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"PLACER MINING - JOBS FOR ALASKA"

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Compiled by
Mary Albanese and
Bruce Campbell

Front cover: *The Colorado Creek mammoth skull being wrapped in a plastic jacket in preparation for shipment to the UAF Museum. Photo courtesy University of Alaska Museum.*

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This publication is dedicated to the spirit and dogged determination of the Alaskan miner.

CONFERENCE PRESENTATIONS

Placer-mining operation near Livengood, Alaska.

This section includes a written summary of most of the papers presented at the Ninth Annual Placer conference, held on March 25-27, 1987, in Fairbanks, Alaska. Also included are summaries from selected addresses and preconference and postconference sessions submitted by various authors.

DEWATERING ALASKA PLACER EFFLUENTS WITH P E O: PRELIMINARY FIELD TEST RESULTS

by

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ABSTRACT

Field testing of the U.S. Bureau of Mines polyethylene oxide (PEO) dewatering technique on effluents from three placer mines in Alaska has shown significant variation. Feed to the unit ranged from 250 to 23,000 NTU; water recovered from the dewatering exhibited turbidities of 20 to 240 NTU. PEO requirements increased as the solids content of the feed slurries increased. Also, flocculations of the placer-mine effluent solids required a long contact time which was accomplished by in-line mixing of the PEO with the waste slurry.

INTRODUCTION

In placer mining, gold-bearing gravels are washed by placing the gravels into a trommel or on vibrating screens where the gravel is sized from 0.5 to 1 in. The undersized material is washed into a sluice box while the small rocks, sand, and fines flow off the end of the sluice box into a sump where most of the rocks and sand settle out. The water containing the fines and some sand flows out the sump and into the existing pond system at the mine site. The rocks and coarser particles are normally removed from the sump by a loader or dozer. In the pond system, the rest of the settleable material drops out, leaving fine-grained silts and clays or nonsettleable fraction. With time, more of the fine material will ultimately settle; the end result is a solution containing ultrafine or colloidal particles that will remain suspended indefinitely.

In the past few years the effluents from placer mining have received considerable attention from a variety of regulatory agencies, such as the U.S. Environmental Protection Agency (EPA), Alaska Department of Environmental Conservations (DEC), and the U.S. Department of the Interior Bureau of Land Management (BLM). EPA has proposed regulations pertaining to water quality, DEC has issued regulations setting a standard for water quality, and DEC has issued regulations setting a standard for water discharge of 5 NTU's above the background of the receiving stream. BLM is enforcing the reclamation standard on federal lands.

Thus, the miner is faced with many operating and economic factors. To meet water standards, a large pond will be required to allow settling of fines. To build such a pond, the miner must have a suitable site (which may not be possible) and must move the required dike material. Operation of a loader or dozer costs about $100/hr. Another cost in the requirement of a pond is that of reclamation. Because the pond will fill with fine solids that will not support land fill, reclamation is often difficult. Smaller ponds are easier to reclaim, but do not provide the retention time needed to produce water by natural settling that is acceptable under the regulations. Therefore, the miner is caught between regulations requiring low-turbidity water and the cost of reclamation.

The U.S. Bureau of Mines (USBM) added flocculant to the sluice discharge to accelerate settling. Tests conducted with the EPA this past summer have shown that a variety of commercially available flocculants produce floccules (flocs) that settle rapidly, resulting in waters containing 15 to 65 NTU. Because the flocculation process is rapid, a small pond may produce high-quality water. However, the pond will rapidly fill with solids, and new ponds will be needed during the operating season. Also, the pond containing flocculated material will have to be reclaimed.

For several years the USBM has been investigating a unique dewatering method. The technique consists of flocculating a waste slurry with PEO and dewatering the resulting flocs on screens (fig. 1). Depending on the waste being dewatered, the static screens may be used alone or in combination with a trommel. The flocculating agent, PEO, is a commercially available, water-soluble, nonionic molecule composed of about 100,000 repeating units of ($-CH_2-CH_2-O-$) with a molecular weight of 5 million.

This technique was tested on a variety of mineral wastes and includes laboratory, small-scale continuous, and field tests. The procedure has been applied to the clay wastes from phosphate, coal, bentonite, and potash operations and the tailings from uranium, talc, copper, gold, silica, and mica mines (Scheiner and others, 1985).

In 1981, the Tuscaloosa Research Center, at the request of the Bureau's Alaska Field Operations Center, conducted a limited number of laboratory studies involving the use of PEO for flocculating solids suspended in water samples from Alaskan placer gold mines. Results from these initial tests showed that the PEO dewatering technique had promise for treating effluents from placer mining. Subsequently, the Bureau conducted a second limited laboratory testing program to assess the state of the art since the original tests.

The technique was tested on 11 different samples, again with promising results. Based on these results, a field testing program was planned for the summer of 1986. This report describes the results obtained from the first three test sites. Results from the fourth site will be included in a report, to be published in 1987, covering the total test program.

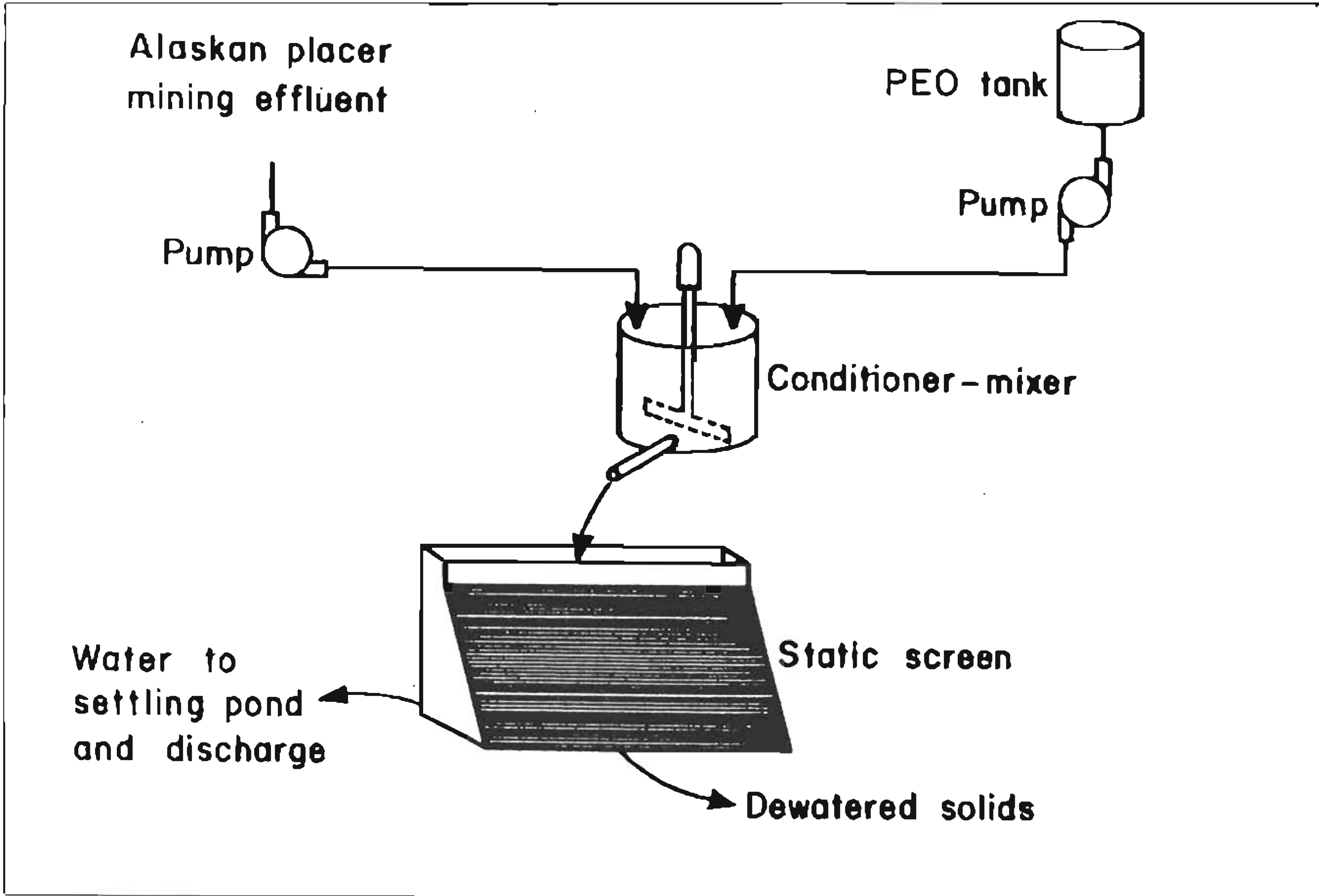

Figure 1. Flow sheet for PEO dewatering unit for Alaska placer effluents.

With tests planned at four sites in 12 wk, it was obvious that it would be difficult to optimize operation of the unit at each site. So the plan was to look at the following items at each mine: 1) polymer dosage, 2) water quality, 3) screen efficiency in retaining solids, and 4) the capacity of the screen.

DESCRIPTION OF TEST UNIT

The test unit (fig. 1) was mounted on a truck (fig. 2). Waste slurry is pumped to the conditioner mixer, where PEO is added. The water and resulting flocs flow out of the tank and are distributed on the screen. The flocculated solids move down the screen, while released water goes through the screen and flows to a pond for either recycling to the sluice or discharge to the stream. The solids are placed in a mine cut or pit where they continue to dewater, reaching a solids content high enough to allow burial or removal by a front-end loader. The static screen was built in two sections of stainless-steel wedge wire, each 8 ft wide by 4 ft long. The upper section of the screen had slot openings 2.75 in. long and 0.02 in. wide and was at an angle of 58°. The lower section of the screen had slot openings 2.75 in. long and 0.01 in. wide and was at an angle of 50°.

The flocculant is made up as a 0.25 percent solution by sprinkling the solid PEO powder through a water spray into a tank equipped with a stirrer. The 0.25 percent solution is diluted to the desired strength, usually 0.01 or 0.02 percent solution, for flocculation.

CROOKED CREEK

The first test site was at the Gelvin Mine, located on Crooked Creek near the town of Central, Alaska. At this mine, 90 to 100 yd^3 of material are treated per hour with a water usage of 1,000 gpm. The material exiting the sluice enters a sump, where most of the sand and gravel settle; they are removed periodically by a front-end loader. The waste stream flows into a large pond, where the rest of the settleables drop out. The nonsettleable slurry then flows into an extensive tundra filter and is finally recycled back to the sluice box. In other words, this mine was on total recycling of process water.

The waste slurry treated in the dewatering unit was fairly uniform, containing 0.8 to 1.0 percent solids (4,000 to 6,000 NTU). This slurry contained only nonsettleables. Water recovered from the waste stream flowed from the dewatering unit to a 54- by 93-ft pond about 3 ft deep. The overflow from this pond reentered the water system at the mine.

Initial tests showed that the nonsettleables could be removed readily from the slurry on the unit with a PEO dosage of 0.05 to 0.10 lb of PEO per 1,000 gal treated. The turbidity of the underflow water received from the screen was 200 to 240 NTU. At the end of the day and again in the morning before starting tests the turbidity of the water in the pond remained in this range. Extensive laboratory experiments also showed that the turbidity could not be lowered appreciably even with large amount of polymers (table 1).

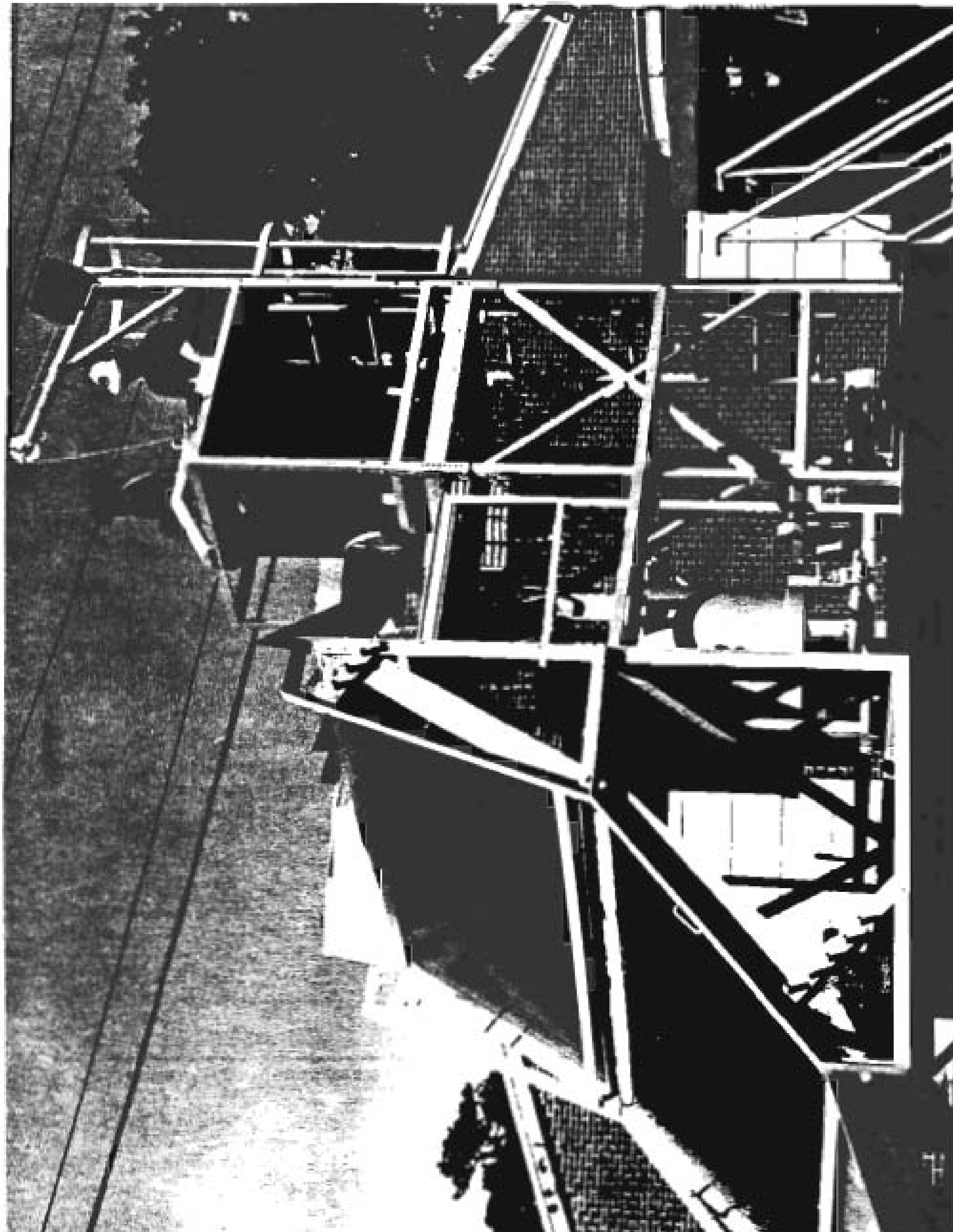

Figure 2. Mobile dewatering unit.

Table 1. Laboratory tests at Crooked Creek.

PEO ($lb/10^3$ gal)	Catfloc ($lb/10^3$ gal)	Turbidity after flocculation (NTU)
0.058	NA	215
0.026	NA	240
0.114	1.17	105
0.056	0.585	160
0.002	0.293	240
0.010	0.585	155
0.012	0.585	155
0.016	0.083	220
0.016	0.008	280

NA-Not applicable.

The tests were conducted with varying amounts of PEO and Catfloc T[1], a cationic polymer. This combination has been shown through previous laboratory tests on Alaskan placer effluents to produce water of good quality.

During these initial tests of the dewatering unit, additional contact time between the PEO and the slurry was required to obtain strong flocs. To accomplish this, a new conditioner mixer was designed and installed on the unit. Initial tests showed that the PEO dosage could be lowered from the previously obtained 0.05 to 0.03 lb of PEO per 1,000 gal of material treated. The capture of solids on the screen was 70 to 80 percent. The rest of the solids passed through the screen, but settled immediately in the water pond to produce water having a turbidity of 200 to 240 NTU. The above tests were conducted at a flow rate of 200 to 300 gpm.

One economic factor in the capital cost of the unit is the screen capacity. To determine the screen capacity, the treatment flow rate was increased from the 200 to 300 gpm to 450 to 500 gpm. The flow pattern in the trough that distributed the flocculated material from the mixer conditioner onto the screen appeared to cause considerable shear action in which flocs were broken.

A new trough was designed, but because of the test schedule for the summer and the delay in construction, it was not installed until testing was completed at Fairbanks Creek.

The results at Crooked Creek showed that the PEO dewatering system worked, but mixing of the waste slurry with PEO was critical, and the method of mixing and distribution of flocculated material onto the screen was far from optimal.

[1]Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

FAIRBANKS CREEK

The second test site was the Cook Mine, located on Fairbanks Creek. At this mine, 50 to 70 yd^3 are treated per hour. Material is moved with a drag line, fed into a trommel to remove rocks, and then fed to a sluice through a hydraulic lift. Water usage is about 1,200 gpm. Waste slurry flows from the sluice box to a pond where most of settleables drop out. Water is taken from this pond and recycled back to the sluice box. Overflow from the pond flows into a second pond, where more settling occurs. Overflow from the second pond flows to several other ponds at the edge of the mine property before being discharged into an overburden drain system.

The dewatering unit was installed below the second pond. A 60- by 60-ft water pond with an average depth of 3 to 4 ft was constructed and used for impounding of water produced during the flocculation process. This pond overflows back into the mine's water system. At the Cook Mine, bedrock drainage of water under the dam of the second pond occurred. This water filled the Bureau's pond at the beginning of the testing program and continued to dilute the pond during the testing program.

During the first week of testing at Fairbanks Creek, the sluice box was being moved and the only water available for testing was that in the second pond. This water exhibited turbidities of 200 to 300 NTU. Initial tests with the conditioner mixer showed that mixing was insufficient and only small flocs could be formed and they could not be retained on the screen. A 16- by 18-mesh wire screen added on top of the first section of wedge wire screen increased solids capture but not the quality of floc formation.

