The sample was divided with a sample splitter yielding two bulk splits for duplicate runs, plus a split of several representative pounds for head value gold determination by flotation.

The samples were desired to be processed through the 12 inch amalgamator on a one pass basis. Accordingly the test plant was set up in a similar fashion as for the Nome material.

The 3.57 specific gravity sands were desired to be processed at 20 per cent pulp density solids by weight and 30 gallons per minute pulp flow. The liquid-solid ratio was determined to be 28 gallons per minute of water flow plus 442 grams of solids per second. The feed arrangement in the plant was accordingly set at 19 gallons per minute water in the circulating load circuit, with nine gallons per minute water and 442 grams per second solids coming from the water mixing launder and vibrating feeder.

The remainder of this test was carried out in a fashion similar to that for the Nome sample. The only divergence being that gold recovery in the amalgamator was based on the weight of recovered gold compared to the head values as determined by gold flotation tests.

The screen analysis and gold size distribution of the Yakataga sample, (4) are as follows:

Screen Analysis Of Yakataga Sample

Tyler mesh	Weight on mesh (grams)	Weight per cent
10	52	0.24
14	122	0.55
20	331	1.50
28	1100	4.99
35	5327	24.15
48	7271	32.97
65	5593	25.36
100	1936	8.78

Yakataga Screen Analysis (Continued)

Tyler mesh	Weight on mesh (grams)	Weight per cent
150	296	1.34
200	18	0.08
270	1	0.00
400	2	0.01
-400	6	0.03
Totals	22055	100.00

Gold Size Distribution Of Yakataga Sample

Tyler mesh	Per cent of gold on mesh
10	0.00
14	0.00
20	0.00
28	3.11
35	6.46
48	5.87
65	15.82
100	11.74
150	14.69
200	18.50
270	20.09
400	1.98
-400	1.74
Total	100.00

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APPENDIX B

INTERIM REPORT, GRANT GF-38893

to

OFFICE OF INTERNATIONAL PROGRAMS
NATIONAL SCIENCE FOUNDATION

COOPERATIVE RESEARCH WITH THE REPUBLIC OF CHINA ON THE EVALUATION AND RECOVERY OF DETRITAL HEAVY MINERALS

by

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OCTOBER 30, 1974

SUMMARY

A cooperative research program agreed upon between the National Science Foundation of the United States and the National Science Council of the Republic of China resulted from proposals submitted by American and Chinese Scientists. The proposals were initiated because the investigators had a common interest, a desire to exchange information and felt a need for greater cooperation between government agencies and educational institutions of their respective countries.

The program was started with investigations and exchange of information concerning the evaluation and recovery of detrital heavy minerals. This topic of common interest has expanded, however, to the general field of mineral beneficiation as it pertains to mineral resource development.

Investigations of detrital heavy minerals were initially confined to the East Coast of Taiwan. As indicated in this report, the potential is limited to possible magnetite - ilmenite concentrations along the shore line between Hualien and Taitung. The regions of greatest interest involve the Northwest and Southwest beaches, and the off-shore bars in the latter area. The presence of monazite in these sands and the potential for developing the rare earth oxides should be fully explored.

With the reorganization of several government research institutes into one administrative unit the cooperative program was expanded to include planning and initiation of one mineral processing facility for common usage. Indications are that a spirit of cooperation and the exchange of information will aid in the assessment of the availability and utilization of mineral resources. Hopefully, this should also include closer ties with those educational institutes, such as Cheng Kung University, which have programs of study in the mineral industries.

The investigators are of the opinion that the program has been successful in promoting a coordinated effort in research activities, a free exchange of information and ideas, and a feeling of cooperation between the United States and the Republic of China. Consequently, proposals for continuation and expansion of the program have been submitted to the respective agencies.

ACKNOWLEDGEMENTS

I would like to express my sincere appreciation to my Chinese Colleagues Mr. Lee Ven-Chung and Dr. Liu Hok-Shing for their excellent cooperation in the project and their hospitality during my visits to Taiwan.

Mr. Feng Tah-Tsung, Director of M.R.S.O. and Mr. Wang Chi-Wu, Director, Division of International Programs of the National Science Council have always supplied supporting assistance when needed. Their interest and support of the cooperative program is appreciated.

The cooperation and assistance of Mr. Chao Ching and other officials of the Institute of Atomic Energy Research is gratefully acknowledged.

I am indebted to many individuals in government agencies, corporations, research institutes, universities and private citizens who have without exception exhibited a fine spirit of cooperation and hospitality during my visits to Taiwan. These individuals too numerous to mention by name have contributed much to the strengthening of scientific and human relationships between the United States and the Republic of China.

This program was made possible through the sponsorship of the United States Bureau of Mines and the Mineral Industry Research Laboratory at the University of Alaska. I wish to express my gratitude for this assistance.

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INTRODUCTION

In accordance with the guidelines for cooperative research programs between the United States and the Republic of China, proposals were submitted to the National Science Foundation and the National Science Council of the respective countries. These proposals with Mr. Lee Ven-Chung and Dr. Liu Hok-Shing sponsored by the Mining Research and Service Organization and Dr. Donald J. Cook sponsored by the U. S. Bureau of Mines and the University of Alaska were approved with an effective date of June 1, 1973 and an expiration date of November 30, 1975.

A common interest of the investigators led to the initiation of the program into research concerned with detrital heavy minerals as found in alluvial type deposits. The June, 1973 visit of the American Investigator to Taiwan was, therefore, primarily concerned with a reconnaissance sampling program for heavy minerals along the Eastern sea coast. The samples obtained, as shown in Table 1, were split and one portion shipped to the University of Alaska for evaluation.

The time of the American Investigator's 1973 visit to Taiwan coincided with the formation of three Ministry of Economic Affairs Insitutes into one administrative organization to be designated as the Industrial Technology Research Institute. Discussions with President T. T Feng of MRSO and Dr. John Hove, Technical Advisor to ITRI, led to the concept of a Mineral Processing Laboratory as a desirable facility for ITRI and the Republic of China. As a result of these discussions, I was asked to assist in the design and organization of this unit.

This interim report, therefore, covers the results of the sampling program, recommendations for the continuation of this study, and an evaluation of the Mineral Processing Laboratory and other cooperative aspects as determined from the July 1974 follow-up visit to Taiwan.

Table 1

Sample Designation

Sample No.	Location	Description	Remarks
A1-A8	Chinguashih, Shvinantang	Shoreline	Core Samples
B	Chinguashih, Shvinantang	Off-Shore	Diver & Core Samples
C1-C5	Hualien, Tsungte	Shoreline	Core Samples (High Tide)
D1-D6	Hualien, Tsungte	Shoreline	Core Samples (High Tide)
D7	Hualien, Tsungte	Shoreline	Concentrate (After typhoon)
E	Hualien, Tsungte	Shoreline	Shovel Sample (Low Tide)
F1-F2	Hualien, Tailuko	Li Wuchi River	Shovel Sample
G	Hualien, Holiu	Li Wuchi	Shovel Sample
H1-H2	Hualien, Tienhsiang	Li Wuchi	Shovel Sample
I1-I4	Hualien, Chengpu	Shoreline	Shovel Sample (High Tide)
J	Taitung, Chang Yuan	Shoreline	Shovel Sample (High Tide)
K1	Taitung, Shih Yu San	Shoreline	Shovel Sample (High Tide)
K2	Taitung, Shih Yu San	Shoreline	Concentrate
L1-L2	Taitung, Cheng Kung	Shoreline	Shovel Samples (High Tide)
M1-M2	Taitung, Hsin I	Shoreline	Shovel Samples (High Tide)
N	Taitung, Pen Shui	Old Beach	Shovel Sample

HEAVY MINERAL PROGRAM

Chinguashih Area

Introduction

The Department of Oceanography of the Chinese Cultural College at Taipei conducted an off-shore sampling program in an area which has been a depository over many years for tailing material from the milling operations of the Chinguashih Copper Mine. Samples taken along nine lines perpendicular to the shoreline were a combination of grab samples from the surface of the deposited material and core samples to obtain some feeling for depth of the deposit and distribution of values.

Unfortunately, the samples were not large enough to obtain a representative split of each sample for processing. However, a small amount of each sample was obtained and this material blended into one composite sample representing the total off-shore sampling program. For purposes of this report this sample will hereafter be designated as Sample "B".

The writer, accompanied by Mr. Lee Ven-Chung and Dr. Liu Hok-Shing also obtained eight shoreline samples as estimated shoreward extensions of the sample lines from the off-shore program. These samples were taken along the shoreline to determine if any concentration had been effected by wave action, to gain some knowledge as to the distribution of the valuable components, and on a preliminary basis determine the processing necessary to recover the valuable constituents. The sand layer above the water line was found to superficial in depth, consequently, they consisted of shovel samples on the surface. These samples are designated A1-A8 respectively.

Sample "B"

Depending upon whether the individual samples were obtained by grab or core methods, they represented a variety of size compositions. A screen analysis of the composite sample is shown as Table 2. These data show that the material is predominantly in the minus 35 plus 150 mesh range with 14.1% as minus 200 mesh material.

TABLE 2

SCREEN ANALYSIS, SAMPLE "B"

Tyler Mesh Passed	Tyler Mesh Retained	Weight Grams	Weight %	Cumulative Wt. % Retained	Cumulative Wt. % Passed
-	10	25.5	1.0	1.0	100.0
10	14	33.1	1.3	2.3	99.0
14	20	81.5	3.3	5.6	97.7
20	28	163.4	6.6	12.2	94.4
28	35	191.8	7.7	19.9	87.8
35	48	305.3	12.3	32.2	80.1
48	65	402.5	16.2	48.4	67.8
65	100	262.7	10.6	59.0	51.6
100	150	505.2	20.4	79.4	41.0
150	200	160.1	6.5	85.9	20.6
200	Pan	346.4	14.1	100.0	14.1
	TOTALS	2477.5	100.0		

Each screen size fraction was submitted to a sink-float test using tetrabromoethane (specific gravity 2.96) as the separating liquid. The results of these tests, presented in Table 3, show the weight percent of the sink and float products in each size fraction and for the total sample. An interpretation of this data seems to indicate that some liberation of heavy sulfides is apparent at about 48 mesh with maximum liberation occurring at about 200 mesh. Microscopic observations of the float products of each screen size also indicated these liberation points.

The float fraction of each screen size was analyzed for copper, gold and silver and the sink fractions were blended together to form a composite sink which was also analyzed for the same elements. A sulfur analysis was also conducted on composite float and sink samples. These data including calculated head analyses are presented as Table 4. These data indicate that the gold values in the float products accompany the heavy sulfides and when liberation of the sulfides is accomplished the gold value in the float product diminishes. The composite sink fraction, which is a mixture of all sizes, shows a gold value of 0.40 troy ounces per short ton. However, this value would increase if the total sample were ground to a finer size to liberate the sulfides and gold from the gangue material.

To illustrate the latter point two preliminary flotation tests were conducted on 500 gram samples of the composite sample with results primarily concerned only with gold analysis and recovery. The first sample was ground for 30 minutes resulting in 84% minus 200 mesh material, and the second sample with a 45 minute grind produced 97% minus 200 mesh material.

The first flotation test was conducted in two stages as a bulk sulfide flotation with no attempt to separate the pyrite from other sulfides. The following results were obtained:

Unit	Weight Grams	Gold Ounces/Ton	Recovery %
Head	500.0	0.22	100.0
Conc. #1	97.9	0.50	44.5
Conc. #2	26.8	0.10	2.4
Tails	375.3	0.15	53.1

TABLE 3

SINK-FLOAT ANALYSIS, SAMPLE "B"

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % Float	Total % Sink
10/14	26.2	67.6	32.4	1.3	0.6
14/20	81.3	64.8	35.2	4.0	2.2
20/28	79.6	65.7	34.3	4.0	2.1
28/35	102.2	67.6	32.4	5.2	2.5
35/48	157.8	74.2	25.8	8.8	3.1
48/65	217.6	75.1	24.9	12.4	4.1
65/100	131.3	76.8	23.2	7.6	2.3
100/150	253.5	87.9	12.1	16.8	2.3
150/200	88.2	77.7	22.3	5.2	1.5
-200	186.4	62.8	37.2	8.8	5.2
TOTAL OR AVERAGE	1324.1			74.1	25.9

TABLE 4

ANALYSES, SINK-FLOAT PRODUCTS, SAMPLE B

Tyler Mesh Float	Copper %	Silver Oz./Ton	Gold Oz./Ton	Sulfur %
10/14	0.06	0.01	0.50	
14/20	0.06	0.01	0.02	
20/28	0.08	0.02	0.40	
28/35	0.07	0.02	0.10	
35/48	0.06	0.01	0.02	
48/65	0.04	Nil	0.20	
65/100	0.03	Nil	0.42	
100/150	0.08	0.02	0.08	
150/200	0.05	0.01	0.02	Composite Float
-200	0.09	0.03	0.04	9.9
Composite Sink Fraction	0.75	0.10	0.40	31.9
Calculated Head Analyses	0.24	0.03	0.21	15.6

The first concentrate was predominantly pyrite but also assayed 0.80% copper with a recovery of 73% of that element. The sulfur content of this concentrate was approximately 37%. Gold recovery in this case was low due both to inadequate grind and no attempt to float free gold. The calculated head assay of 0.22 ounces/ton checks well with the analysis shown in Table 3.

In the second flotation test, pyrite was depressed in the first stage and reactivated for the second stage flotation. The necessary reagents were also added in the first stage to float the free gold. Results of this test are shown as follows:

Unit	Weight Grams	Gold Ounces/Ton	Recovery %
Head	500.0	0.21	100.0
Conc. #1	32.4	1.84	58.9
Conc. #2	84.5	0.22	18.4
Tails	383.1	0.06	22.7

It is evident from this test that a higher gold recovery can be obtained by finer grinding and by the addition of reagents to float liberated gold. It is presumed that most of the gold may be associated with the copper bearing minerals in the first concentrate, but this is not conclusive. The calculated gold head is again consistant. These tests are encouraging, but are only preliminary in nature and must be followed by a comprehensive testing program to obtain the best results.