Laboratory tests showed that when the addition of PEO was followed by vigorous mixing, the flocs consolidated and settled quickly, leaving a fairly clear supernatant. The PEO was added to the pump line between the pump taking water from the second pond and the dewatering unit. This increase in turbulence and retention time produced large flocs that could be dewatered on the screen.

As additional lengths of hose were added, the floc size increased and solids capture increased dramatically. Thus, considerable time was spent optimizing the turbulence and retention time required for maximum floc formation. (For ease in discussion, both effects are referred to as retention time.) Different retention times were required for different concentrations of PEO; thus, the conditioner mixer could be abandoned. Seventy to 80 sec gave optimum results for 0.01 percent PEO solutions, whereas 60 to 70 sec was near optimum for 0.02 percent PEO. With 0.05 percent PEO, the retention time required was near 10 sec. However, longer PEO retention requirements were observed when 0.01 percent PEO was used.

During the testing program, placer effluents with turbidities ranging from 150 to 3,100 NTU were treated in the dewatering unit. The higher turbidity waters were obtained from the miner's first pond (the same pond used for recycled water to the sluice). Results indicated that PEO consumption increased as the solids content of the placer effluent increased. Also, screen captures of 98 percent or greater were obtained if enough PEO was used. The water produced by flocculation had turbidities in the 30- to 40-NTU range for tests where optimum conditions are used.

PEO dosage of 0.01 lb per 1,000 gal produced high screen capture and high water quality for placer effluents with turbidities of 1,000 NTU or less. When the turbidity was in the 2,500- to 3,000-NTU range, the PEO dosage required for 98 percent capture was 0.045 lb per 1,000 gal. If the PEO dosage is lowered to 0.03 lb per 1,000 gal, the capture falls off to 70 percent and the water has a turbidity of 130 NTU. However, the solids going through the screen still settle rapidly, and after settling overnight the water in the pond was 28 NTU.

Experiments to determine the capacity of the screen indicated that if well-flocculated material was screened, the capacity was greater than 700 gpm.

During the last days of testing, the combination of PEO and Catfloc T was tested. Catfloc T was shown in laboratory test to have a synergistic effect with PEO. Data from these tests indicated that the use of PEO in combination with Catfloc has the potential of increasing the clarity of water and lowering the cost of polymer required for dewatering as well. Because of the tight schedule, however, the effect of Catfloc was not optimized.

OLIVE CREEK

The third test site was at the Geraghty Mine, located on Olive Creek near Livengood. At this mine, 60 yd^3 of gravel per hour are sluiced using 1,000 gpm of water. This mine had a series of settling ponds and recycled all its water, which is usually in short supply on Olive Creek. The pond system consisted of a primary pond, where most of the sand and gravel settled out. This pond flowed into a second, 4-ft-deep pond, 165 by 125 ft, where some settling of fines occurred. Water from this pond is pumped to a third pond, which is used to supply water to the sluice box. This elaborate pond system has been used by the Geraghty's for the past 6 to 8 yr.

The test unit was set up at the second pond. Waste slurry was taken near the inlet to the second pond. Clean water produced by flocculation was put back into the second pond near the pumps used for pumping water to the third pond. Dewatered solids were discharged from the screen and placed on a hillside adjacent to the unit.

Initial dewatering tests were conducted using the conditioner mixer. Results indicated that at a dosage of 0.042 lb of PEO per 1,000 gal, water with a turbidity of 88 NTU could be produced with a screen capture of 90 percent. Lowering the dosage to 0.031 lb per 1,000 gal resulted in water having a turbidity of 195 NTU and a screen capture of only 34 percent. Thus, the conditioner mixer would not provide the retention time needed to obtain high-quality water at low dosages.

Mixing the PEO with the waste slurry in the line used to carry the slurry from the pond to the dewatering unit was also tested. Initial tests showed that better quality water at lower PEO dosages could be obtained (table 2).

Table 2. Effect of PEO addition on unit performance at Olive Creek.

Feed (% solids)	Feed water turbidity	Screen H_2O (NTU)	PEO dosage ($lb/10^3$ gal)	Screen capture (%)
0.545	2,000	25	0.016	99
0.444	1,550	35	0.016	98.6
0.526	1,900	44	0.010	98.3
0.378	1,450	62	0.008	4.2
0.403	1,600	245	0.004	10.0

The data indicate that at a dosage of 0.01 of PEO per 1,000 gal, high screen capture was obtained while producing water with a turbidity of 44 NTU. Note that the feed solids ranged from 0.378 to 0.545 percent. This variation was observed throughout the testing program. In fact, raising or lowering the intake hose in the pond produced a wide range of solids contact. This allowed an investigation of feeds with turbidities ranging from 350 to 23,000 NTU.

During the study, factors investigated included PEO concentration (0.01 and 0.02 percent), mixing time, and the effect of feed concentration of the PEO dosage. Experiments were conducted to obtain greater than 90 percent screen capture. Data indicate that the optimum mixing time was between 60 and 70 sec. A comparison of 0.01 and 0.02 percent PEO solutions is shown in table 3.

Table 3. Effect of PEO concentration on dewatering efficiency.

Feed (% solids)	Water turbidity (NTU)	PEO dosage ($lb/10^3$ gal)	Capture (%)	PEO conc. (%)
1.05	33	0.026	77.4	0.02
0.93	60	0.034	98.8	0.02
1.01	32	0.016	80.5	0.01
1.09	18	0.023	99.4	0.01

In terms of water quality and dosage requirements, the 0.01 percent PEO concentration performed better than the 0.02 percent PEO concentration. For screen-capture percentage in the high 90s and for this range of feed solids, the 0.02 percent PEO concentration requires a dosage near 0.03 lb per 1,000 gal, whereas for 0.01 percent PEO, the dosage requirement is only 0.02 lb per 1,000 gal.

During the program, 99 experiments were conducted using various PEO concentrations, PEO dosages, and mixing times; table 4 shows the PEO dosage required for various feed concentrations to obtain greater than 90 percent screen-capture water with a turbidity of 20 to 25 NTU.

Table 4. PEO dosage requirements for various feed turbidities at Olive Creek.

Feed turbidity (NTU)	PEO dosage ($lb/10^3$ gal)
350 - 1,500	0.01
1,500 - 2,500	0.01 - 0.012
5,000 - 7,000	0.02
10,000	0.03

As at Fairbanks Creek, a 16- by 18- mesh screen was laid on the top section of wedge wire screen. This screen functioned very well, resulting in good quality water and high screen capture. Lack of time precluded further testing.

The screen capacity for feed containing less than 3,000 NTU was estimated to be in excess of 750 gpm; for concentrated feed, the capacity was estimated at about 500 gpm.

DESIGN OF OPERATING PLANT

The data indicate that the flow sheet for removal of solids from placer effluents has to be altered. In fact, the use of in-line mixing has simplified the system considerably, as seen in figure 3.

The dewatering plant consists of a pump to deliver the waste slurry to the unit. The capacity of pump will depend on the amount of slurry to be treated. This is the major cost for the dewatering plant. The pump for injecting the PEO has to have a variable-speed drive and be of the positive displacement type. Depending on the PEO required and the PEO concentration, the capacity of this pump could vary from 12 to 36 gpm. The diameter of the pipe to deliver the slurry to the dewatering unit would depend on the size of the slurry pump. The other consideration would be the 60- to 70-sec mixing time that is required for optimum results. The screen would be constructed with wire screen, 16 by 18 mesh, which worked well in field testing. The screen setup would also contain a system for collecting the water coming through the screen and a device such as a chute to carry the solids from the screen to the disposal site.

The PEO solution is made in a mixer that consists of a tank equipped with a stirrer, a vibrating feeder to add the PEO, and a water spray system to wet the PEO particles as they fall in the tank. Making up the 0.25 or 0.30 percent PEO solution usually takes 30 to 60 min. The concentrated solution is then diluted to either 0.01 or 0.02 percent. The size of the tanks required if PEO was made up once a day is shown in table 5.

Of course, concentrated PEO could be made up for several days' operations and dilution done when required. The time for diluting the concentrated solution would depend on the flow capacity of the pump.

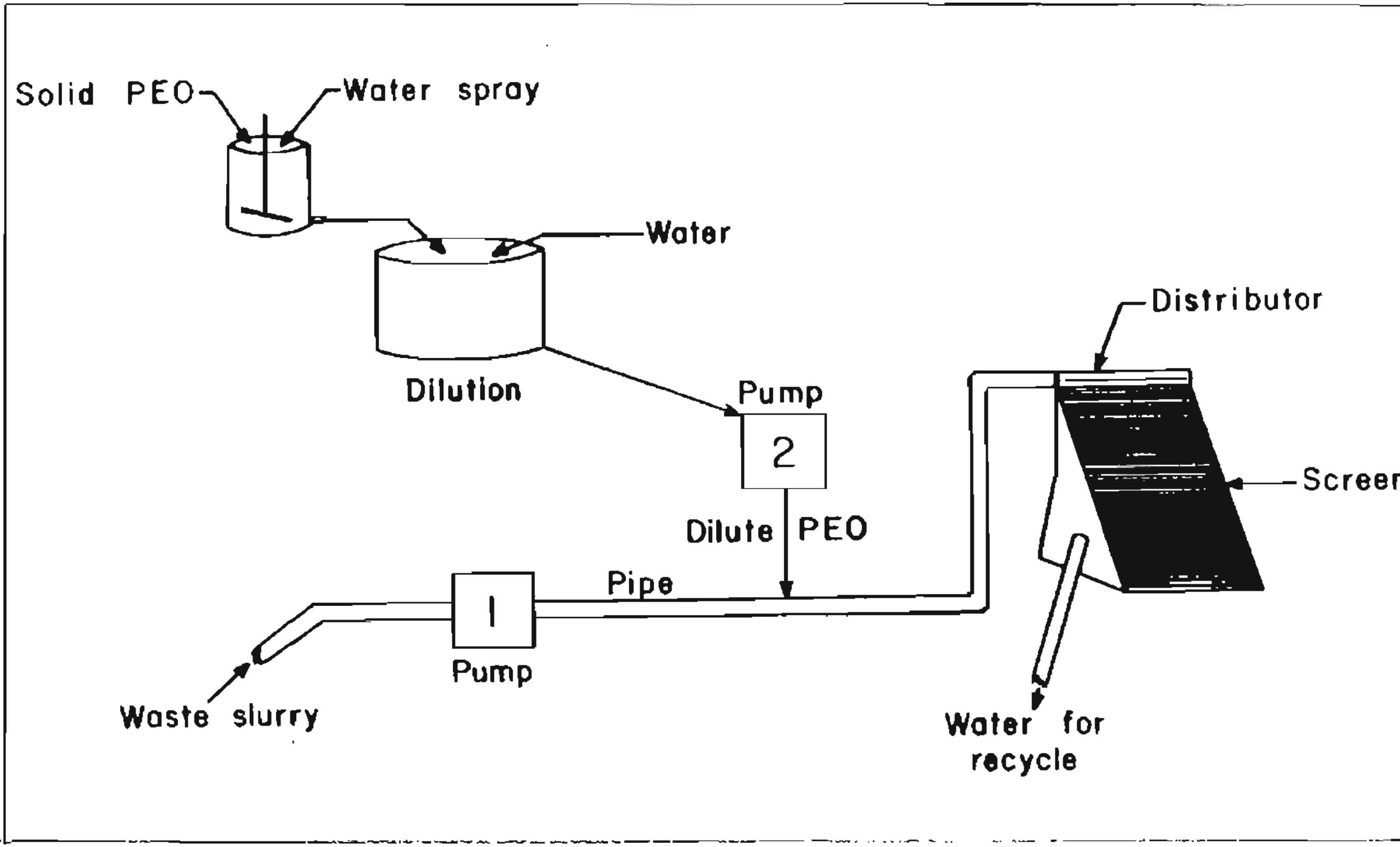

Figure 3. Proposed flow sheet for treating placer-mine effluents.

Table 5. Quantities of PEO solution needed for 1,000-gpm feed.

Dosage required ($lb/10^3 gal$)	PEO/8-hr/day (gal)		Dilute PEO/8-hr/day (gal)	
	0.25 %	0.30 %	0.01 %	0.02 %
0.01	231	193	5,775	2,888
0.02	462	386	11,550	5,776
0.03	693	579	17,315	8,664

CONCLUSIONS

Field testing has shown that the response of the effluents from each mine site varied significantly. At Crooked Creek the best water quality obtained exhibited 200 to 240 NTU, whereas 20 to 40 NTU water consistently was obtained at Fairbanks Creek and Olive Creek.

The time of contact between the effluent and PEO required to form strong flocs that dewatered easily on the screen was shown to be 60 to 80 sec for 0.01 and 0.02 percent PEO solutions. This was accomplished by using in-line mixing, which eliminated the need for the conditioner mixer.

The PEO dosage requirements were shown to be dependent on the feed solids' concentration. For slurries with turbidities around 1,000 NTU, the PEO dosage was 0.01 lb per 1,000 gal treated. At 2,500 NTU, the dosage varied between 0.01 and 0.045 lb per 1,000 gal, depending on the mine site. High retention of solids on the screen was obtained at Fairbanks Creek and Olive Creek, where in-line mixing was used. At Crooked Creek, the capture of solids was less, but probably could have been improved with in-line mixing.

The testing program showed that placer effluents are amenable to solids removal by flocculating with PEO followed by screening. Whether the technique can be used economically will have to be decided on a mine-by-mine basis.

REFERENCE CITED

Scheiner, B.J., Smelley, A.G., and Stanley, D.A., 1985, Dewatering of mineral waste using the flocculant polyethylene oxide: U.S. Bureau of Mines Bulletin 681, 18 p.

THE ROLE OF THE UNIVERSITY OF ALASKA-FAIRBANKS IN PLACER-MINING EDUCATION AND RESEARCH

by

Donald J. Cook, Dean
School of Mineral Engineering

Under one name or another, the present School of Mineral Engineering has been a component part of the University of Alaska since its inception as the Alaska Agricultural College and School of Mines.

Throughout this 64-yr period, this unit has trained professionals, performed service functions, and conducted research in support of a viable mineral industry in Alaska. Because our graduates perform a function as suppliers of natural-resource commodities, our welfare will also be closely allied with the well-being of the industries we represent.

To illustrate this interdependence and the cyclic nature of the industry, I refer you to figures 1 and 2, which depict the status of gold production in dollars and troy ounces, respectively, over an 85-yr period. Comparing these data with the School of Mines organization chart for 1939, a peak year of gold production (fig. 3), it is evident that our position as an educational unit follows the same cyclic path. Thus, it behooves us within our financial capability to assist the industry wherever possible.

The University of Alaska is charged with responsibilities for teaching, research, and public service in support of the general welfare of the state. The present School of Mineral Engineering has the same responsibilities, as it concerns the well-being of the state's mineral industry (fig. 4).

Through our graduate programs, the Mineral Industry Research Laboratory (MIRL), and Mining Extension, we visualize opportunities to develop a closer alliance with the placer-mining industry in fulfilling our research and public service obligations and to be of technical assistance to the industry.

Placer miners often generate innovative ideas that they may not have the time or the finances to test or the media available to transmit them to operating colleagues. On the other hand, although financial support is limited, the School of Mineral Engineering has the faculty and students available to test these ideas in areas of expertise that encompass mineral exploration, mining methods, rock mechanics, ventilation, slope stability, mineral beneficiation, hydrometallurgy, mineral economics, frozen ground, and environmental concerns.

MINING AND MINERAL RESOURCES RESEARCH INSTITUTE

Public Law 95-87, Title III, established the Mineral Institute program administered by the U.S. Department of the Interior. The University of Alaska was one of 31 schools nationwide qualified to receive this designation in 1978.

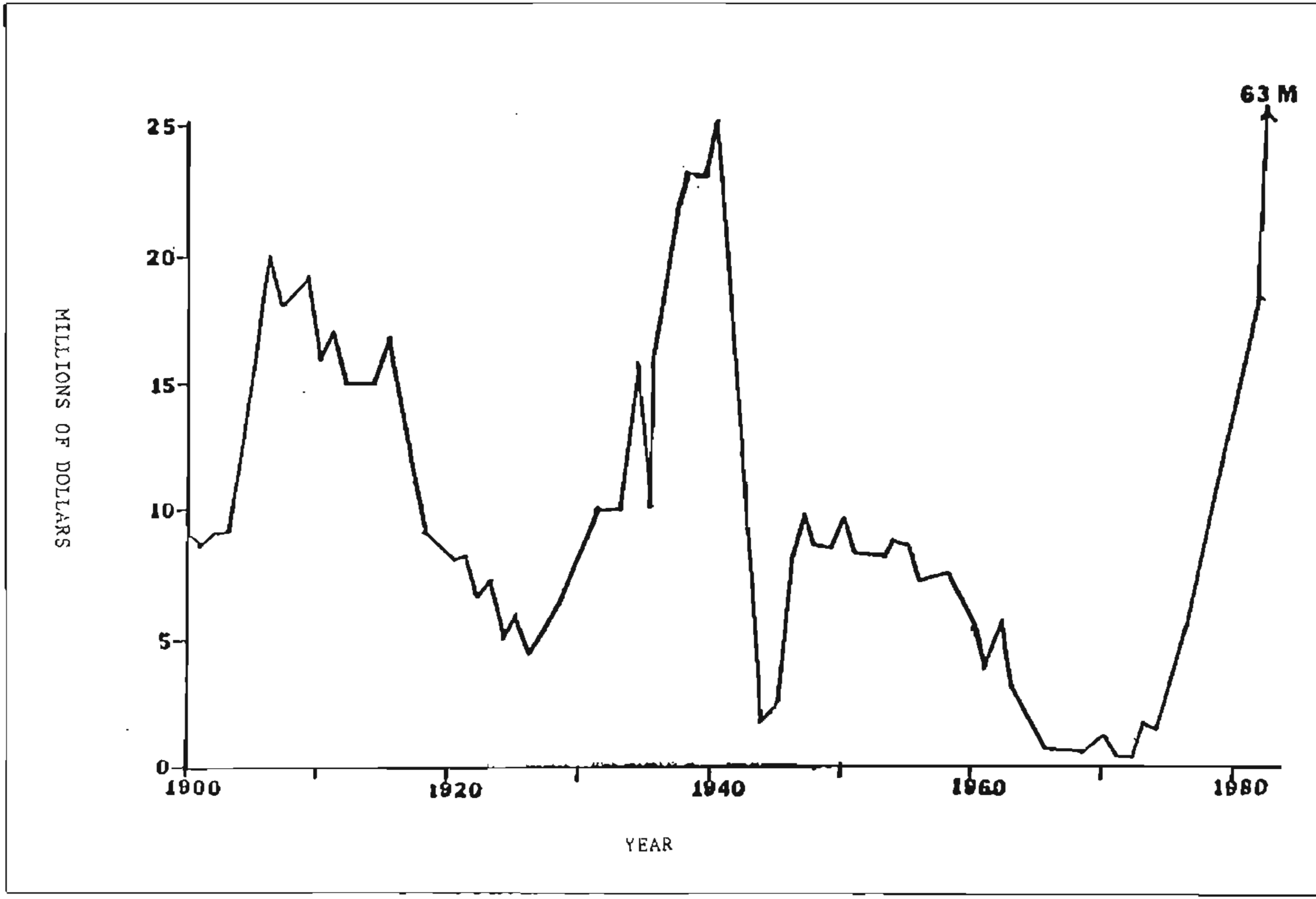

Figure 1. Alaska gold production in millions of dollars, 1900-85.