Sample A1-A8

These samples were taken above water line as approximate extensions of the off-shore sample lines. A composite screen analysis of these samples is shown in Table 5. In general, the material is coarser than the composite off-shore sample which is to be expected by the wave action along the shoreline.

As there appeared to be only a minimum of free sulfides in the size ranges coarser than 48 m[illegible] each sample was subjected to sink-float tests on the minus 48 mesh material only. The results of these tests are shown in Tables 6-13 inclusively.

TABLE 5

COMPOSITE SCREEN ANALYSIS, SAMPLES A1-8

Tyler Mesh		Weight	Weight	Cumulative Wt. %	
Passed	Retained	Grams	%	Retained	Passed
-	10	145.6	21.4	21.4	100.0
10	14	37.5	5.5	26.9	78.6
14	20	42.7	6.3	33.2	73.1
20	28	50.6	7.4	40.6	66.8
28	35	56.4	8.3	48.9	59.4
35	48	106.9	15.7	64.6	51.1
48	65	111.3	16.3	80.9	35.4
65	100	67.4	9.9	90.8	19.1
100	150	40.6	5.9	96.7	9.2
150	200	13.6	2.0	98.7	3.3
200	Pan	9.1	1.3	100.0	1.3
TOTALS		681.7	100.0		

TABLE 6

SINK-FLOAT ANALYSIS, SAMPLE A1

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % -48 Float	Total % -48 Sink
48/65	186.9	99.7	0.3	32.5	0.1
65/100	186.3	97.7	2.3	31.8	0.7
100/150	115.2	87.8	12.2	17.7	2.4
150/200	34.8	26.4	73.6	1.6	4.5
-200	49.2	0.6	99.4	0.1	8.6
TOTAL	572.4			83.7	16.3

TABLE 7

SINK-FLOAT ANALYSIS, SAMPLE A2

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % -48 Float	Total % -48 Sink
48/65	85.2	89.2	10.8	55.1	6.7
65/100	33.9	78.5	21.5	19.3	5.3
100/150	12.1	39.7	60.3	3.5	5.3
150/200	4.8	22.9	77.1	0.8	2.6
-200	1.9	78.9	21.1	1.1	0.3
TOTAL	137.9			79.8	20.2

TABLE 8

SINK-FLOAT ANALYSIS, SAMPLE A3

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % -48 Float	Total % -48 Sink
48/65	124.9	93.0	7.0	57.6	4.3
65/100	49.7	83.9	16.1	20.6	4.0
100/150	19.5	39.0	61.0	3.8	5.9
150/200	5.7	14.0	86.0	0.4	2.4
-200	2.1	52.4	47.6	0.5	0.5
TOTAL	201.9			82.9	17.1

TABLE 9

SINK-FLOAT ANALYSIS, SAMPLE A4

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % -48 Float	Total % -48 Sink
48/65	81.9	88.5	11.5	52.0	6.7
65/100	36.0	83.9	16.1	21.6	4.2
100/150	15.5	47.7	52.3	5.3	5.8
150/200	4.0	35.0	65.0	1.0	1.9
-200	2.1	76.2	23.8	1.1	0.4
TOTAL	139.5			81.0	19.0

TABLE 10

SINK-FLOAT ANALYSIS, SAMPLE A5

Tyler Mesh	Total Wt. Grams	Mesh % Float	Mesh % Sink	Total % -48 Float	Total % -48 Sink
48/65	139.4	63.3	36.7	24.2	14.0
65/100	91.4	40.8	59.2	10.2	14.8
100/150	89.8	9.4	90.6	2.3	22.4
150/200	36.9	1.9	98.1	0.2	9.9
-200	7.5	6.7	93.3	0.1	1.9
TOTAL	365.0			37.0	63.0

TABLE 11

SINK-FLOAT ANALYSIS, SAMPLE A6

48/65	178.7	90.3	9.7	51.7	5.5
65/100	74.3	68.1	31.9	16.2	7.6
100/150	40.8	19.9	80.1	2.6	10.5
150/200	14.0	4.3	95.7	0.2	4.3
-200	4.4	29.5	70.5	0.4	1.0
TOTAL	312.2			71.1	28.9

TABLE 12

SINK-FLOAT ANALYSIS, SAMPLE A7

48/65	61.4	75.2	24.8	34.3	11.3
65/100	42.0	61.7	38.3	19.2	11.9
100/150	23.5	37.4	62.6	6.5	10.9
150/200	5.9	20.3	79.7	0.9	3.5
-200	2.1	33.3	66.7	0.5	1.0
TOTAL	134.9			61.4	38.6

TABLE 13

SINK-FLOAT ANALYSIS, SAMPLE A8

48/65	32.3	93.8	6.2	41.6	2.7
65/100	25.8	93.4	6.6	33.1	2.3
100/150	8.7	71.3	28.7	8.5	3.4
150/200	2.9	65.5	34.5	2.6	1.4
-200	3.2	90.6	9.4	4.0	0.4
TOTAL	72.9			89.8	10.2

These data show the percent sink and float in each size fraction and the percent sink and float as a function of the total minus 48 mesh material. Those sulfides not liberated will, of course, not be included as a part of the sink fraction. The sink fractions, as determined by x-ray diffraction, were predominantly composed of pyrite, with smaller amounts of magnetite and enargite. It is noted that there is a larger percentage of sink material in samples A5 through A7 especially in the 48/150 mesh range. These samples were taken in the vicinity of the heavy media plant tailing disposal area.

Table 14 presents the analyses of the plus 48 mesh material, minus 48 mesh float, minus 48 mesh sink and a calculated head analysis on a weight percentage basis. Samples A1 through A4 show gold analyses of the head material to be in fair agreement with the composite off-shore sample. A higher analysis is noted in each case for the plus 48 mesh material indicating incomplete liberation, and some concentration by wave action. Samples A5 through A8 indicate anomalous gold values especially in the unliberated material of the plus 48 mesh sizes. This is probably a combination of concentration action along the shoreline and losses of sulfide minerals in the heavy media plant tailings. In all samples the gold appears to be correlated with the sulfide minerals and concentration will only be accomplished with complete liberation of the sulfides from the gangue material.

Conclusions and Recommendations

The preliminary work performed on the Chinguashih off-shore and on-shore tailing samples is only intended to determine the feasibility of conducting a more comprehensive program of investigation of this material. Results obtained in this work indicate that a potential exists for the recovery of gold, copper and pyrite as a source of sulfur. The actual recoveries to be expected and the economic parameters involved, however, can only be determined after a comprehensive testing program.

This testing procedure should include an adequate sampling program, grinding studies to determine the optimum liberation point of the sulfides and complete flotation studies. The latter should include both bulk and differential flotation, reagent selection, reagent concentration and cleaning circuits. As the tailing material shows evidence of incomplete liberation some consideration should be given to the present grinding and classification circuit in the Chinguashih mill.

TABLE 14

ANALYSES, SINK-FLOAT PRODUCTS, SAMPLES A1-A8

Sample No.	Sample Unit	Copper %	Silver Oz./Ton	Gold Oz./Ton	Sulfur %
A1	10/48 ROM	0.06	0.01	0.25	-
	-48 Float	0.03	-	0.11	3.20
	-48 Sink	0.38	0.07	0.64	40.80
	Head	0.08	0.01	0.41	9.33
A2	10/48 ROM	0.10	0.03	0.26	-
	-48 Float	0.05	0.01	0.10	4.80
	-48 Sink	0.50	0.09	0.08	42.70
	Head	0.11	0.03	0.21	12.46
A3	10/48 ROM	0.14	0.04	0.29	-
	-48 Float	0.05	0.01	0.04	5.50
	-48 Sink	0.54	0.09	0.06	45.20
	Head	0.14	0.03	0.22	12.29
A4	10/48 ROM	0.19	0.05	0.42	-
	-48 Float	0.04	0.01	0.04	3.70
	-48 Sink	0.58	0.09	0.07	46.70
	Head	0.18	0.05	0.36	11.83
A5	10/48 ROM	0.28	0.07	22.24	-
	-48 Float	0.04	0.01	0.04	4.20
	-48 Sink	0.71	0.10	0.30	46.00
	Head	0.38	0.07	10.03	30.53
A6	10/48 ROM	0.19	0.05	4.20	-
	-48 Float	0.07	0.02	0.12	8.20
	-48 Sink	0.61	0.09	0.28	41.50
	Head	0.20	0.05	2.69	17.82
A7	10/48 ROM	0.41	0.08	8.68	-
	-48 Float	0.07	0.02	0.05	5.00
	-48 Sink	0.76	0.10	0.50	43.60
	Head	0.39	0.07	6.43	19.90
A8	10/48 ROM	0.22	0.05	1.19	-
	-48 Float	0.09	0.03	0.04	3.50
	-48 Sink	1.33	0.11	1.67	41.00
	Head	0.22	0.05	1.07	7.36

I would recommend this study as a project suitable for the laboratory facilities of MRSO within the structure of the Industrial Technology Research Institute. The possibility of recovering gold at the present world price and the feasibility of reclaiming the pyrite for its sulfur content is in keeping with the purposes of ITRI for the conservation of mineral resources.

Hualien-Taitung Area

Introduction

In the initial stages of the heavy mineral program it was deemed prudent to confine our studies to the East coast of Taiwan because of the interest of the Chungshan Institute in the West coast area. Consequently, the coastal area from the Li Wu River south to Tungho was selected for investigation after a review of the available literature. A previous survey in this area conducted by Tsai Tong-Chien in 1956 indicated unusual mineral concentrations located at Chichi, Hsinshe, Paishonlien and Toulan.

Due to noted occurrences of gold along the beach at the mouth of the Li Wu Chi River, samples were taken at this location as well as at the major tributary streams entering the Li Wu Chi. These locations North of Hualien designated as samples C, D, E, F, G and H are indicated on Figure 1. Samples I, J, K, L, M and N, as indicated in Figure 2, were taken along the shore line where natural concentrations were evident.

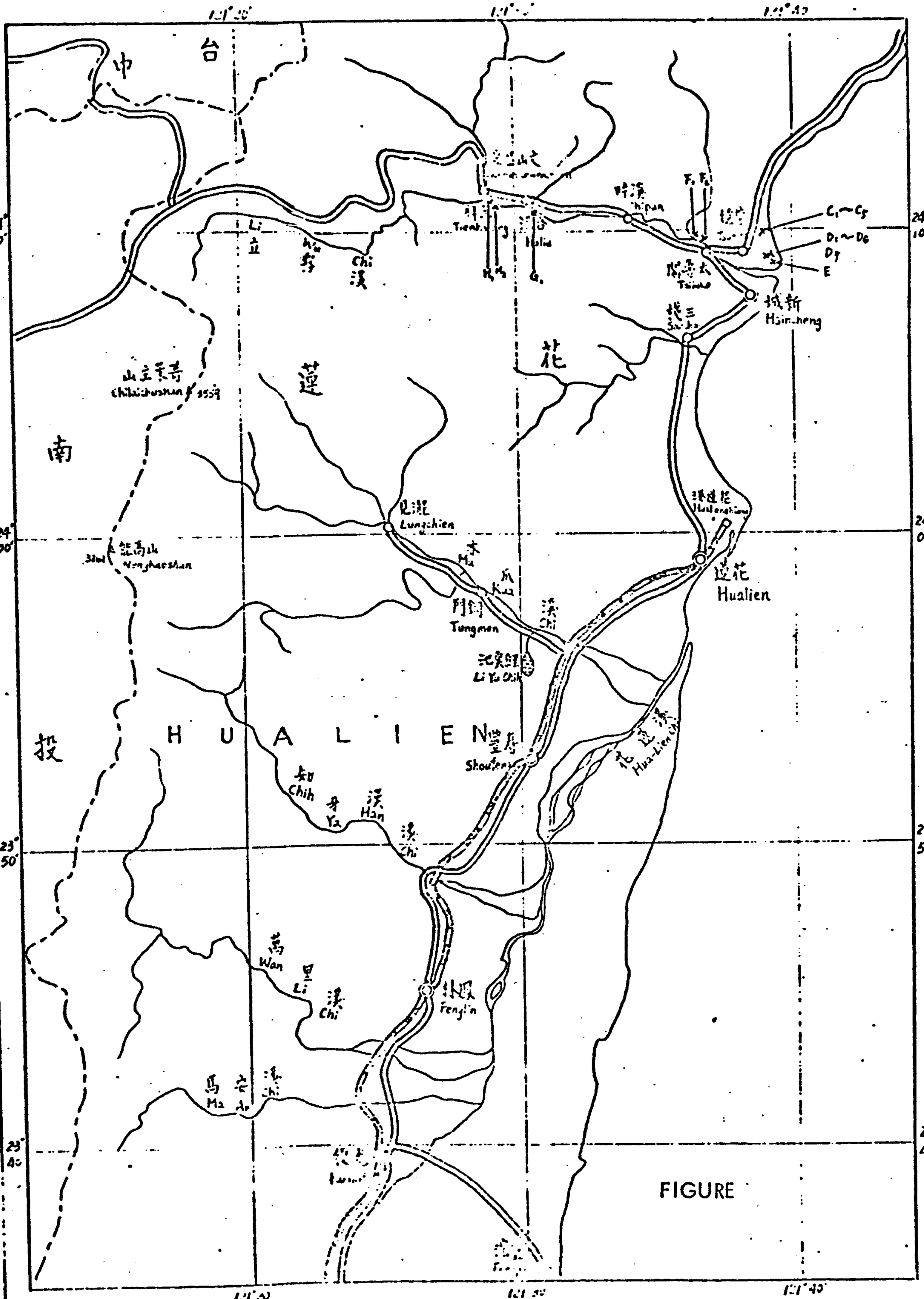
Those samples found to contain unusual amounts of magnetite were processed by a standard laboratory procedure consisting of size analysis, low-intensity magnetic separation, sink-float separation and high-intensity magnetic separations. This procedure is indicated in the flow diagram of Figure 3.