Figure 2. Alaska gold production in troy ounces, 1900-85.

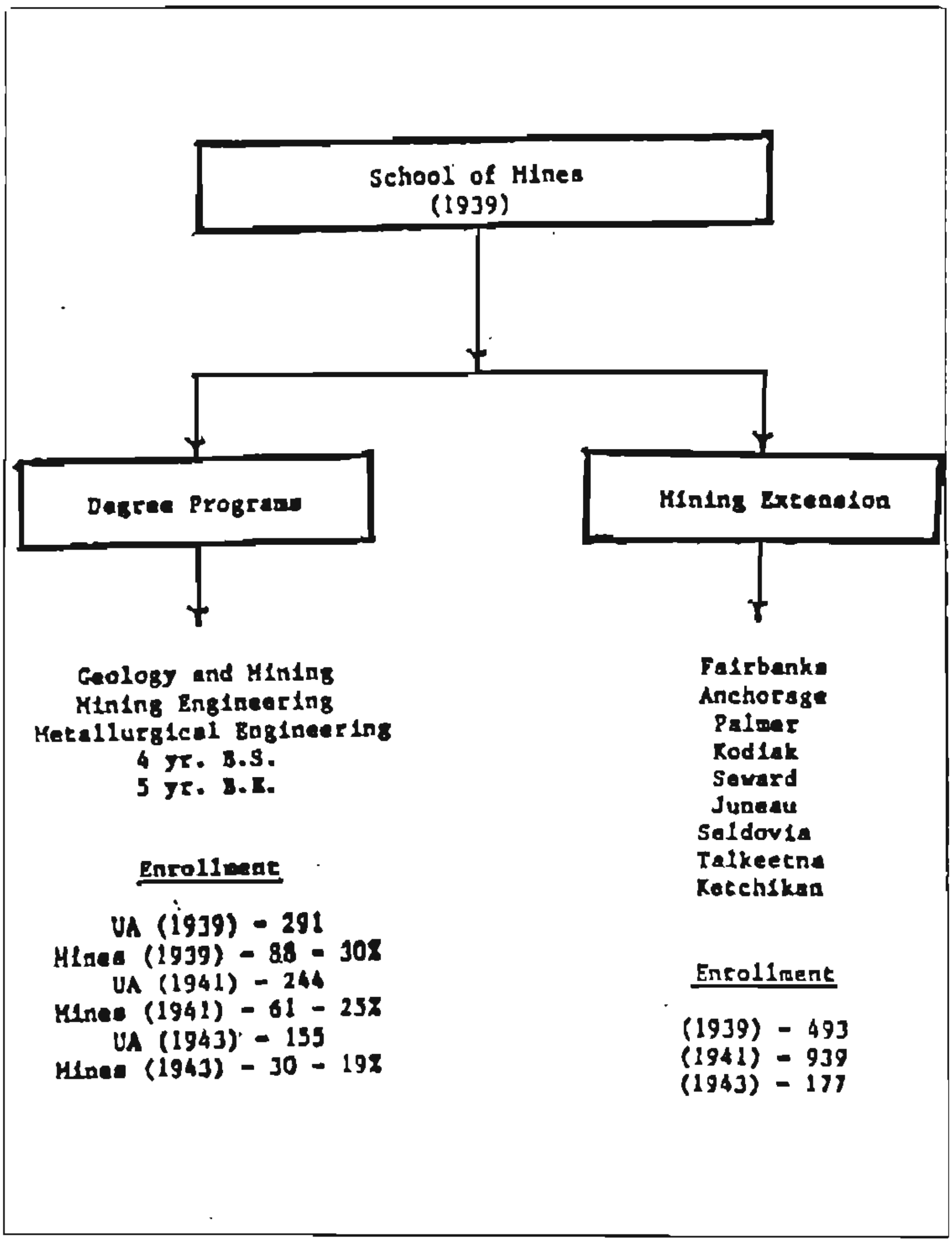

Figure 3. School of Mines organization chart, 1939 (a peak year for gold production).

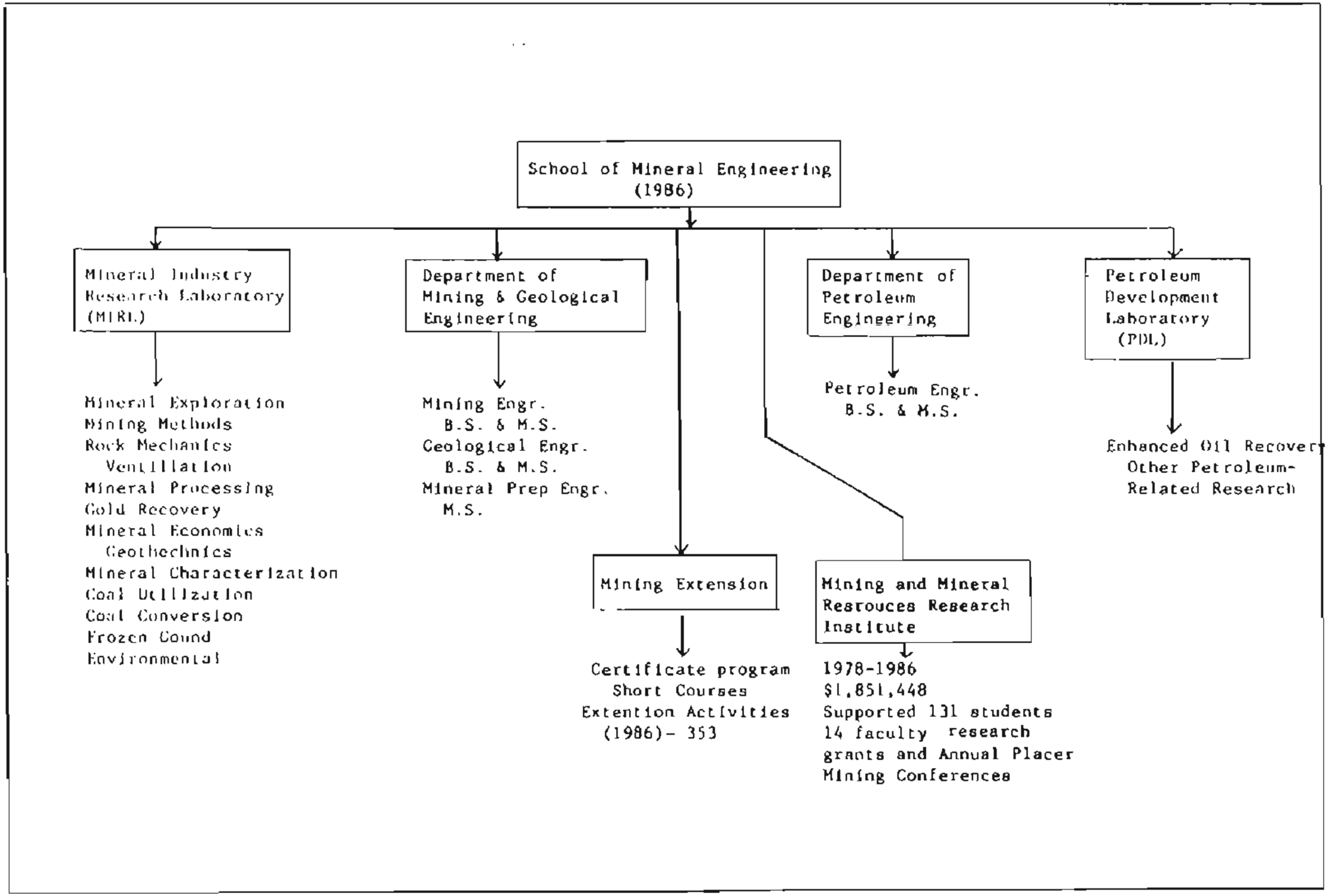

Figure 4. School of Mineral Engineering organization chart, 1986.

The grants awarded under this program fall under the two broad categories of basic allotments and research grants. During the period 1978-86, $1,339,000 has been awarded to the School of Mineral Engineering under the basic allotment program and $512,450 under the research program.

The basic allotment has supported faculty research, and 131 students, primarily at the graduate level. In addition, seven Placer Mining Conferences have been sponsored, in part, by these funds.

In recognition of the problems facing the placer-mining industry, we have allocated some of these funds over the past 2 yr in support of faculty and graduate student research. Two graduate students have been conducting flocculation studies; their results are reported in this volume (p. 27).

A faculty research grant was also awarded to Daniel Walsh at MIRL, which also employed a graduate student to study the benefits of using a spiral classifier-hydrocyclone system to reduce solid content of sluice-box discharge at the GHD Resources Company's operation at Eagle Creek.

We will continue to allocate funds for placer-mining research and are open to suggestions of study areas that would be beneficial to your interests.

MINERAL INDUSTRY RESEARCH LABORATORY

In recognition of the need for a research facility in conjunction with mining-related instructional activities, the Mineral Industry Research Laboratory was created in 1963 by the Alaska State Legislature. Mining and geologically related research activities are administered through this unit, which also publishes research results.

Some of these activities relative to placer mining have been presented at past conferences. Two current projects involve placer miners who were recipients of the State Placer Grant Program. Papers by Dr. Frank Skudrzyk and graduate student David Ziegler regarding a grant on underground placer mining at Wilbur Creek in the Livengood area awarded to Stan and Rose Rybachek are included in this volume. Dr. P.D. Rao and Daniel Walsh have completed a study on using a static screen, jigs, spirals, and a compound water cyclone in conjunction with a fine-gold recovery grant awarded to Harold Ellingson of EVCO.

For many years, the Territory and State of Alaska furnished a service to Alaskan residents interested in prospecting. This service, supplied through assay offices at College, Nome, Anchorage, and Ketchikan, provided free analysis, testing, and consultation to individuals seeking information. Unfortunately, these facilities have all been closed, leaving a general public with no point of contact for this type of assistance.

The Mineral Industry Research Laboratory has complete and modern laboratory facilities to conduct analyses and research on all phases of the mineral industry. Public-service functions are also supplied, but these are limited by finances and personnel available.

The State Division of Geological and Geophysical Surveys (DGGS) currently occupies space adjacent to MIRL and also has some equipment installed and jointly used by University personnel in space assigned to MIRL. Combining MIRL and DGGS equipment and facilities to provide analytical services to the general public along lines similar to the past has been proposed.

MINING EXTENSION

The Mining Extension program provides a primary community service in addition to the service functions expected of individual faculty members. This program has been associated with the school since its inception as the School of Mines and, over the past six decades, has provided a needed community service for individuals interested in aspects of the mineral industry.

Two instructors are employed in the present program which, through short courses in allied subjects, offers a 2-yr certificate in mineral exploration. To be more responsive to the needs of the industry and to act as a liaison between industry and the school, we plan that more emphasis will be paced on field extension activities and less on class instruction.

The change in emphasis in this program will be initiated this coming summer. Daniel Walsh and Jim Madonna will spend time in the field as extension agents to determine how the school can be of assistance to placer miners through the facilities available at the University.

With the recent establishment of a separate Division of Mining in the state Department of Natural Resources, we see an opportunity to cooperate with this unit in assisting the placer-mining industry. We envision our role as one of engineering assistance and technology dissemination through the personnel and facilities available at the University. These functions, in conjunction with the regulatory responsibilities of the Division of Mining, can be compatible, and we are currently discussing specific areas for cooperative efforts.

In conclusion, the School of Mineral Engineering desires to have an increased involvement in research and public-service activities in support of placer mining. We solicit your ideas on how we can be of assistance in this effort.

THE ALASKA MINERAL RESOURCE KIT

by

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This paper is sponsored by AMEREF, the Alaska Minerals and Energy Resource Education Fund. The State of Alaska and the mining and oil industries are collaboratively bringing minerals education to the public schools. The Minerals Kit was created by teachers, miners, and geologists, and then pilot-tested in the schools and revised. Initiated with money from the Legislature, AMEREF, through donations, sponsored the completion, assembly, and distribution of the kit as well as the current training involved with the curriculum.

What events could inspire youth to choose mining as a career? The AMEREF board members were asked what events inspired them to choose mining as a career. A variety of responses resulted:

-"Geological museum and display of minerals & rocks"
-"Opportunity for travel ("to 'see the world by studying the earth itself'")"
-"Exposure to the industry in my younger years"
-"High-school studies in physics and energy"
-"Books (e.g., Richard Hallisberstar's 'Marvels')"
-"Visiting ghost towns on family travels"
-"Career day in high school when told 'If you like the out-of-doors and math you would like geological engineering.....'"
-"Interest in natural resources and discovering a manganese deposit when I was 19"
-"I was 12 years old before I knew there was anything besides mining."

Though our objective is not to make all students miners, we are interested in tapping those positive experiences.

The goal of the Minerals Kit is to provide students with the knowledge, attitudes, and skills to make informed decisions regarding the mineral resources of Alaska. The kit does that with four teaching modules and support materials, including mineral samples (fig. 1), posters, filmstrips, videotapes, and reference books.

The Alaska Resources Kit's minerals curriculum is now helping teach between 12,000 and 15,000 Alaska students valuable lessons about resource management, thanks to joint efforts of industry and the Alaska Departments of Education and Commerce and Economic Development.

Over 400 resource kits were distributed to more than 200 schools statewide in the first phase of the project, which began 4 yr ago. This year

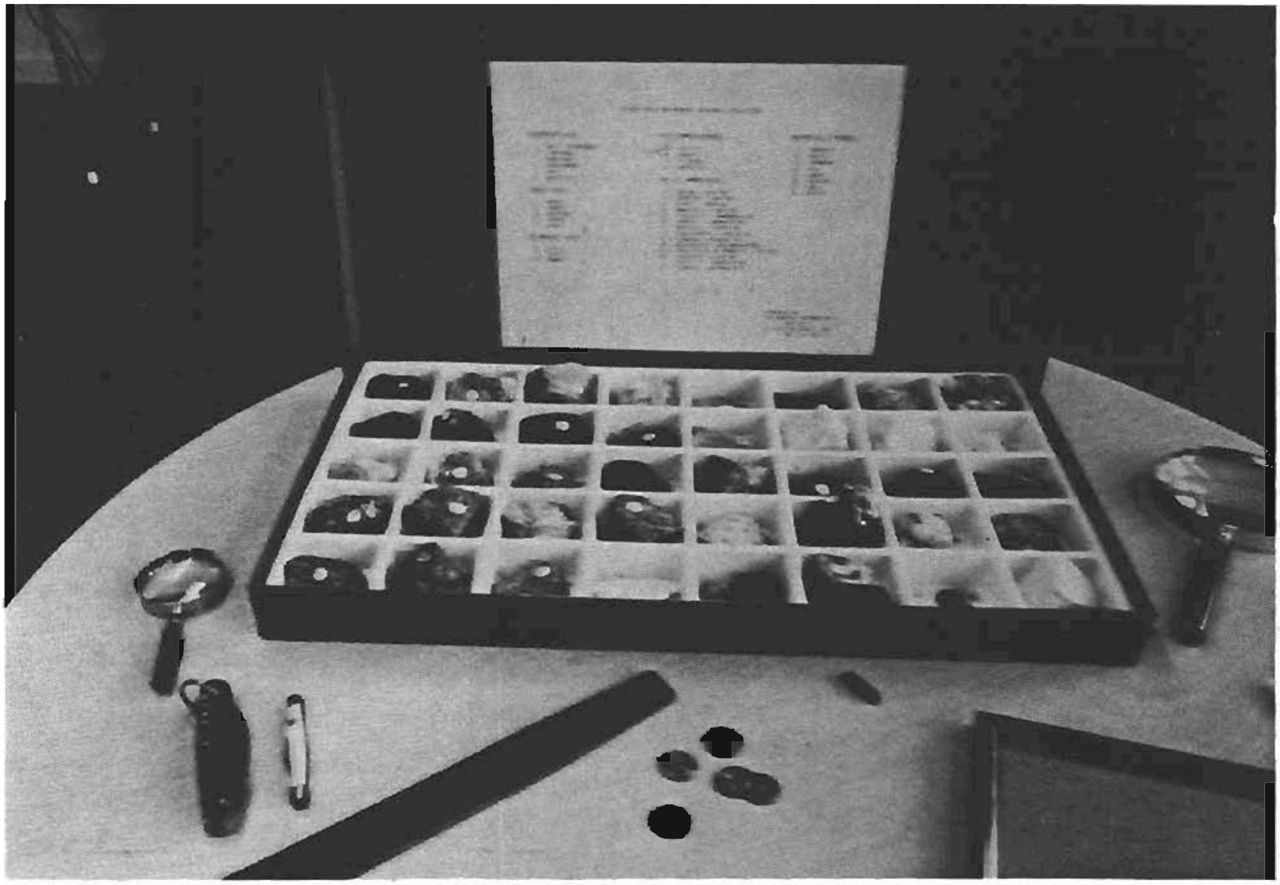

Figure 1. Mineral samples in the Alaska Minerals Resource Kit.

a second phase of the project is working to develop community participation and partnerships between education and industry.

This year's model communities---Anchorage, Fairbanks, Iditarod, and Petersburg school districts---are working to integrate the kit into their district programs and increase cooperation between education and industry at the local level. Four target communities--Alaska Gateway, Juneau, Mat-Su, and Nome--are next in line to receive the support program.

Former teacher Judith Entwife is working with district representatives to correlate Minerals Kit materials with goals in each district's science and social studies curriculums. In addition, the department will:

- Present in-district sessions to train teachers in using the Minerals Kit
- Help districts develop contacts with local mineral professionals and create lists of potential speakers and field-trip sites for classes
- Establish short courses in minerals education for graduate credit.

Through newsletters and audioconferences, model communities share lesson plans, keep up on local minerals education activities, update community contacts, and exchange feedback on kit components and ways to use them.

These activities are funded by a $25,000 grant to the Department of Education from the Alaska Minerals and Energy Resources Education Fund, an organization of mining and oil industry executives under the leadership of John Blackwell of Anchorage. AMEREF board members include:

Vigo Anderson
NC Machinery Co.

Earl H. Beistline
Mining Consultant

Robert Bettisworth
Former Alaska Representative

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Mead Treadwell
Yukon Pacific Corp.

Joseph Usibelli
Usibelli Coal Mine, Inc.

Curt McVee
Executive Director
Alaska Miners Association

Emma Walton
Alaska Science Teachers
Association, ex-officio

To date, the Alaska Mineral project has been supported by $400,000 in contributions from industry and an initial $150,000 from the State of Alaska.

Miners interested in participating in the next target community phase of integrating minerals education with community goals and involvement are invited to contact Peggy Cowan at the Department of Education Instructional Center, P.O. Box F, Juneau, Alaska 99811, ph. 465-2841.

APPLICATION OF POLYMERS IN THE REMOVAL OF SUSPENDED SOLIDS IN PLACER-MINING EFFLUENTS

by

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ABSTRACT

Effluent waters from two placer mines on creeks with significantly different characteristics were tested to determine the applicability of polymers in reducing turbidity. The waters from a Gilmore Creek mine and a Crooked Creek mine were characterized both as to their chemical and physical properties. The inorganic minerals were usually identified, and the principal cations were identified. The fine solids were studied as to their mineral identification and extent of their electrical charges.

Tests were run to determine optimum mounts of polyethylene oxide (PEO) and a liquid cationic polymer (with additives) in yielding minimum consumption and optimum settling efficiency. A treatment-plant design is proposed.

INTRODUCTION

The placer-mining industry in Alaska is faced with increasingly stringent specifications for the water quality of its effluent. Inevitably, this will entail the use of increasingly sophisticated technology to achieve the desired levels of suspended solids and turbidity. The use of settling ponds has become common in surface mines in Alaska. This technique will eliminate up to 90 percent of the coarser suspended solids. However, the residual ultrafine suspended solids are unaffected with conventionally sized settling ponds, even when placed in series. This paper deals with the application of various coagulants to further reduce turbidity levels and suspended solids levels in a 'polishing' step following conventional settling techniques. Previous work has been done by many researchers on the application of inorganic and organic flocculants; these range from well-known coagulants such as starch and alum to polymeric settling aids. The principal coagulants tested in this study were polyethylene oxide (PEO) and a liquid cationic polymer. The PEO was secured from three different suppliers (two domestic); the cationic polymer was provided by one American manufacturer.

CHARACTERIZATION OF MINE WATERS

At the Gilmore Creek operation, samples were drawn from the effluent of ponds 1-5 and from a site several miles downstream. Only two samples were drawn from the Crooked Creek operation, one from the first pond and one from the second settling pond. These effluents were tested for pH, solids content, specific gravity, settling in the Imhoff standard test, and residual turbidity in NTU (table 1).

Table 1. General characteristics of samples (A1 to A6 from Gilmore Creek area, B1 and B2 from Crooked Creek area).

Sample	pH	Solid content (%)	Specific gravity	Imhoff cone test (ml/L)	Residual turbidity (NTU)
A1	6.6	3.23	1.028	0.7	30,000
A2	6.6	1.35	1.008	<0.1	15,600
A3	6.7	0.96	1.006	<0.1	12,000
A4	6.9	0.72	1.005	<0.1	8,900
A5	6.8	0.66	1.004	<0.1	8,100
A6	6.7	0.33	1.001	<0.1	2,700
B1	7.3	1.90	1.007	0.4	3,200
B2	7.4	1.53	1.004	<0.1	2,800

pH Measurement

A pH of approximately 7, or near neutral, is required for normal aquatic growth and fish survival. Changes in pH may also affect the solution of trace metals in water (Hammond and Mueller, 1979). Chang (1979) noted that the pH value of the Livengood and Harrison areas ranged from neutral to slightly basic (7.6 to 8.6), whereas in this study, the pH values ranged from slightly acidic to neutral range (6.6 to 7.3). Generally, the mining does not have substantial adverse effect on pH value in water downstream. The results of the pH tests at Gilmore Creek and Crooked Creek areas are listed in table 1.

Solids Content and Specific-gravity Measurements

From the results of this study, the solid content was shown to decrease continuously from sample A1 at 3.23 percent to sample A6 at 0.32 percent. However, the rate of solid-content decrease lessens after sample A2. For the samples from Crooked Creek, there was a substantial difference in solid content. These data demonstrate that settling ponds are only effective for certain size particles and are thus unable to reduce turbidity to the required standard.

Also, the Crooked Creek samples have lower solid content than the samples from Gilmore Creek, which might have affected flocculant performance.

Particle-size Distribution

In the Gilmore Creek samples, almost all the particles were less than 1 micron in diam (fig. 1). Figure 2 shows the size distribution for the Crooked Creek samples. The results show that settling ponds can have an effect on the sedimentation behavior of particles less than 10 micron diam

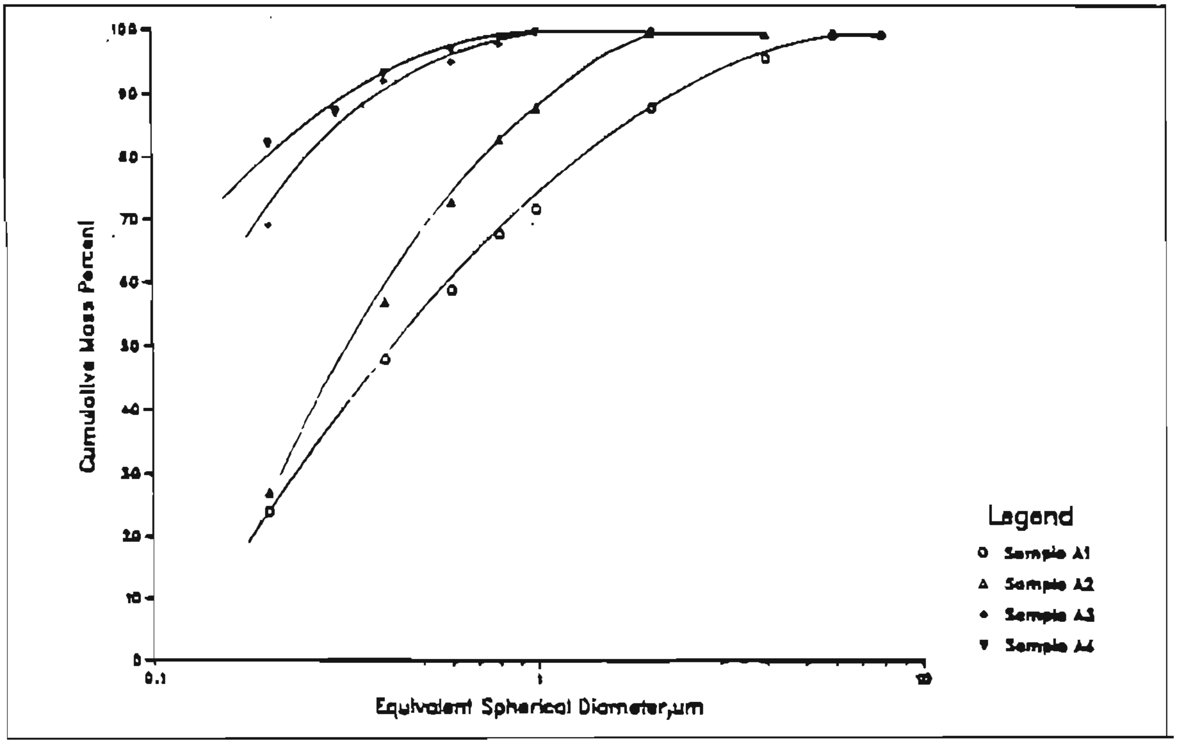

Figure 1. Equivalent diameter vs cumulative mass percent, Gilmore Creek samples.

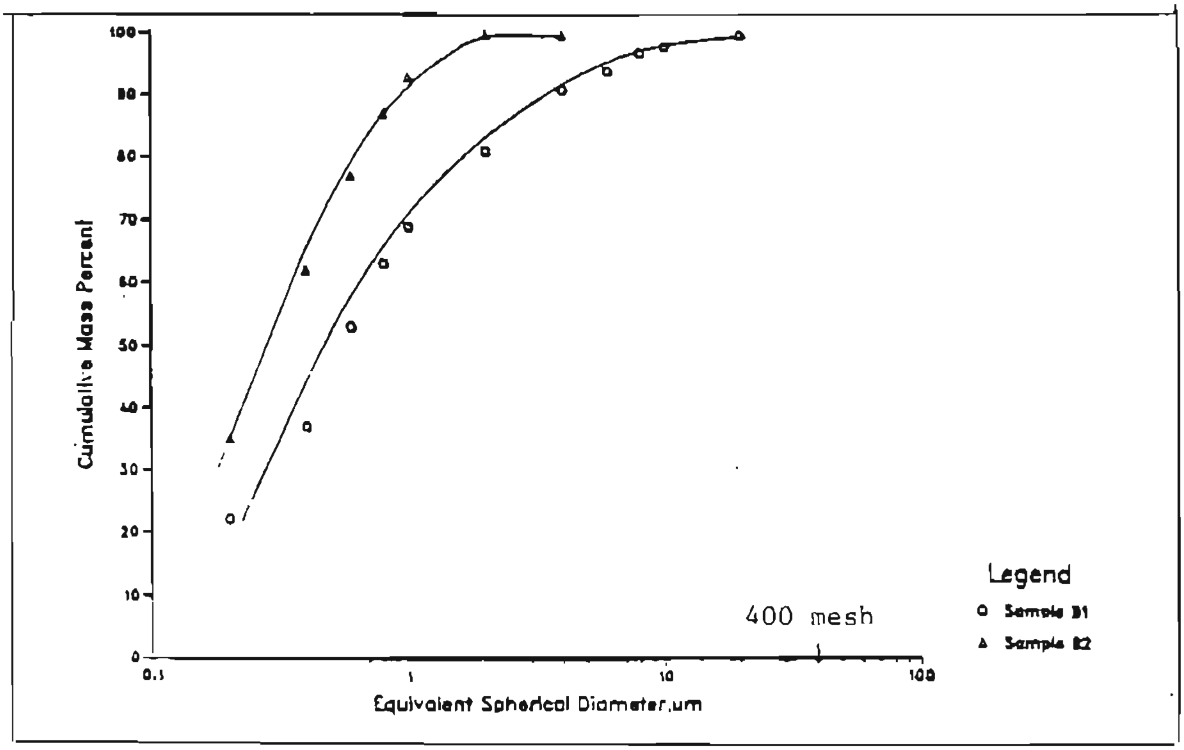

Figure 2. Equivalent diameter vs cumulative mass percent, Crooked Creek samples.

if the residence time has not been taken into account. Also, the Crooked Creek samples contain more very fine particles than the Gilmore Creek samples. For instance, sample B2, taken at the end of the first settling pond, contained 93 percent of particles less than 1 micron diam, whereas sample A1, located at the outflow of the first pond at Gilmore Creek, contained only 73 percent.

Imhoff Cone Test

In addition to solid content and size distribution, an Imhoff cone test was used to help understand the sedimentation situation.

From the Imhoff cone data in table 1, the difficulty of fine-particle settling is easily apparent. Even from sample A1 and B1, the settleable volume is still less than 1 ml/l for a 1-hr settling time. Fine particles have difficulty in settling, which explains why most settling ponds are unable to reduce turbidity to the required standard.

Turbidity Measurement

Turbidity is an expression of the optical property that causes light to be scattered or absorbed rather than transmitted in straight lines through the water sample. It is therefore a measure of light transmission and not a measure of total suspended solids. However, turbidity is used as a surrogate measurement in determining sediment concentration because it is convenient. Results of this study illustrate that higher levels of solid content will cause higher turbidity for samples from the same area.

Zeta Potential

Although zeta potential has been subject to criticism for its ill-defined character, it is still a reference in determining stability of fine particles in dispersion. Riddick (1968) indicated that zeta potential values less than -14 mv always represent the onset of agglomeration, whereas those from -14 to -30 mv indicate a plateau region of slight coagulation.

The zeta potential measurements for the Gilmore Creek and Crooked Creek samples were -16 and -21 mv (samples A1 and B2), respectively. Within this range, the particles may contact each other automatically and slightly coagulate.

Mineralogical Analysis

In his semiquantitative X-ray analysis, Maneval (1985) discovered a great deal of amorphous matter in the samples collected from Interior Alaska placer mines. Common clay minerals such as illite and kaolinite were also identified in several samples. However, in this study, the only clay mineral found in samples from both creeks was kaolinite. Because the diffraction intensity was quite low and the peaks were obscure, it can be deduced that amorphous material is predominant in both samples.

Chemical Composition Analysis

Because of isomorphous substitution factors and layer structures, clay minerals are capable of accommodating several cations from their exterior environment. These cations have a significant effect on the behavior of clay minerals, particularly those in liquid dispersion. Therefore, direct couple current plasma (DCP) was useful in revealing compositions involved with the dewatering phenomenon of clay particles.

Table 2 shows the chemical composition of samples from Gilmore and Crooked Creeks. The sample from Gilmore Creek apparently contained more divalent and less monovalent absorbed cations than the sample from Crooked Creek. This difference seems to be a major cause of the different behavior between the two samples.

Table 2. Results of chemical composition analysis from direct couple current plasma (DCP).

Element	Gilmore Creek sample weight (%)	Crooked Creek sample weight (%)
SiO_2	30.6	31.3
Al_2O_3	20.9	26.7
Fe	18.2	19.7
CaO	4.5	2.0
MgO	3.3	1.4
Na_2O	0.9	1.3
K_2O	1.3	1.8

FLOCCULANT TESTS

The authors examined 21 commercially available flocculants of cationic, anionic, and nonionic composition. Two flocculants, PEO and a cationic polymer, were chosen for testing.

Because a variety of factors are involved in the flocculation process, it is nearly impossible to create theoretically optimum conditions. Thus the determination of the best performance must be determined empirically, by step-by-step experimentation. In this work, various parameters that were likely to be important in the flocculation process were tested in the laboratory.

Mixing Speed and Time

There is a distinction between mixing and conditioning. Mixing (to homogenize the dispersion) involves water and applied chemicals, whereas conditioning is a gentle stirring that promotes flocculation of this homogeneous mixture.

Fast mixing worked better than conditioning; the postmixing conditioning did not apparently enhance the efficiency of flocculation. Higher speed mixing caused lower residual turbidity in the same period of mixing time. Also, both overmixing and undermixing greatly reduced polymer effectiveness. As a result, a 9-min mixing time with the stirrer was determined as standard for the experiments that followed.

Optimum Dosage

Van Olphen (1979) indicated that naturally occurring calcium clays are more unstable and more easily to be flocculated in a dispersion system. This phenomenon also was confirmed by Stanley and Scheiner in 1986, who in their ion-exchange research of the clay flocculation process, concluded that the efficiency of performance was inversely proportional to the ratio of ionic radius to charge (R/C). The R/C for Na and Ca ions is 0.95 and 0.495, respectively. Therefore, $CaCl_2$ (lime) can be used as an aid in clay flocculation technology. The results of this study, shown in figure 3, agree. Without $CaCl_2$, the residual turbidity could only be reduced to about 90 NTU, regardless of the amount of PEO solution. Adding $CaCl_2$ before PEO lowered the turbidity to 20 NTU. Sample A5 clearly illustrates this ability of $CaCl_2$. Polymer dosages that oversaturate the available surfaces of the dispersed particles will produce a restabilized colloid because no sites are available for the formation of bridges. However, underdosage decreases the process's capability of achieving the required standards.

The volume of flocs has to be considered, as does choosing the best dosage. If the volume of settled flocs is too large to be readily controlled, it may not be practical in field processing. Thus, the optimum dosages for the Gilmore Creek sample for PEO and $CaCl_2$ were chosen as 0.05 lb/1000 gal and 0.46 lb/1000 gal, respectively.

As shown in figure 4, the experimental results of sample B2 from the Crooked Creek area, using both PEO and $CaCl_2$, did not adequately reduce turbidity. The lowest turbidity that could be achieved was 70 NTU. Nevertheless, the acid ions played a positive role in performance. The edges of clay minerals will carry a positive charge with decreasing of pH (Van Olphen, 1979). To enhance the intensity of the positive charge and thereby promote edge-to-face particle attraction, sample B2 was diluted with HCl to decrease the pH value. At the pH values shown in figure 5, the required turbidity standard was achieved. By using a pH of 6, the optimum dosage can be chosen from figure 6.