It should be stressed here that the samples taken were of a reconnaissance nature for the purpose of isolating areas where more detailed investigations might be warranted.

Samples C, D and E

Sample C was a composite of core samples taken along the shoreline just North of the entrance of the Li Wu River. Sporadic gold placer mining has occurred in this area usually after a typhoon has caused a concentration of heavy minerals along the beach.

As only minor amounts of magnetite and ilmenite were evident in this sample, it was not subjected to magnetic processing. However, several flotation tests were conducted on the sample to evaluate the free gold content. A average concentration



FIGURE

1:250,000

HUALIEN → TAITUNG　　東臺→蓮花

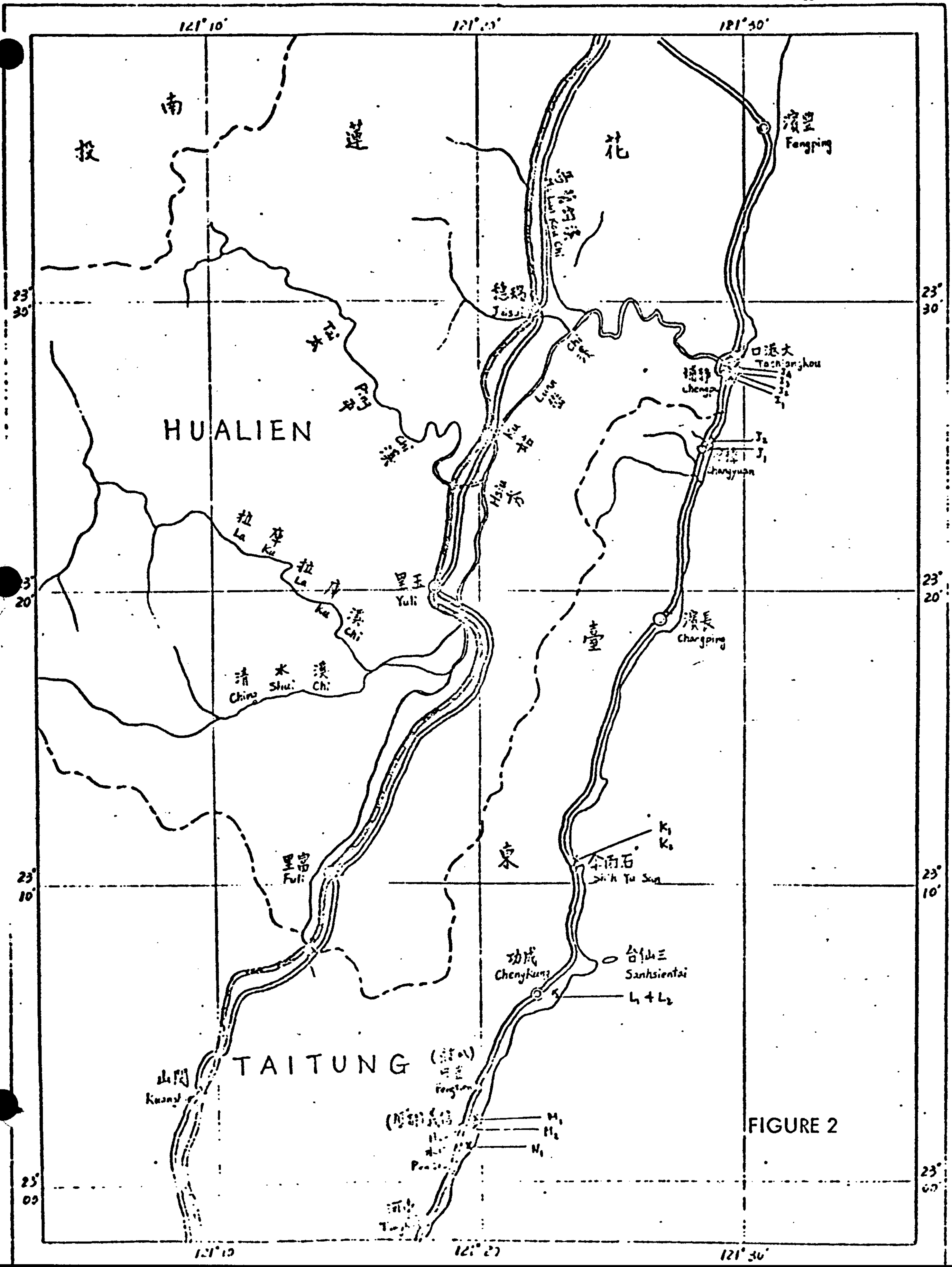


FIGURE 2

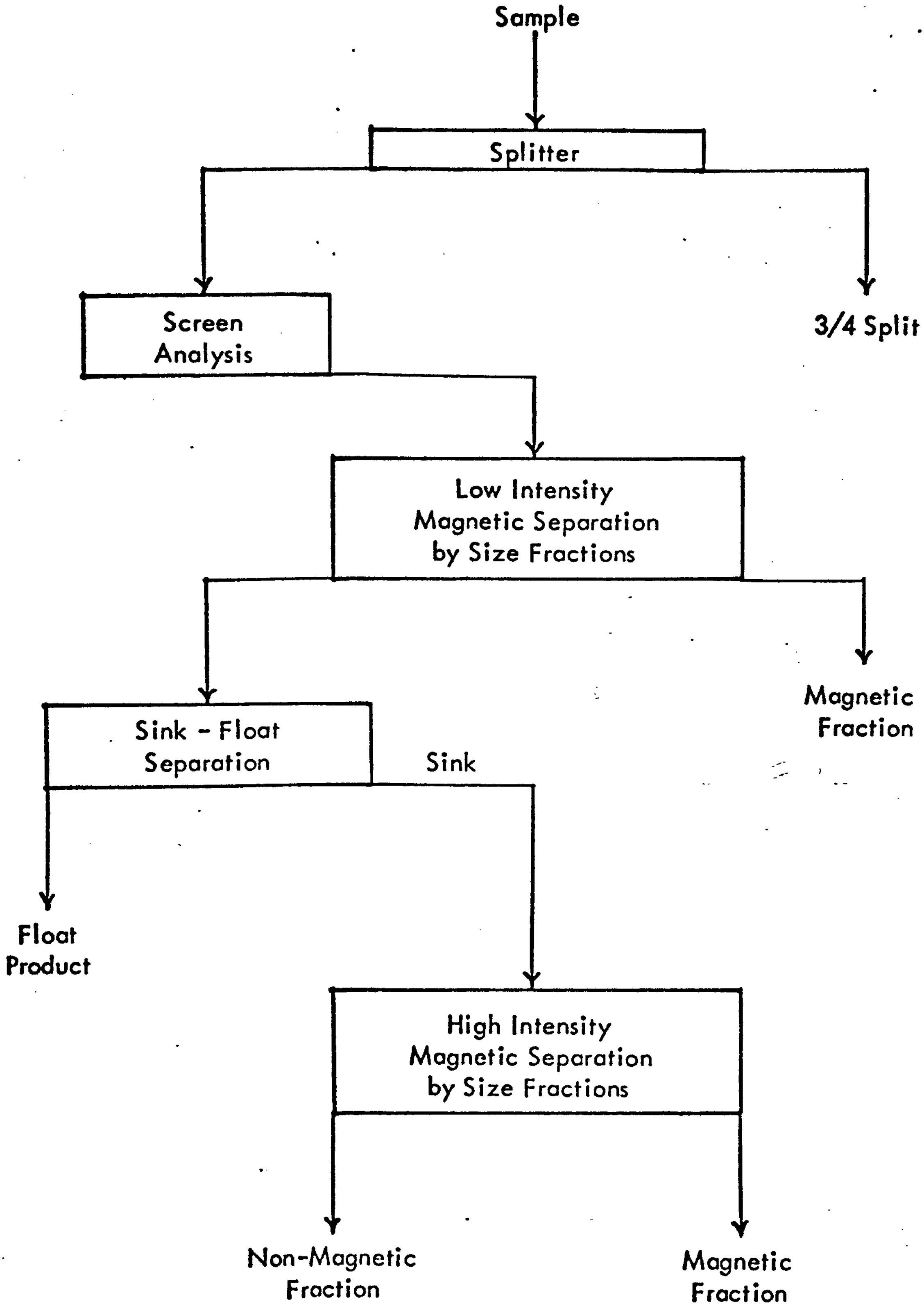


FIGURE 3

FLOWSHEET OF LABORATORY PROCEDURE

ratio of 84 was obtained in these tests with a concentrate grade of 0.22 ounces per ton. This gives a calculated head value of 0.003 ounces per ton for the minus 20 mesh sand.

Samples D1-D6 were of similar nature to Sample C, but obtained closer to the mouth of the Li Wu River. Flotation tests on this sample resulted in an average concentration ratio of 28 with a concentrate grade of 0.02 ounces per ton. A calculated head value of 0.001 ounces per ton for the minus 20 mesh sand is indicated.

Sample D7 was obtained from a local citizen and consisted of 530 grams of natural beach concentrate subsequent to a typhoon action. This sample consisted of 13.3% magnetite, 5.4% high-intensity magnetic material and 81.3% non-magnetics. The non-magnetic material of sample D7 was processes by flotation for evaluation of free gold. A concentration ratio of 11 was obtained with a concentrate garde of 0.89 ounces per ton. This gives a calculated head value of 0.08 ounces per ton for the non-magnetic fraction.

Sample E was taken in the same vicinity as samples C and D, but at the low tide range. No magnetite or other heavy minerals were noted in this sample and flotation tests for gold were negative.

Samples F, G and H

These samples were taken along the Li Wu River at the confluence of streams feeding into the left limit. No indications of gold or other heavy minerals were found in samples F and H, and only a trace of gold was noted in the flotation concentrate of sample G.

Samples I and J

Samples I and J were obtained from the shoreline south of the mouth of the Hsiu Ku Luan River. Sample I, from the vicinity of the river entrance, contained no significant concentration of heavy minerals, so was not processed beyond the low-intensity magnetic stage which yielded only trace amounts of magnetite.