In comparing samples A5 and B2, the size distribution and chemical composition offer an interesting contrast. Sample B2 contained more ultrafine particles than sample A5; thus, the edges of particles might perform a more significant role in flocculation. The DCP data revealed that sample B2 consisted of more Na_2O and K_2O, whereas sample A5 contained more MgO and CaO. Both Mg and Ca ions had lower R/C ratio than K and Na ions. Therefore, there is no doubt that sample B2 would be more difficult to clarify.

The optimum dosage of the cationic polymer for the Gilmore Creek samples was A4, 130 ml/1000 gal; A5, 100 ml/1000 gal; and A6, 70 ml/1000 gal. The optimum dosages of the cationic polymer for the more difficult Crooked Creek samples were 150 ml/1000 gal for B1 and 190 ml/1000 gal for B2.

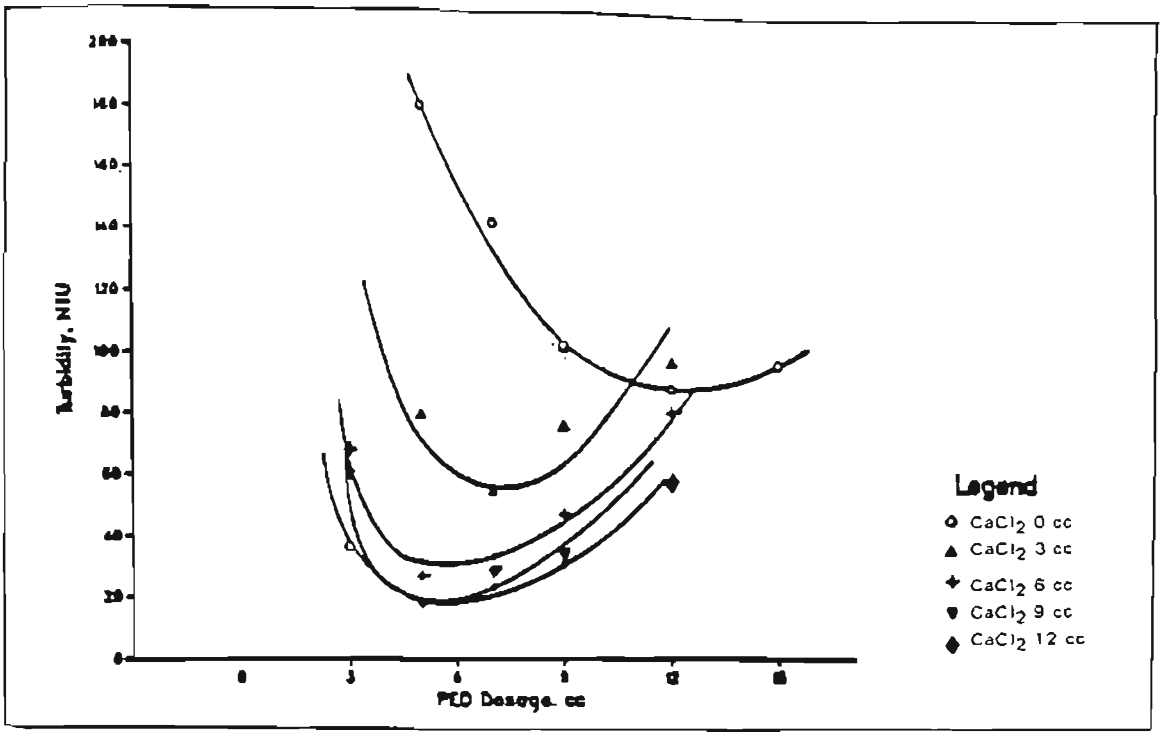

Figure 3. PEO dosage vs turbidity at various $CaCl_2$ addition rates, Gilmore Creek samples.

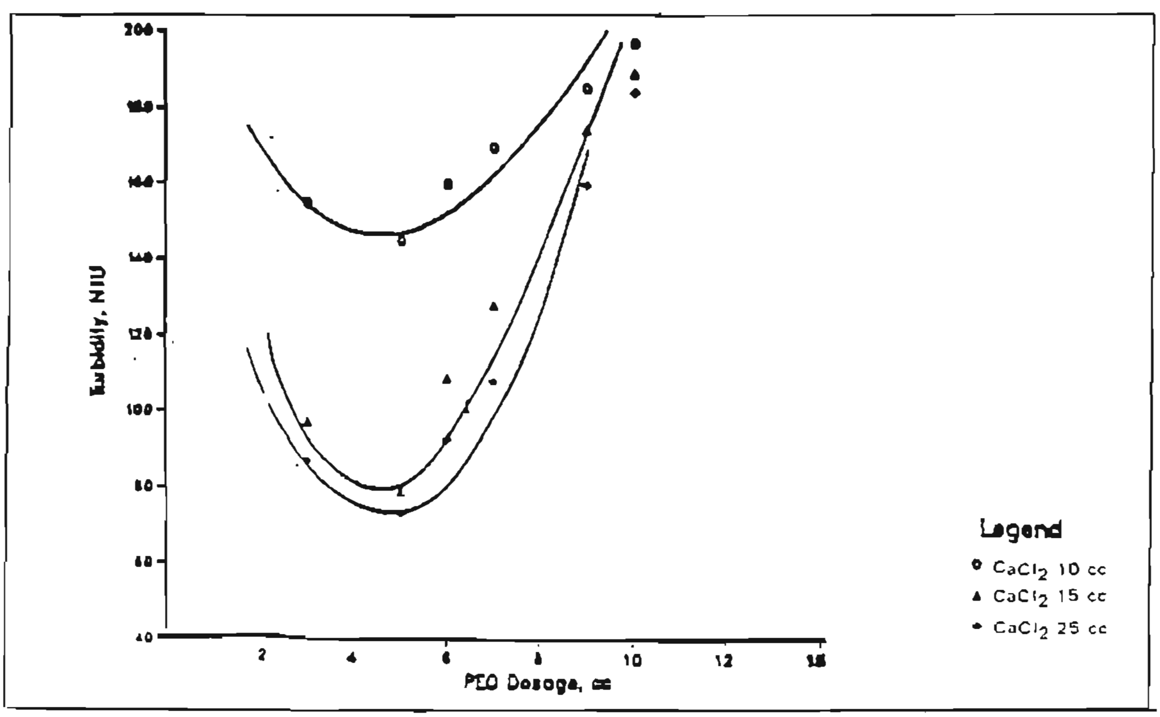

Figure 4. PEO dosage vs turbidity at various higher $CaCl_2$ addition rates, Crooked Creek samples.

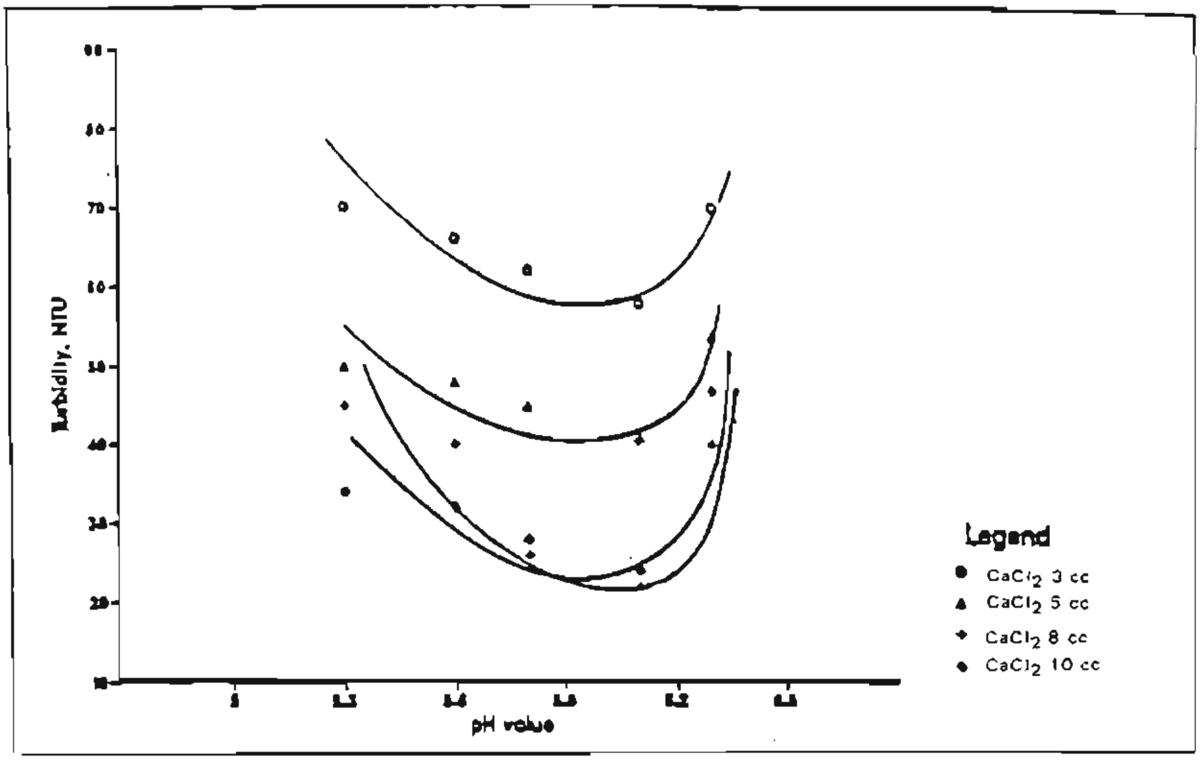

Figure 5. PEO dosage vs turbidity at various $CaCl_2$ addition rates, Gilmore Creek samples.

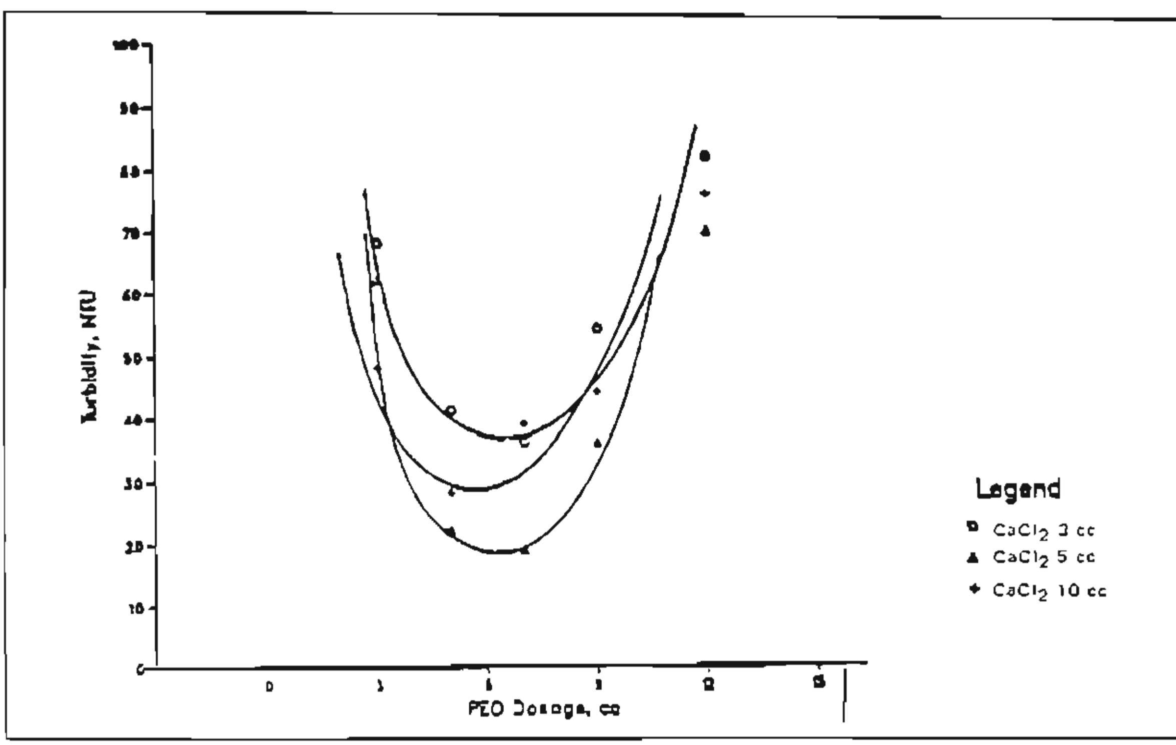

Figure 6. PEO dosage vs turbidity at various $CaCl_2$ addition rates, Crooked Creek samples.

Flocculant Dosage and Pulp Density

To determine the optimum cationic-polymer flocculant dosages for different pulp-density water samples, samples A4, A5, and A6 were chosen as samples representing high, medium, and low pulp densities, respectively. Samples B1 and B2 were also chosen to represent high and low pulp density, respectively.

The results of these tests are presented in figures 7 and 8, which show that the optimum flocculant dosage increases with the increasing pulp density of the tested samples. In short, the higher the pulp density, the higher the number of solid particles in the water sample and the larger the surface area that consumes flocculant.

The ability of resisting overdose decreases with decreasing pulp density of the tested sample. In other words, the greater the pulp density of a sample, the greater its ability will be to resist overdose. Decreasing pulp density results in decreasing particle surface area and the number of interparticle collisions during agitation.

Figure 9 presents the PEO flocculation results of three different samples of varying solids content. These data indicate that the higher the solid weight-percent, the larger the PEO dose that has to be used to achieve some degree of clarification. Overdosage occurs for samples with lower amounts of solids. Particle collisions are probably far less important in the mechanism of polymer flocculation, whereas collision is essential for traditional metal coagulants. In traditional alum and iron coagulants, no bridging action occurs, and particles are connected only by collision.

Synergistic Factors

PEO is expensive, and the PEO dewatering technique may be economically unattractive. However, if widespread use were made of PEO, 'economy of scale' would probably lower the cost of this chemical. One method for improving the economics of using PEO would be to find a low-cost reagent that could be used to replace a part of the PEO.

Smelley and Scheiner (1980) concluded that the synergistic effect is nil when cationic and nonionic polymers were used with PEO. For anionic reagents, even when a synergistic effect was observed when PEO was used with polyacrylamide, there was no economic advantage over PEO.

Natural guar gum, a nonionic polymer with a molecular weight of about 220,000, has been recommended by Smelley and Scheiner (1980) because it may produce effective synergism. Figure 10 indicates that the PEO dose can be substantially reduced when gum and PEO are used together. Guar gums also seem capable of improving the performing efficiency. With 0.2 lb/1000 gal of guar gum, turbidity was reduced to 13 NTU in sample A5 by using only 0.02 lb/1000 gal of PEO. For sample B2, the same dual functions was observed. Therefore, guar gums were chosen as a synergistic aid for joint use with PEO.

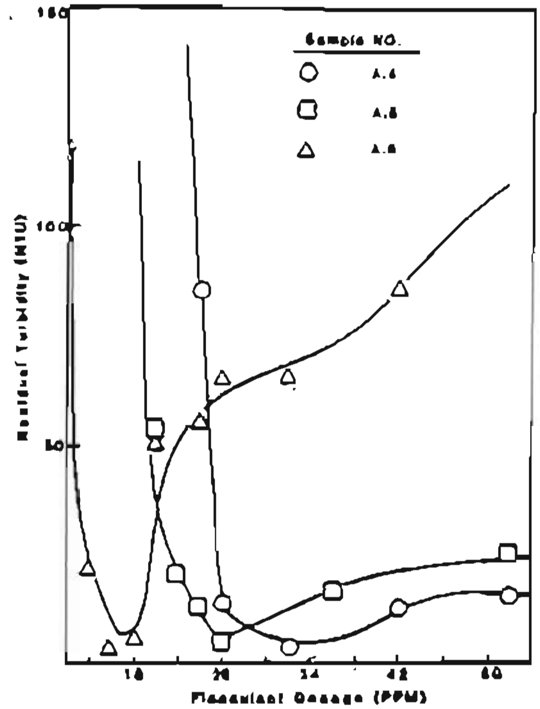

Figure 7. Effect of different pulp densities on optimum cationic-polymer flocculant dosage, Gilmore Creek samples.

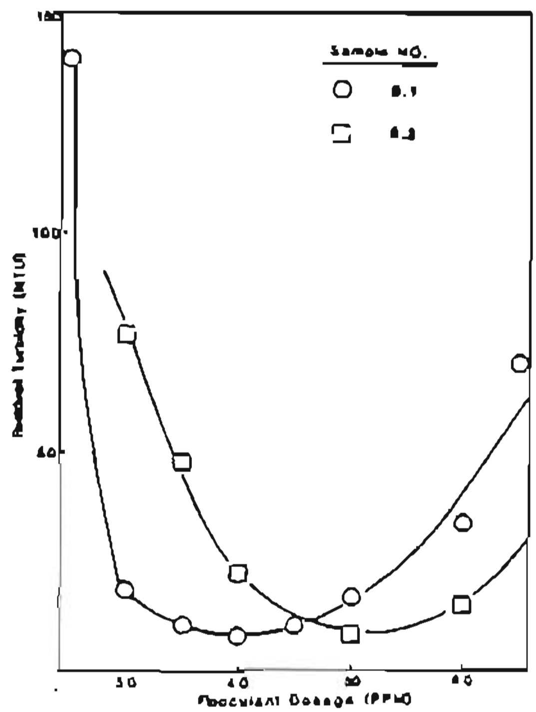

Figure 8. Effect of different pulp densities on optimum cationic-polymer flocculant dosage, Crooked Creek samples.