Sample J, obtained from the vicinity of Changyuan, was subjected to magnetic and sink-float processes with the results shown in Table 15. Magnetite, comprising 9.1% of the sample was predominantly in the 48/150 mesh range. The sink fraction, excluding magnetite, consisted of 45% of the total sample. That portion of the sink

TABLE 15

MINERAL DISTRIBUTION DATA SAMPLE J

Tyler Mesh	Weight % Magnetite	Sink-Float Analysis			Wt. % Total Sample
		Sink Percent		Float Percent	
		Magnetic	Non-Magnetic		
+35	0.00	2.44	7.31	12.06	21.81
35/48	0.10	3.27	8.25	12.70	24.32
48/65	1.60	4.56	11.03	10.82	28.01
65/100	3.37	1.86	4.08	2.42	11.73
100/150	3.56	0.45	1.66	1.60	7.27
150/200	0.41	0.14	0.00	0.93	1.48
-200	0.07	0.00	0.00	5.31	5.38
Total or Average	9.11	12.72	32.33	45.84	100.00

TABLE 16

MINERAL DISTRIBUTION DATA SAMPLE K-1

Tyler Mesh	Weight % Magnetite	Sink-Float Analysis			Wt. % Total Sample
		Sink Percent		Float Percent	
		Magnetic	Non-Magnetic		
+35	0.02	0.01	0.07	0.52	0.62
35/48	0.07	0.08	0.58	1.08	1.81
48/65	2.48	5.66	26.85	9.50	44.49
65/100	8.08	6.50	27.18	2.39	44.15
100/150	3.81	0.44	1.56	0.04	5.85
150/200	1.55	0.41	0.78	0.12	2.86
-200	0.00	0.03	0.08	0.11	0.22
Total or Average	16.01	13.13	57.10	13.76	100.00

fraction which was magnetic at 0.5 amperes constituted 12.72% of the total and consisted of ilmenite and hypersthene. The non-magnetic fraction was predominantly hypersthene and diopside.

Samples K and L

Samples K and L were obtained from the shoreline in Taitung County in the vicinity of Shi Yu San and Cheng Kung respectively.

Mineral distribution data for sample K1 is shown in Table 16. These data show a magnetite recovery of 16.0% found mainly in the 48/200 mesh range. The magnetic portion of the sink fraction amounts to 13.13% of the total and is composed of ilmenite and hypersthene. The non-magnetic sink fraction consisting of hypersthene and diopside constitutes 57.1% of the total sample, and the float fraction is composed mainly of quartz and feldspar.

A pile of concentrated heavy minerals from a small scale mining operation was found near the beach at Shih Yu San. A sample of this material, which had evidently been reconcentrated by gravity methods, was selected and designated as K2. This material was found to consist of 63.1% magnetite, 28.7% high-intensity magnetic material (primarily ilmenite), and 8.2% non-magnetics consisting of hypersthene, diopside and quartz.

Sample L, obtained from a natural concentration, was processed with resulting data shown in Table 17. Magnetite was present in the amount of 17.19%, and the high-intensity magnetic fraction comprised 24.81% of the total sample. Mineral components of the different fractions, as determined by x-ray diffraction, were similar to sample K.

Samples M and N

Samples M and N were obtained along the shore line in the vicinity of Hsin I and Pen Shui respectively.