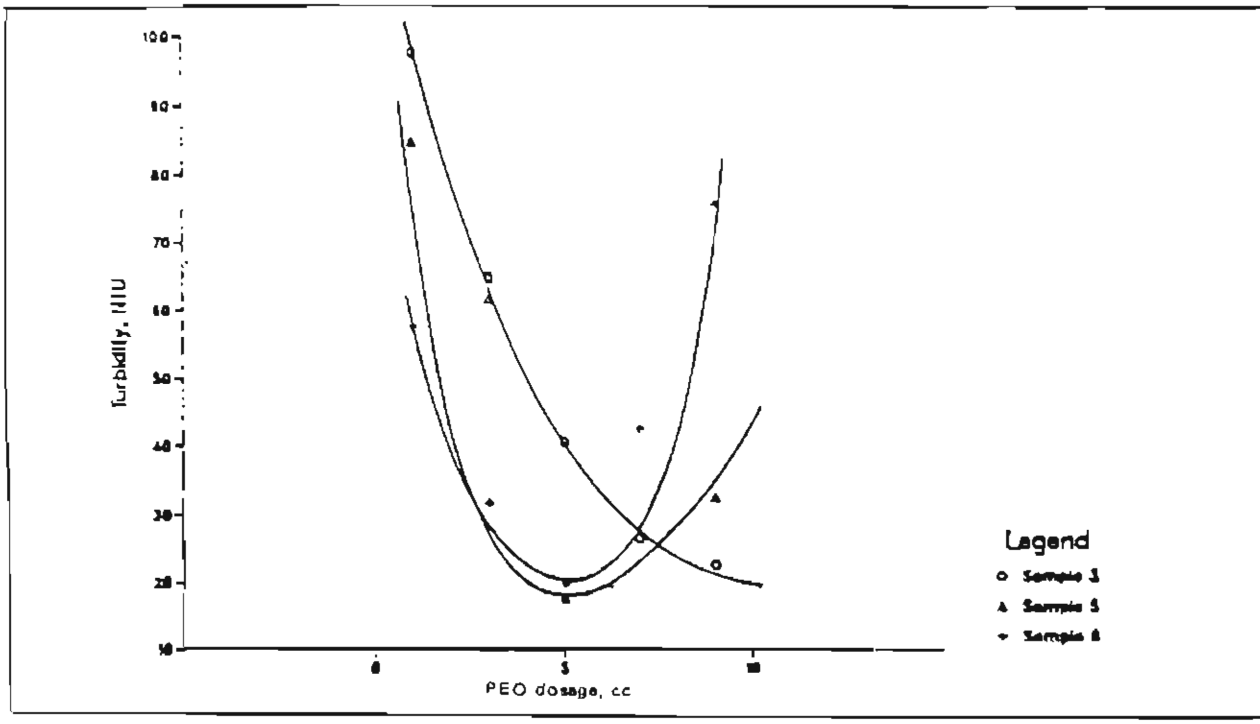

Figure 9. PEO dosage vs turbidity as a function of solids content, Gilmore Creek samples.

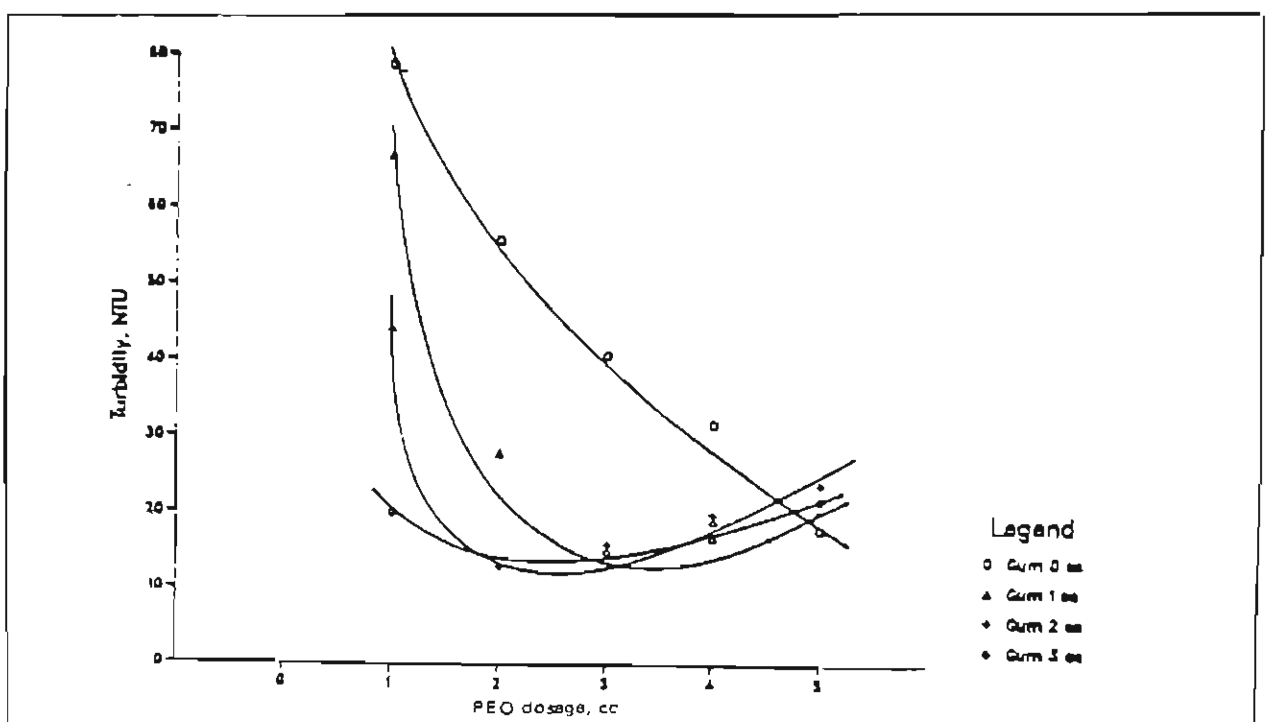

Figure 10. The synergistic affect of guar gum on PEO dosage for sample A5, Gilmore Creek.

Zeta Potential of Particles

In the above tests, the solid contents of samples A4 and A5 were much higher than those of samples B1 and B2. If the optimum flocculant dosage increases with increasing pulp density, the optimum flocculant dosage for samples A4 and A5 should be higher than that for samples B1 and B2. However, when comparing figures 7 and 8, the opposite result was observed. This can possibly be explained by the concept of zeta potential. Because the nonspecific electrostatic interaction between cationic flocculant and negatively charged particle surface play an important role in the flocculation process, the particles with higher negative zeta potential (that is, higher negative surface charge) are more amenable to flocculation with a cationic flocculant. Thus, the optimum flocculant dosage of sample A (zeta potential of -21 mv) will be less than that of sample B (zeta potential of -16 mv), regardless of the effect of pulp density.

Flocculant Addition Rate

The residual turbidity and the floc settling rate of tested samples were measured after being flocculated at different cationic-polymer flocculant addition rates. The results, shown in figure 11, indicate that there is an optimum flocculant addition rate for decreasing turbidity. Figure 12 indicates that a slow flocculant addition rate always helped floc settling; this affect is depicted in figure 13. When the flocculant was added in small doses, the enormous effective surface area of the particles was reduced each time. The overall affect is to prevent local over absorption and waste of flocculant and to form uniform large-size flocs. This helps the overall affect of turbidity removal. On the other hand, adding the flocculant too slowly will cause the floc to disintegrate because of a faster agitation rate, thus jeopardizing the overall affect for decreasing turbidity.

pH Value

To determine the affect of pH value on the flocculation process, the pH was varied from 2 to 10 by adding either nitric acid or sodium hydroxide.

The results shown in figure 14 indicate that a higher pH value resulted in a higher residual turbidity after cationic polymer flocculation. At high pH values, the particle surface is highly negative as it absorbs the hydroxyl ions, resulting in higher negative zeta potential.

Concept for a Full-scale System

Following laboratory experiments, a field plant should be designed to study the feasibility of using either PEO for quick solids removal or a cationic-polymer flocculation process where there is room for settling ponds. Before a well-designed plant can be economically and practically built, there are two features that must be considered. Because the settling pond is the most popular and inexpensive technique, it has to be included to get rid of settleable particles, thus reducing the consumption of PEO or cationic-polymers. Another consideration is the technique of recycling, which will lower the amount of waste water that has to be treated by a chemical process.

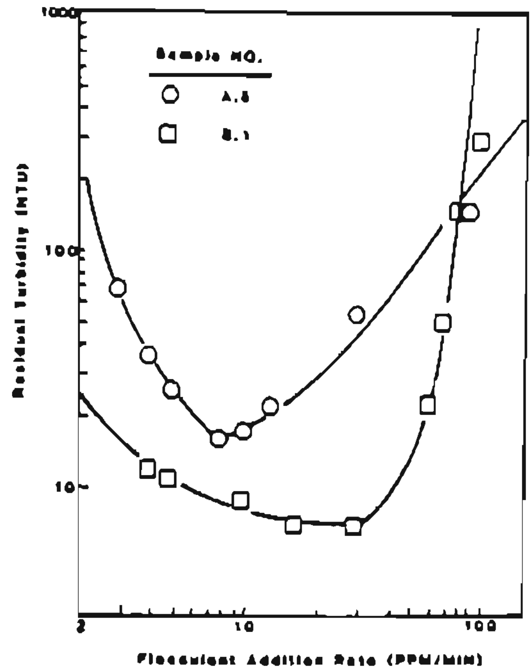

Figure 11. Affect of cationic-polymer flocculant addition rate on residual turbidity, samples A5 and B1.

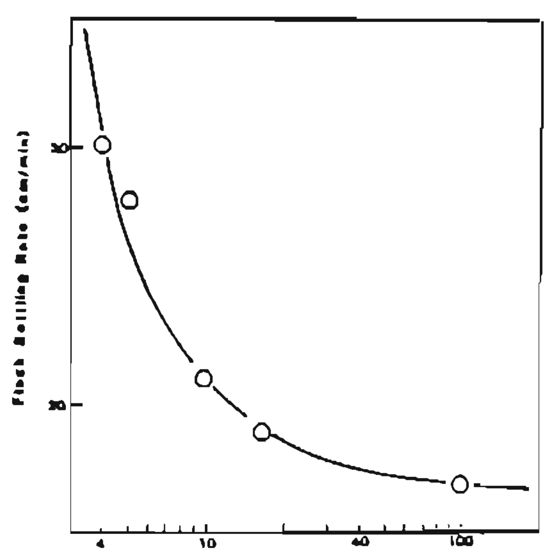

Figure 12. Affect of cationic-polymer flocculant addition rate on floc settling rate.

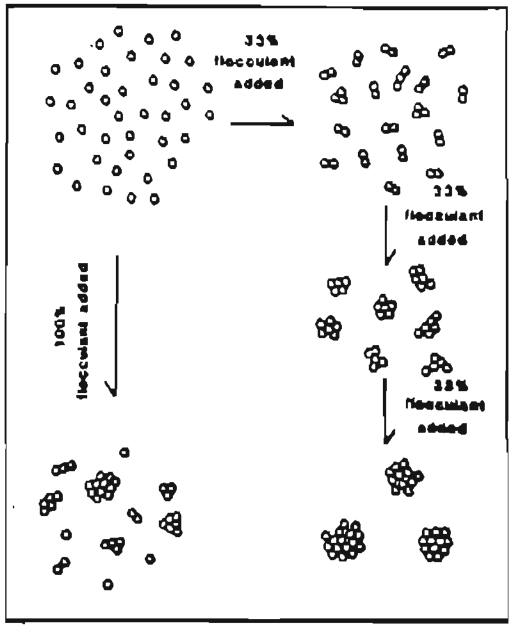

Figure 13. Affect of multistage and single-stage addition of flocculant during flocculation process.

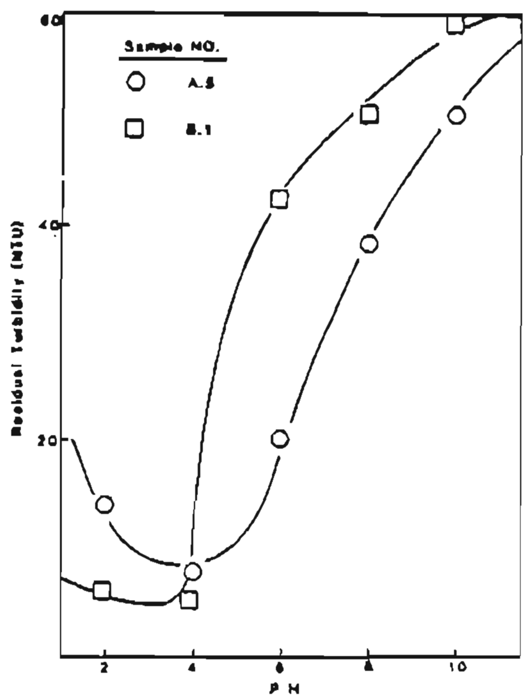

Figure 14. Affect of pH value on residual turbidity during cationic-polymer flocculation.

Therefore, the only effluent that needs treatment is the surplus waste water to be discharged into a stream.

Settling Ponds

Even if the normal solids content of processed water has been proven not to have an adverse effect on gold recovery, it should still be reduced to a point as low as possible to eliminate any risk of fine gold loss in the sluice box and to prevent a decrease in the affect of the PEO or cationic-polymer dosage. To gain the greatest efficiency in settling ponds, several factors should be taken into account: 1) retaining water long enough for coarse particles to settle, 2) prevention of short circuiting, 3) minimizing scour and resuspension of solids during periods of high flow, and 4) preventing washouts during floods (Weber and Post, 1985; Peterson, 1987).

Chemical Treatment Plant

After well-designed settling ponds have been constructed, the recycled water will have very low solids content, which can minimize the negative affects on gold recovery. A chemical treatment plant can then be designed and fabricated based on the amount of surplus waste water. A treatment plant would consist of a holding tank for partially treated water, one or more tanks for reagent, a conditioner-mixer with a propeller-agitator to mix the water to be treated with PEO or cationic-polymer solution, and a rotary screen to aid dewatering and consolidation of the solids. The treatment plant should also have two positive displacement pumps to feed the slurry and the flocculant at the predetermined rate.

Because the equipment size primarily depends on the amount of surplus waste water that has to be handled, miners ought to conduct an individual calculation to design an effective plant. Smelley and Feld (1980) reported that for a 30 gal/min output, the miner needs:

Slurry holding tank	---	1,000-gal capacity.
Reagent holding tank	---	20-gal capacity.
Conditioner-mixer	---	100-gal capacity.
Rotary screen trommel	---	2 ft diameter by 15 ft (48 mesh opening at first part).

For promoting efficiency, several factors such as speed and slope of trommel and feeding rate of slurry waste have to be considered in detail. Another consideration is the retention time of the slurry in the conditioner-mixer and in the trommel.

CONCLUSION

In conjunction with settling ponds, the use of PEO or cationic-polymer in treating effluent from placer mining has been proven in this study to be economically effective in reducing turbidity levels.

The solid content found in the waste water in this study ranged from 3.23 to 0.05 percent. Most of the particles were less than 10 microns in diameter and carried -16 to -21 mv of zeta potential. Because placer mining operations do not have an adverse effect on pH value, it is difficult to cause settling of these ultrafine particles, and severe turbidity results.

The effectiveness of PEO or cationic-polymer treatment of placer-mining slurry depends on mixing speed and time, content of fine particles, and the dosage and concentration of the reagent solution. Other significant factors to consider are the physical properties and chemical composition of the fine solid particles. Flocculated particles settle very rapidly, so no conditioning settling period is needed for further clarification. $CaCl_2$ is useful in promoting the bridging process, but cannot handle the entire operation of solids removal by itself. Optimum reagent dosages for the most difficult-to-treat waters at the Gilmore and Crooked Creeks sites are given in table 3.

Table 3. Reagent dosages found to give optimum results in reducing turbidity in Gilmore Creek (A5) and Crooked Creek (B2) mine water samples.

	Chemical dosage per 1,000 gallons				
Sample	PEO only	PEO + $CaCl_2$ (lb)	PEO + $CaCl_2$ (lb)	Guar gum (lb)	Cationic polymer (ml)
A5	0.12 lb	0.05 + 0.46	0.02 + 0.2	0.46	70
B2	NA	0.06 + 0.40	0.03 + 0.2	0.40	150

Guar gums where found to be a synergistic aid for joint use with PEO. The guar gum not only reduces the required dose of PEO up to 50 percent, but also improves flocculation performance.

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GOLD OCCURRENCES AND CHARACTERISTICS IN THE CHANDALAR-KOYUKUK AREA

by

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ACKNOWLEDGMENTS

Geochemical studies of Alaskan gold deposits were begun in 1982 as a joint study by the U.S. Geological Survey and the State of Alaska, Division of Geological and Geophysical Surveys, but in reality, this study is made possible by and is dependent upon the placer miners of Alaska. Without your enthusiastic response to our request of samples of gold, your willingness to let us visit your properties, and the overall splendid cooperation you have shown us, this project would never have been possible. So to you, the Alaskan miners, we once again express our thanks and sincere gratitude for your help.

INTRODUCTION

The Alaskan Gold Project is a part of the Alaska Mineral Resources Appraisal Program. The objectives of the study are to characterize the deposits, to determine relationships of gold in placer deposits to possible lode sources, to identify possible sources for gold in placer deposits, to study processes of placer formation, to contribute to existing knowledge of the principles of prospecting for placer deposits, and to determine if minerals associated with placer deposits might suggest economic deposits of other metals.

THE STUDY AREA

Gold is a significant resource in the Koyukuk and Chandalar mining districts, which are located mainly in the Brooks Range in north-central Alaska (fig. 1).