Sample M consists of two separate samples in which the mineral distribution data are shown in Tables 18 and 19. A much higher percentage of magnetite, high-intensity magnetic product and non-magnetic sink is indicated in Sample M2 (Table 19). X-ray diffraction analysis show both samples to be similar in mineral composition, however, with the high-intensity magnetic fraction composed of ilmenite and hypersthene,

TABLE 17

MINERAL DISTRIBUTION DATA SAMPLE "L"

Tyler Mesh	Weight % Magnetite	Sink-Float Analysis			Wt. % Total Sample
		Sink Percent		Float Percent	
		Magnetic	Non-Magnetic		
+20	0.04	0.08	0.45	9.22	9.79
20/28	0.34	1.86	4.24	18.22	24.73
28/35	4.68	13.07	11.84	9.13	38.72
35/48	5.25	6.03	2.18	0.79	14.25
48/65	5.90	3.41	0.66	0.28	10.25
65/100	0.93	0.31	0.09	0.13	1.46
100/150	0.01	0.01	0.00	0.03	0.05
150/200	0.03	0.03	0.02	0.11	0.19
-200	0.01	0.01	0.01	0.53	0.56
Total or Average	17.19	24.81	19.49	38.51	100.00

TABLE 18

MINERAL DISTRIBUTION DATA SAMPLE M-1

Tyler Mesh	Weight % Magnetite	Sink-Float Analysis			Wt. % Total Sample
		Sink Percent		Float Percent	
		Magnetic	Non-Magnetic		
+35	0.01	0.12	0.43	13.80	14.36
35/48	0.08	1.00	3.08	26.53	30.69
48/65	0.28	4.25	15.18	11.77	31.48
65/100	0.85	4.49	8.92	1.80	16.06
100/150	3.23	1.39	1.53	0.19	6.34
150/200	0.17	0.08	0.09	0.62	0.96
-200	0.00	0.00	0.00	0.11	0.11
Total or Average	4.62	11.33	29.23	54.82	100.00

TABLE 19

MINERAL DISTRIBUTION DATA SAMPLE M-2

Tyler Mesh	Weight % Magnetite	Sink-Float Analysis: Sink Percent, Magnetic	Sink Percent, Non-Magnetic	Float Percent	Wt. % Total Sample
+35	0.00	0.00	0.00	0.00	0.00
35/48	0.00	0.02	0.03	0.14	0.19
48/65	0.00	1.01	2.06	0.99	4.06
65/100	0.61	8.06	33.34	5.48	47.49
100/150	14.43	5.01	17.08	1.25	37.77
150/200	7.75	1.50	0.31	0.00	9.56
-200	0.79	0.12	0.02	0.00	0.93
Total or Average	23.58	15.72	52.84	7.86	100.00

TABLE 20

MINERAL DISTRIBUTION DATA SAMPLE N

Tyler Mesh	Weight % Magnetite	High Intensity Wt. %: Magnetic	High Intensity Wt. %: Non-Magnetic	Wt. % Total Sample
+35	0.00	0.63	13.39	14.02
35/48	0.00	1.94	36.71	38.65
48/65	0.07	1.64	28.39	30.10
65/100	0.12	1.48	7.92	9.52
100/150	0.39	1.30	4.61	6.30
150/200	0.31	0.15	0.36	0.82
-200	0.06	0.00	0.53	0.59
Total or Average	0.95	7.14	91.91	100.00

and the non-magnetic fraction consisting mainly of hypersthene and diopside.

Sample N, being of a much lower heavy mineral composition, was not subjected to sink-float tests. The magnetite, high-intensity magnetic and non-magnetic mineral distributions are shown in Table 20. The mineral components in this sample were similar to those of samples M1 and M2.

Conclusions and Recommendations

As the heavy mineral samples obtained along the shoreline in the Hualien-Taitung area were of a reconnaissance nature for the purpose of determining the types of minerals concentrated by wave and current action, any attempt to give a quantitative evaluation would be meaningless.

From the results of this survey and that conducted by Tsai Tong-Chien in August of 1956, areas of interest between Hualien and Taitung appear to be centered around Chichi and Cheng Kung. In both of these areas the heavy mineral fraction contains substantial amounts of magnetite and ilmenite. Although there are no significant amounts of other minerals of possible economic potential, further study may be warranted on the composition of the ilmenite fraction and possible use as a pigment material. Magnetite may find some use as a by-product but, unfortunately, no industrial use is presently known for hypersthene which constitutes the bulk of the heavy mineral fraction.

West Coast Area

Introduction

As mentioned previously, during the first year of the cooperative research program on detrital heavy minerals investigations were confined to the East Coast area of Taiwan where concentrations of magnetite and ilmenite were noted from a search of the literature. The West Coast area, although of much greater interest because of concentrations of zircon and monazite, was not investigated as it was under the jurisdication of the Institute of Nuclear Energy Research formorly the Chungshan Institute.

A review of the literature indicates that zircon and monazite may be found in virtually all streams flowing through the coastal plain of Taoyuan, Hsinchu and Miaoli counties of Northwestern Taiwan. The same is true for streams flowing to the Southwest coast in Yulin, Chiayi, Tainan, Kaohsiung and Pingtung counties. The greatest concentrations are noted on beaches, in sand dunes near the beaches and on off-shore bars as caused by the reconcentrating actions of waves and shore currents.

The minerals of primary interest in these deposits are monazite, zircon, magnetite and ilmenite (leucoxene). Associated minerals are usually garnet, hypersthene, augite, epidote, hornblende, rutile, tourmline and staurolite. Monazite probably offers the greatest economic potential as a source of the rare earth oxides for use in the growing industries of Taiwan. The presence of yellow monazite in these deposits has been known for a long period of time, but it has been only recently that a black variety has been recognized in the deposits along the Southwest Coast.

Through the cooperation of Mr. Chao Ching, an invitation was extended to visit the beach area of Northwest Taiwan and the pilot plant facilities of the Institute of Nuclear Energy Research. A tour of the latter facility, designed to separate the individual component minerals from the heavy sand concentrate on a pilot plant basis, was informative and impressive. Valuable work has been conducted by this group on the evaluation and processing techniques necessary for the exploitation of heavy mineral concentrations along the shore line. The assistance of this organization is appreciated and is of considerable help in the cooperative program.

Off-Shore Bars

Probably the most interesting area for investigation of heavy mineral concentrations is in the off-shore bars which roughly parallel the Southwest Coastline off Yulin, Chiayi and Tainan counties. Enriched concentrations of heavy minerals are known to exist on these bars which may be amenable to a mining method that emphasizes portability. This would be essential as the enriched zones caused by wave ans current action are erratic in both horizontal and vertical directions.

Some eleven off-shore bars in this area have been investigated to some degree for heavy mineral content, but those designated as Tungshanchou, Chingshankang-chou, Wangyehchou and Putaichou appear to offer the most promise from literature references. As these bars are continuously changing, due to wave and current actions, evaluation and study of their potential must be a continual process.

The black monazite content of the heavy mineral fraction probably offers the most significant economic value as a source of rare earth oxides for industrial use. The economic potential of zircon, ilmenite and magnetite should also be considered.

During my 1974 visit to Taiwan, with the assistance of the National Science Council, permission was obtained to visit several of the more promising off-shore bars. However, upon travelling to the designated area with Dr. Liu Hok-shing, Mr. Lee Ven-Chung and Mr. Lee Ming-I, we found that it would be necessary to obtain clearance through the local defense garrisons. Unfortunately, we were informed that the permit to obtain access to the off-shore bars had not had time to clear through the normal Department of Defense channels. As a time period of approximately two months was necessary for this procedure, we were forced to temporarily abandon this phase of the investigation.

Conclusions and Recommendations

The off-shore bars in Southwest Taiwan appear to contain the highest concentrations of heavy minerals. Explotation of these bars would probably depend upon a portable mining unit for the selective extraction of enriched areas. Monazite content of the heavy mineral fraction may be economically significant depending upon the ability [illegible] extract the rare earth oxides for industrial use. Consideration of the potential [illegible] for accessory minerals such as zircon, ilmenite and magnetite would also be important to the economic evaluation of the heavy sands.

Close cooperation and the exchange of information between the Institute of Nuclear Energy Research and the Industrial Technology Research Institute is urged in the effort to evaluate and exploit the monazite and other heavy mineral potential of off-shore and beach deposits.

MINERAL BENEFICIATION DEPARTMENT, M.R.S.O., I.T.R.I.

Introduction

At the time of my 1973 visit to Taiwan, the formation of three MOEA Institutes into one administrative unit was under consideration. This was finalized within the next few months as the Industrial Technology Research Institute. During the reorganization period the concept of a Mineral Processing Laboratory was recognized as a desirable capability for ITRI and the Republic of China.

Past work in the field of Mineral Processing had been conducted to some degree in each of the individual institutes which resulted in duplication of effort and facilities. Mr. Feng, Director of MRSO, requested my assistance in the planning and organization of one central facility as a consolidated effort for testing and research in the area of mineral beneficiation. An assessment was made of equipment availability and a preliminary report submitted to MRSO and the National Science Council regarding laboratory design, equipment acquisition and personnel requirements.

Present Status

The proposed laboratory has been initiated and located within the physical facilities of the Union Industry Research Institute at Hsinchu. My recommendation to offer employment to Mr. Ming-I Lee, a University of Alaska Master of Science graduate from Taiwan, was accepted and Mr. Lee has subsequently been placed in charge of the new department. This facility was visited with Mr. V. C. Lee, Deputy Director of MRSO, and Mr. Ming-I Lee for the purpose of making further recommendations on physical needs and project goals for this new department.

At the present time the laboratory is operating with a minimum amount of equipment necessary to accomplish the goals for which this unit was designed. Some equipment has been ordered, but full advantage has not been taken of equipment that was thought to be available for use from other research institutes. Probably the greatest obstacle is the inability to obtain analytical results in a reasonable length of time. Any tests conducted in this laboratory will depend on the ability to obtain rapid analytical data, consequently, the progress of research will be governed by this capability.

Therefore, I recommend that the laboratory be provided with its own atomic absorption unit for this purpose. X-ray equipment is present at the Union Industry Research Institute and should be made available to personnel of the Mineral Beneficiation Department.

The department is presently working on four projects concerned with the beneficiation of clays, pyrite, feldspar and heavy minerals. It would be desireable that each project be conducted by an individual knowledgeable in the field of mineral beneficiation to assure maximum progress in each area. Unfortunately, Mr. Lee is the only individual in the laboratory with this training and a good portion of his time is taken up with administrative matters. Availability of trained personnel who can work independently with a minimum amount of supervision and also have the capability of proposing new projects for study will be important to the future of this department. A University with undergraduate and graduate programs in Mineral Engineering offers a natural source to meet this requirement and a recommendation for this purpose is presented under the following heading.

Graduate Assistant Program

Communication and Cooperation between research institutes concerned with mineral development and the graduate program in the Mineral Engineering Department of the Cheng Kung University is at a low level. This is not to the best interest or goals of either entity. Although the remarks stated here will be confined to the field of mineral beneficiation, the general concept would hold true for all fields in the mineral industries.

The Mineral Beneficiation Department of M.R.S.O. is responsible for conducting research designed to up-grade raw materials into marketable products that can be utilized by industry. The capability to accomplish this is vested in the facilities and technical staff available to this department. The viability of this unit will depend, to a great degree, on the ability to generate ideas and projects which offer a good chance of success. If we consider the mineral resources of Taiwan and the limited technical staff available, it is a formidable task.

The Department of Mineral Engineering at the National Cheng Kung University has a graduate program designed to supply the trained professional people needed for the development of natural resources. However, this program cannot become effective without financial and technical cooperation from research institutes and industry. Graduate students need both a source of income and a research project that may be undertaken as a thesis topic. This research should be conducted at a level that will give practical experience to augment their theoretical education.

It is evident that each entity has something to offer the other on a cooperative basis. Guided, student conducted research can serve as a sounding board in determining the advisability of pursuing a project as a more comprehensive program within the facilities of M.R.S.O. This can be accomplished at a minimal cost to I.T.R.I. without budgeting large sums of money on projects that may be highly speculative in nature. The capability of generating useful projects is also enhanced by involving the professional staff and students at Cheng Kung University.

The following procedure is suggested:

1. I.T.R.I. through M.R.S.O. to provide three annual stipends for graduate student support. This would amount to $24,000 N.T. per student per year. An additional $11,000 N.T. per student is recommended for supplies and analytical support. A total of $70,000 N.T. per student over a two year period would then be required.
2. Professional staff of M.R.S.O. and Mineral Engineering Department of Cheng Kung University would suggest research topics for student consideration.
3. Problems encountered by industry should also be considered, in which case the industrial organization should be invited to share in the financial arrangements.
4. Conduct of the research to be guided primarily by University personnel with suggestions and recommendations by M.R.S.O. and industry staff members. Consequently, periodic progress reports should be submitted to the supporting agency to encourage these recommendations.

5. The completion of each research topic would require a thesis which would constitute a final report to M.R.S.O. and I.T.R.I. The thesis would be evaluated to determine the feasibility of continuing the investigation on a more comprehensive scale within the Mineral Beneficiation Department of M.R.S.O.

6. Upon completion of his graduate program the student should not be obligated to continued employment to the supporting agency. However, in some cases, it may be of benefit to offer employment for either continuing research or unusual capability of the student.

7. Staff members of M.R.S.O. should regard a cooperative arrangement of this nature as part of their professional responsibility to the research institute. University faculty members submitting proposals which have been accepted for study should benefit financially in accordance with University policies on such matters.

A program of the type discussed above would benefit all organizations concerned with mineral development and the training of competent people to assure that this development progresses. However, this will require that continued liaison be promoted between Industry, the Research Institutes and the University.

Survey of Mineral Resource Requirements

The Republic of China is going through a period of industrial growth, which to a large degree will be governed by the availability of metallic and non-metallic mineral commodities. Therefore, it is important that every effort be made to assess the availability of these resources and to conduct studies on the beneficiation of domestic and foreign raw materials necessary for present and future needs.

In the process of proposing research projects for the Mineral Beneficiation Department priority should be given to those commodities which are presently in short supply or will be in the near future. At the present time there is no comprehensive data source which indicates present or future industrial demands, how much of this demand is obtainable from domestic sources, specifications of the raw material, source of foreign raw material and cost of the impor[illegible] products.

It is recommended that a survey be made, with emphasis on industrial minerals, to determine the present, short term and long term industrial consumption of these commodities. This survey could be conducted either by mineral commodity or by individual industry demands. The latter method may be more expedient and significant as present and future demands can be correlated with the growth or decline of individual industries.

With the availability of this data, the Mineral Beneficiation Department and other Departments with M.R.S.O. could concentrate their research efforts on those mineral commodities deemed to be most critical for industrial growth.