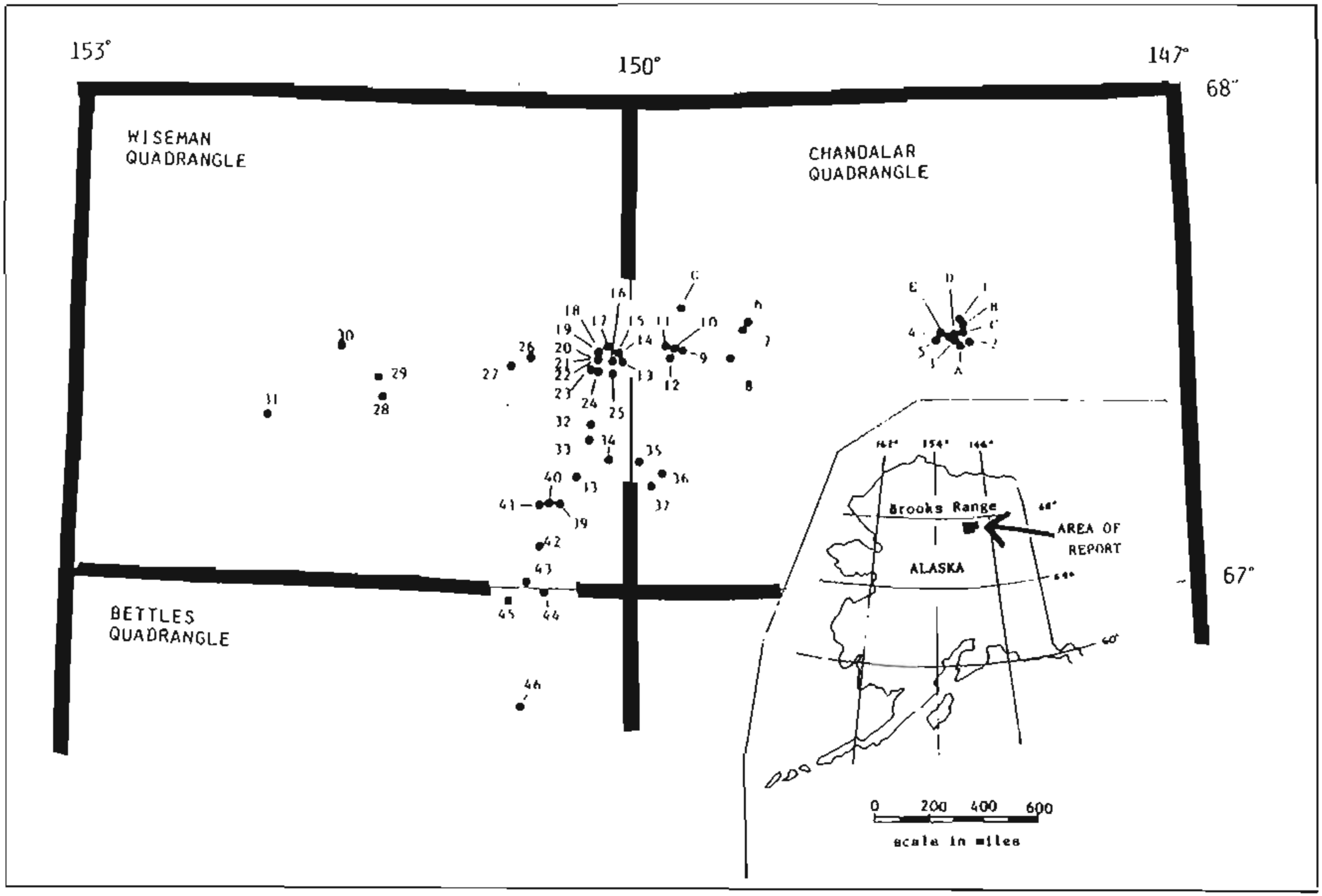

Figure 1. Map of the lode-gold (A-C) and placer-gold (1-46) sample localities in the Koyukuk-Chandalar mining districts, Alaska.

The study area is about 160 km long and lies almost entirely within the Wiseman and Chandalar 1° x 3° Quadrangles. Gold has been produced in the districts since the early 1890s. Most gold production has come from Quaternary fluviatile placer deposits, but gold-bearing quartz veins have also been mined.

The geology of the study area is complex, being complicated by three or more episodes of metamorphism and multiple episodes of faulting, including some thrust faulting. The study area comprises a central metamorphic belt of metaigneous, metavolcanic, and metasedimentary rocks of early Paleozoic and/or Precambrian age that are predominantly schists with some phyllite and quartzite. The schist belt is flanked on the north by a broad belt of mostly Devonian sediments and metasediments. Cretaceous sedimentary rocks lie along the southern border of the Wiseman Quadrangle, and Cretaceous granitic plutons lie along the southern border of the Chandalar Quadrangle. Jurassic, Triassic, and Permian mafic volcanic rocks occupy parts of the area south of the central metamorphic belt. The lower Paleozoic and Proterozoic basement rocks experienced pre-Mississippian metamorphism, and all the rocks were regionally metamorphosed twice during Mesozoic time. Late Mesozoic tectonism caused widespread obduction thrusting, lateral faulting, and uplifting, resulting in an accreted terrane. Quaternary glacial, alluvial, and colluvial deposits containing placer gold unconformably overlie the polymetamorphic rocks.

SAMPLING TECHNIQUES

Placer-gold samples were collected from 46 localities in the Koyukuk-Chandalar mining districts. Six lode-gold sites were also sampled, five in the Chandalar area and one from Sukakpak Mountain (fig. 1). Most of the claims in the districts that were active in 1982 and 1983 are included in the study. Miners very generously supplied the gold or gold concentrate or allowed panning in the deposit. At a few localities, gold was recovered by panning stream alluvium.

Gold for analysis was obtained from concentrates by handpicking in the laboratory using a binocular microscope. No other laboratory treatment was applied to the gold.

The gold grains studied ranged in size from less than 0.15 mm to about 3.0 mm, but most were intermediate in size from 0.2 to 1.0 mm. The gold grains exhibited a variety of crystalline morphologies ranging from flat, smooth grains to highly irregular crystals to wire gold. The placer and lode gold samples were analyzed by a D.C.-arc direct-burn procedure. A total of 38 elements were determined and Au content was determined by summation of the 38 elements determined, and subtracting from 100. Because native gold exhibits extremely variable composition, multiple analyses were made for each sample location when enough gold was available. A total of 460 analyses were made on the 46 placer locations and 28 analyses on the six lode-gold samples.

In this report, emphasis was placed on the Au-Ag-Cu ternary system. These three elements have very close chemical properties. Gold, silver, and copper are members of Group IB of the periodic system, exhibit monovalency, and are crystallographically alike. Gold and silver have an atomic radius of 1.44 Å compared to copper's radius of 1.28 Å; as a result, most samples of native gold contain less copper than silver. In the laboratory, gold and silver are continuously miscible in all proportions and form solid solutions; in nature, however, the series appears to be discontinuous. No native gold-silver alloy has been reported with a fineness less than 400. Copper is miscible in all proportions in the liquid state with either gold or silver and forms a solid solution with gold, but has a limited range of miscibility with silver in the solid state.

RESULTS OF GOLD ANALYSIS

In this study, all math calculations were made on the average of site values and not on the average of the total analytical observations. The gold content for the 46 placer-gold localities ranged from 76.1 to 94.9 weight percent, with an average value of 88.0. Silver content ranged from 3.5 to 22.9 weight percent, with an average value of 10.5; copper content ranged from 0.006 to 0.282 weight percent, with an average value of 0.040. These data represent a wide range of values.

When normalized copper is plotted against normalized silver, five distinct geochemical types of placer gold are identified in the Koyukuk-Chandalar districts (fig. 2). Normalized values are obtained by dividing the site silver or copper value by the average silver or copper value for the 46 sites; 10.5 percent for silver and 0.040 percent for copper. Normalized values are used in place of real values to get the numbers in a range somewhere near 1 and thereby simplifying the numbers in the graph. A value of 1 correlates with the average. These geochemical types of placer gold can be described as: Type 1 having a high Ag, low Cu content; Type 2 having a average Ag, low Cu content; Type 3 having an average Ag, moderate Cu content; Type 4 having below average Ag, above average Cu content; and Type 5 having low Ag content and high Cu content. Type 2 gold may be divided into two subgroups, types 2A and 2B. Golds of type 2B are similar in composition to Sukakpak Mountain lode gold and are derived from creeks nearby Sukakpak Mountain.

The gold in the study area was also analyzed for fineness. Fineness is calculated as true fineness and expressed as parts per thousand using the equation Au/(Au + Ag) x 1000. Disregarding type 2B samples, the data showed a systematic increase in Au, fineness, Cu, and Au/Ag ratio values from type 1 to type 5. Note that the Cu content found in type 2B fits between type 1 and 2A.

Conversely, except for type 2B, there is a systematic decrease in silver, gold-copper ratio, and silver-copper ratio values from type 1 to type 5. The silver content of type 2B fits between types 3 and 4, whereas the copper content for type 2B was between types 1 and 2A. From these analyses, type 2B gold, which doesn't fit the pattern established by the chemistry of the other placer-gold types, appears to be anomalous.

An interesting feature is that when a regional plot is made of the placer-gold deposits by 'type' (fig. 3), type 1 gold placers occur at the

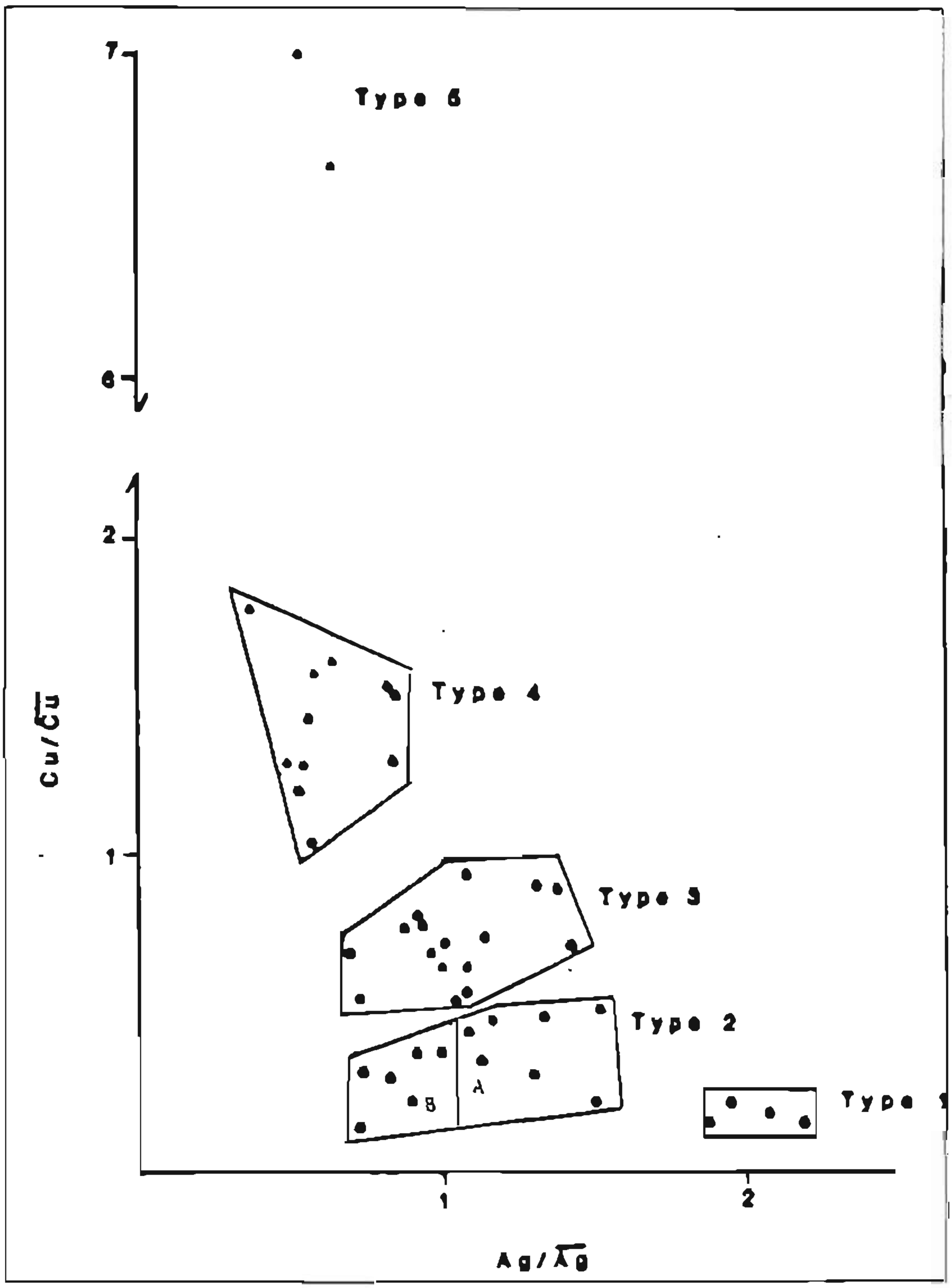

Figure 2. Plot of normalized Cu vs normalized Ag showing five separate geochemical types of placer gold in the Koyukuk-Chandalar mining district, Alaska.

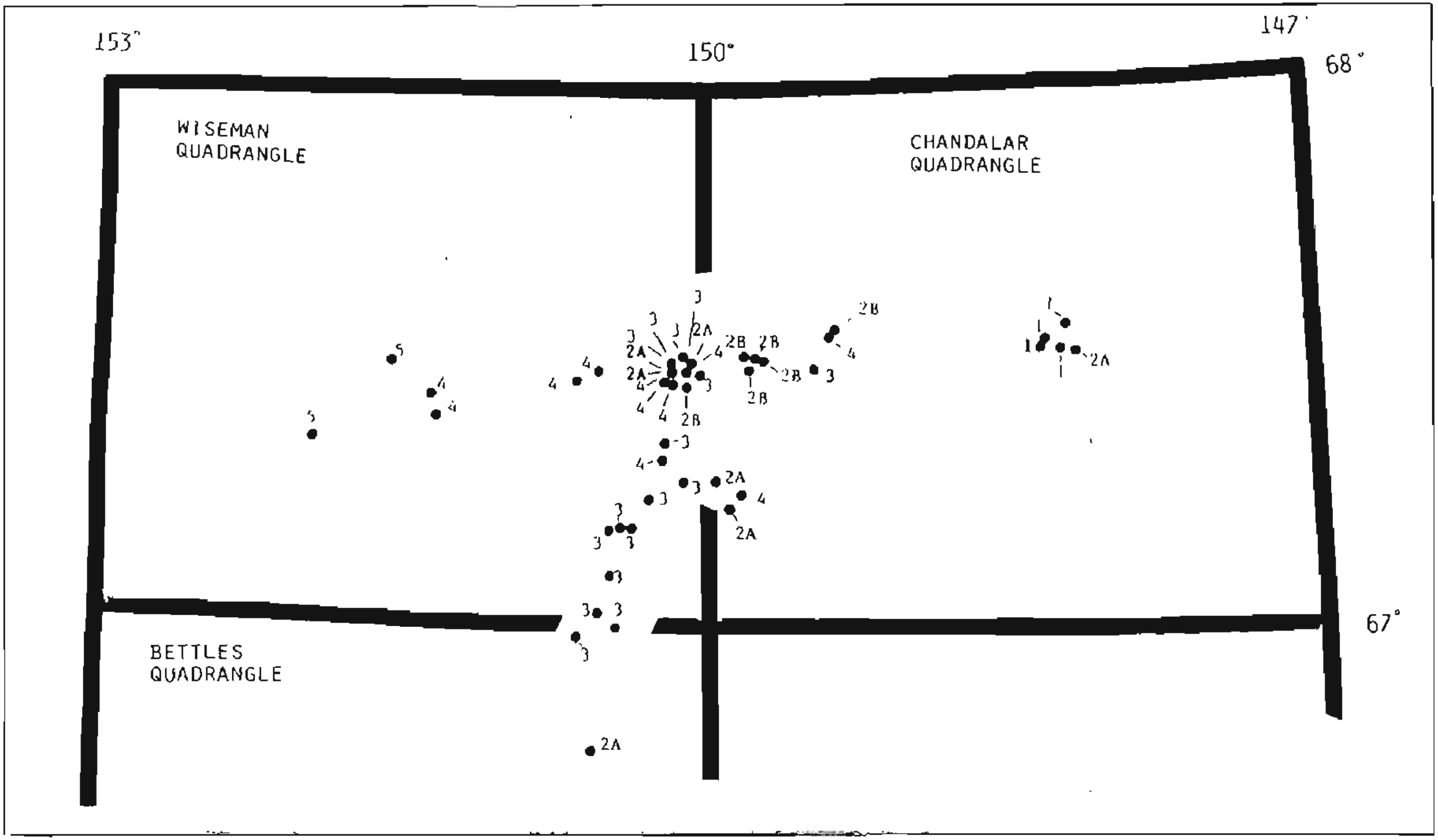

Figure 3. Regional plot of placer gold by "type," Koyukuk-Chandalar mining district, Alaska:
Type 1 - high Ag, low Cu content
Type 2 - average Ag, low Cu content
Type 3 - average Ag, moderate Cu content
Type 4 - below average Ag, above average Cu content
Type 5 - low Ag, high Cu content.

east end of the study area and types 2B, 2A, 3, 4, and 5 generally occur in order proceeding west across the district. The north-south-trending string of type-3 samples are all from one drainage, the Koyukuk River draining from the north, so they do not indicate a regional north-south systematic pattern. We interpret the regional pattern of the placer-gold types to be related to the depositional environment of the primary gold sources. The fineness of gold in any ore deposit varies with depth from the surface at which the deposit was formed. Gold enrichment (a high gold-silver ratio) occurs mostly in higher temperature, deeper seated deposits, whereas silver enrichment (a low gold-silver ratio) is characteristic of low-temperature ore deposits of intermediate or near-surface depositional depths. The fineness of the placer gold is a measure of the fineness of the gold shed from the outcrops of the lode-gold sources. Admittedly, placer gold does not necessarily correspond exactly to the fineness of gold as deposited in the ore, primarily because of differential solution of surface silver on gold particles in the alluvial environment. This removal of silver from gold particles is not uncommon in placer golds. It is referred to as surface refining and takes place only to a limited depth from the surface of the individual grains; it therefore causes only slight changes in the overall gold-to-silver ratio.

SILVER CONTENT

In our study, enough gold was obtained from 15 of the placer gold locations to permit analyses of a -0.5 mm fraction and a +0.5 mm fraction. Values for silver content of the -0.5 mm fraction plotted against that of the +0.5 mm fraction showed that four of the sites displayed significantly higher silver content in the larger size fraction and that four of the sites showed significantly higher silver in the smaller size fraction. Two sites revealed a moderate increase in silver content in the smaller size fraction, and the remaining five sites were nearly equal in silver content in the two size fractions. The -0.5 mm fraction contained a greater surface area, and if the gold grains were being altered by preferential chemical leaching of silver from the surface, one would expect this fraction to have a lower silver content. However, in our study, that is not the case. Data from our study suggest the placer gold may not have been significantly altered from the lode-gold source. Therefore, on the basis of geochemical data for the placer gold, an ore depositional gradient from east to west appears to exist. The Au-Ag-Cu ternary system further identifies this gradient.

When normalized copper is ratioed to normalized silver, the resulting value represents copper enrichment or depletion with respect to silver. The relationship of silver and copper with gold are shown by plotting this value against fineness (fig. 4). The line of solid solution for the ternary system could depend on the physical-chemical parameters at the time of ore deposition and, therefore, reflect the depth of deposition for the primary gold. The physical-chemical parameters that influence metal solubilities (and, as a corollary, ore deposition) are factors such as temperature, pH, hydrogen and oxygen fugacity, chlorine and bisulfite activity, and hydrogen-sulfide fugacity. These factors are complexly interrelated, and definitive answers are difficult to quantify as well as difficult to describe. In the ternary system for native gold, higher copper values correlate with high fineness. Copper and silver show an inverse relationship. All three

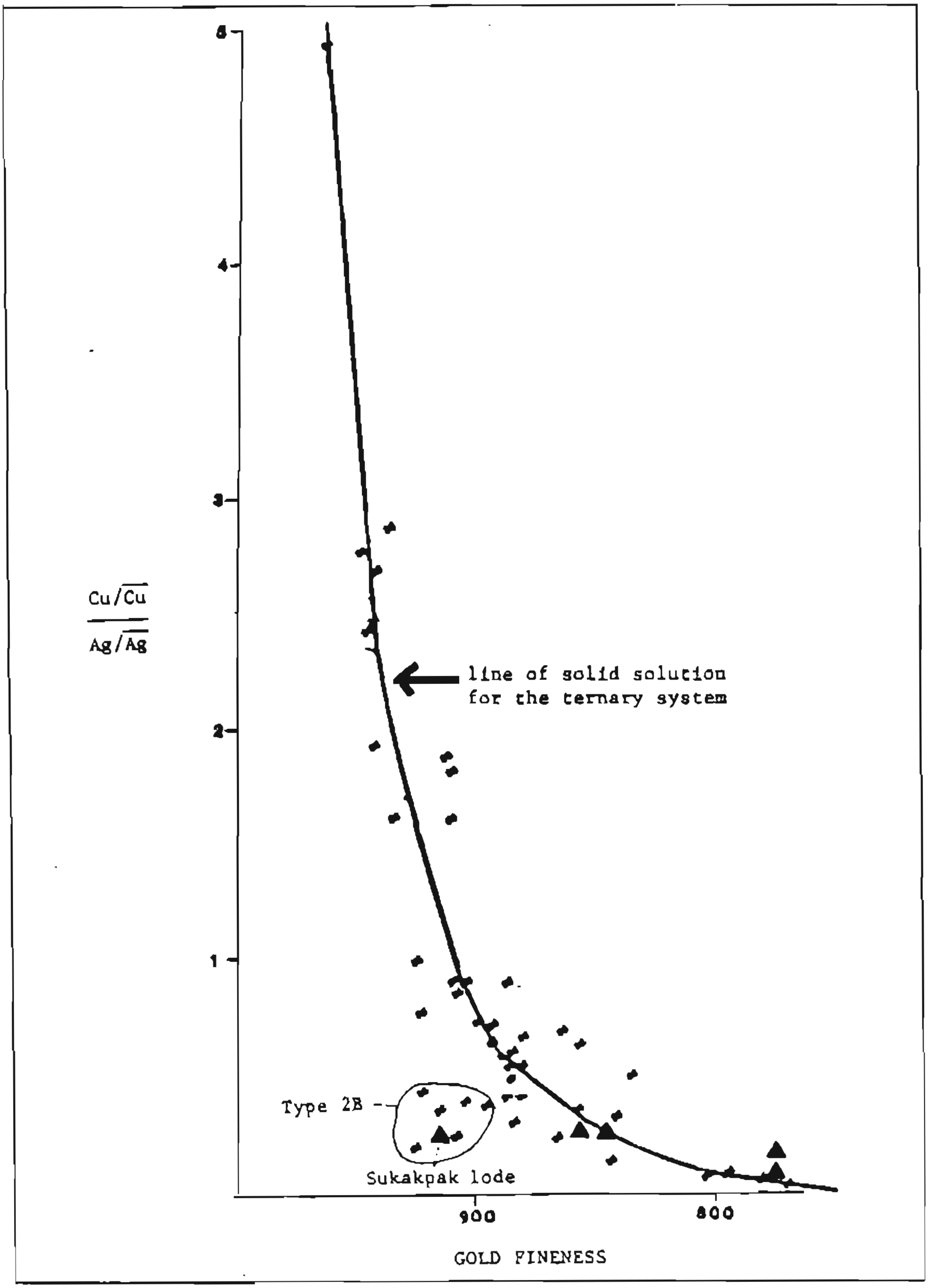

Figure 4. Normalized copper to normalized silver ratios vs gold fineness for placer (•) and lode (▲) samples in the Koyukuk-Chandalar mining district, Alaska.

elements were probably present in the ore-forming hydrothermal solutions. At increasing depth, copper appears to have a greater opportunity relative to silver to form a solid solution with gold. In the hydrothermal models described by Reed and Spycher (1986) and by Buchanan (1981), base metals occupy an early paragenetic position. The base-metal horizon occurs below the boiling level, and the precious-metal horizon occurs above the boiling level. At the level of boiling, a mixed zone of precious- and base-metal mineralization occurs. The Cu/fineness relations of placer gold in the Koyukuk-Chandalar districts can generally be predicted from the solid-solution curve. Chandalar lode golds also fit the predictable ternary curve. The exception to the ternary curve is Sukakpak Mountain lode gold and the placer gold from nearby creeks (our type-2B placer gold). This gold is all either 1) depleted in copper relative to the fineness or 2) depleted in silver (yielding very high fineness) relative to copper. We don't know why the ternary relations of these golds differ. We do know, however, that the lode gold from Sukukpak Mountain occurs in a stibnite vein, at the contact between marble and schist, whereas the Chandalar lode gold is in quartz veins. These quartz veins are dilatant quartz veins, commonly seen at the Little Squaw Mine. In type 2B gold, antimony probably had an influence on the proportioning of silver and/or copper content in native gold during gold deposition.

CONCLUSIONS

The geochemical types of gold give a clue to the thermal history of the south-central Brooks Range and indicate that the placer types may be related to depth of ore deposition.

On the basis of the gradient that is inferred from the gold analyses, late Mesozoic tectonism of the Brooks Range---which may have caused the ore-forming hydrothermal solutions to be emplaced in progressively lower thrust plates from east to west across the Koyukuk-Chandalar district---probably influenced the chemistry of the lode and placer golds in the district.

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THE ASSESSMENT OF REPROCESSING HARD-ROCK TAILINGS

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INTRODUCTION

Mill tailings have been a subject of conversation in the mining industry since they first came out of the discharge pipes from the old mills. Older technology and less efficient processing systems of the early operations would imply some minerals might be present that were not recoverable at the time the mills operated. However, experience has taught us that the tailings grades are generally very low and uneconomical to reprocess. The problem to date for the mining industry has been how to move large volumes of the tailings materials with low capital cost for equipment and overhead and yet maintain acceptable standards of mineral recovery and environmental control.

One of the more promising approaches to solving this problem comes from FreeGold Recovery, Inc., which combines proven equipment from different industries to make one highly compatible process.

EQUIPMENT

The old mills provide an already milled product for the head feed to a reprocessing system. A Toyo submersible slurry pump, which can handle up to a 50-percent-solids slurry with particle sizes up to 1 in. diam, was chosen as the primary unit for feeding the circuit. The pump can be lowered to the surface of the materials to be mined and it will generate its own 'feed sump.' For ponds, it can be lowered on a winch or chain-block to the desired level where pumping of solids can be initiated. Pumping rates can be monitored by the power draw of the pump motor. For dry tailings deposits, mining can be started up by generating a small pit and adding the required amount of water for slurry generation. The same system of feed control can be used. The Toyo pump discharge is directed to a 48-in.^{2} Sweco screen fitted with a -10 mesh (1.65 mm) screen deck.

The +10 mesh material and coarse debris are discarded. The -10 mesh underflow product is then pumped to a bank of 6-in.-diam dewatering cyclones, which dewater the feed product, producing a 62-percent-solids underflow;

they also act as a buffer to control any fluctuation within the feed system. This enables the circuit to maintain a constant feed to the Reichert cones and spirals.

The cyclone underflow product is pumped to a Reichert Cone Concentrator (type 4DSV), which is used for primary upgrading. Concentrate from the cone is fed to two twin-start LG7 Reichert spirals for a secondary upgrading. The Reichert cones and spirals were chosen because of their ability to process a large volume of material with low capital cost for equipment and overhead and yet still maintain high mineral recovery ratios with solid environmental control.

Concentrates for the LG7 spirals are gravity fed onto a Gemini gold table that achieves unique separation capabilities when fed black-sand 'preconcentrate.' It has a distinct advantage over previous table designs, because it is able to produce a clean free-gold product with the exceptional recovery capability of a finishing table. The combination of the Reichert cone, spirals, and Gemini gold table routinely recovers gold down to 400 mesh size and finer, depending on the physical characteristics of the gold particles.

When all of these pieces of equipment are put into a system, the basic flow sheet would look something like that shown in figure 1.

There could be modifications made to this basic flow sheet to adjust for each specific property situation such as a grinding circuit, multistage screening equipment, and magnetic separators. The whole system is modular and could be sized to fit specific needs from 3 tons per hr to several hundred tons per hr.

The system can be used as a complete plant or as a primary concentrator to provide an upgraded feed ahead of a flotation or leach plant. This would reduce the amount of material that has to be chemically treated or processed, allowing a more environmentally sound mining operation with a higher operation efficiency.

Another important benefit of this system is that it is not labor intensive; only two men per shift are required. There is virtually no maintenance. Also, high-specific-gravity minerals other than gold and silver can be recovered.

EXPLORATION AND TESTING

An attractive feature of reprocessing of old mill tailings is that past-producing mining companies have already installed the infrastructure for a producing mine. The cost of exploration and evaluation is greatly reduced because of the existence of roads, buildings, electricity, and so forth.

Accordingly, a small exploration crew can examine a property and, within a few days, develop enough hard data to determine if the property warrants further exploration.

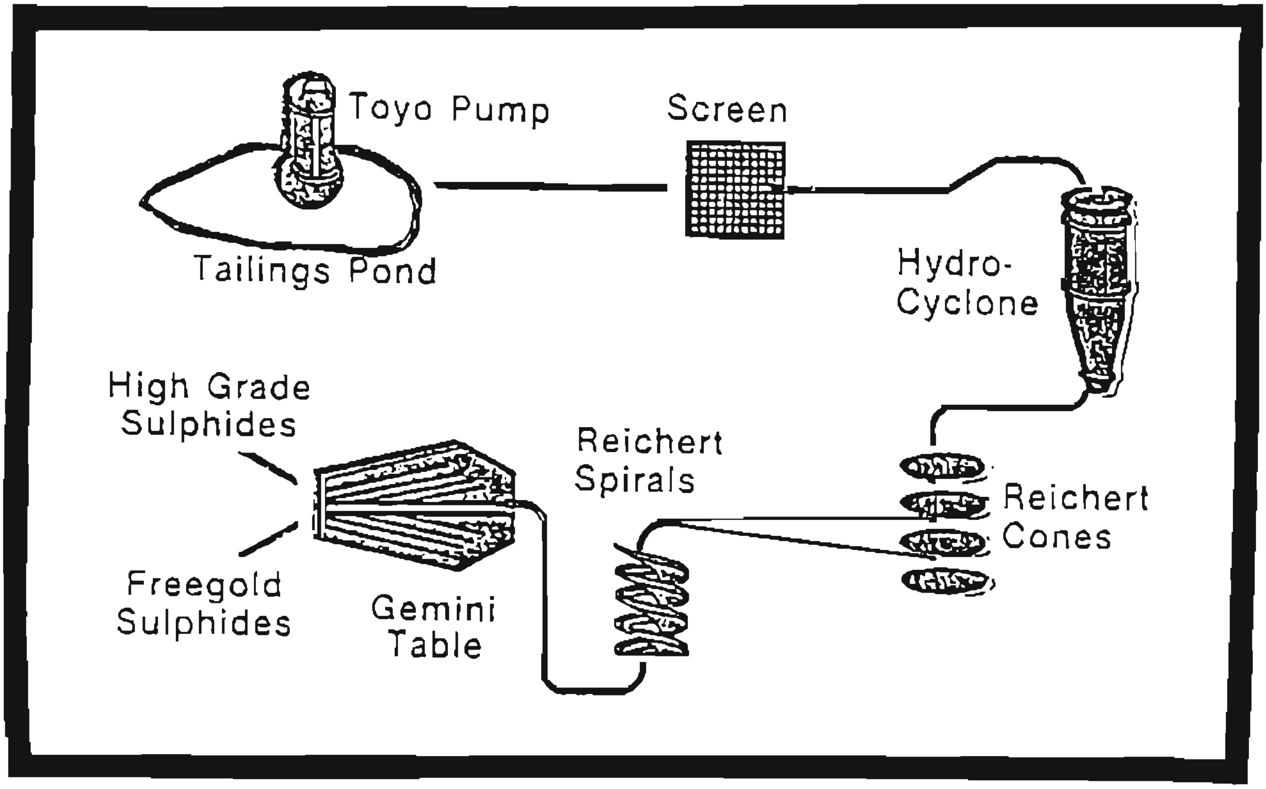

Figure 1. Operating flowchart, Freegold Recovery, Inc.

If so, an exploration team can complete a comprehensive evaluation program in a matter of weeks. By using the same equipment in sampling that would be incorporated in a production plant, the management team can not only complete its exploration program to establish reserves and tonnages, but can also eliminate an extremely risky problem in the mining industry, namely, the scaling up of exploration projections to commercial-plant performance.

This enables a company to explore tailings deposits, develop reserves, and make a production decision in a short time. It results in a significantly lower up-front development cost, compared to the mining industry in general.

SUMMARY

The future for applications using this process looks very bright. Part of the circuit was just installed as a scavenger unit in a large hard-rock operation in northern Ontario with very good results. Continuous testing and evaluating will be carried out. Past producing mining companies in North America number well into the thousands; there should be no shortage of projects that will be amenable to this process.

SUMMARY OF THE EFFECTS OF SUSPENDED SOLIDS, VISCOSITY, AND CLAY ON FINE-GOLD RECOVERY

by

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INTRODUCTION

There is no doubt that using recycling techniques reduces the volume of water discharged from a mining operation and that this reduction, combined with a clear-water bypass system, definitely improves downstream water quality. There is concern that recycling causes a buildup of suspended solids that leads to gold migration and loss in a sluice box. However, it is unknown whether gold migration and loss result from total suspended solids, clay, viscosity, riffle packing, other factors, or a combination of factors. This paper summarizes information from two pilot-scale sluice-box studies conducted because of the concern that recycle water may cause gold migration or loss.

TOTAL SUSPENDED SOLIDS

The literature contains many statements relating total suspended solids to gold loss. Many authors (Redmond, 1948; Cook and Rao, 1973; DINA, 1981) imply that gold losses occur when the solids content of the water becomes too high, although there is no quantitative evidence supporting this contention. According to Redmond, dredge-water solids may reach 3 or 4 percent (31,000 to 41,000 mg/L) before being considered too serious. DINA (1981) cites a dredge consultant who stated that the water should have a specific gravity of less than 1.1 or a maximum concentration of about 150,000 mg/L suspended solids to reduce the risk of losing some fine gold from the sluice. Environment Canada (1983) concluded that suspended-solids concentrations up to at least 60,000 mg/L are not an important consideration. Lin (1980) reports the recovery of -48+65 mesh gold in a sluice box is not adversely affected by a sluicing-water solids concentration of 10 percent by weight (107,000 mg/L) and concludes that the recovery of coarser gold will also be unaffected. Russian research reported by Zamyatin and others (1975) notes that sluice water should not exceed 160,000 mg/L because of gold losses and there is no recovery loss up to 30,000 mg/L suspended solids. They also note that the smallest gold grain size recoverable at 160,000 mg/L is 150 mesh and that the smallest gold grain recoverable at 70,000 mg/L is 200 mesh.

Because of the wide range in suspended solids concentrations reported to affect gold recovery (31,000 to 160,000 mg/L) and because of the general lack of supporting evidence for a specific level of total suspended solids that affects gold recovery, two pilot-test sluice-box studies were conducted in 1984 to determine the approximate level of suspended solids that would cause gold migration or loss (Peterson and others, 1984; Shannon & Wilson, 1985a).

These tests were run using a sluice box that was 6 in. wide and 8 ft long, with ribbed rubber matting and expanded metal riffles (figs. 1 and 2). The 8 ft of riffles were cut into four 2-ft-long sections so that each section could be cleaned separately to allow measurement of gold migration within the sluice box. All pay dirt was screened to minus 3/4 in., and one set of six test runs was made with washed pay dirt, where the pay dirt was run through the box three times to remove all native gold before conducting the tests (Shannon & Wilson, 1985a). Another set of six test runs was made with unwashed pay dirt (Peterson and others, 1984). Salted gold was added to all 12 test runs. Testing was started with a known quantity of -30+60 mesh gold. Recovery of +30, -30+50, and -50+80 mesh gold was measured for each riffle section after each test run. The average Corey shape factor of the gold used was 0.26 for the -30+40 mesh gold, 0.29 for the -40+50 mesh gold, and 0.36 for the -50+70 mesh gold.

There was a low correlation between the influent total suspended-solids concentration and gold migration (fig. 3); gold migration and loss were negligible in these 12 test runs. Figure 3 is arranged in order of increasing total suspended-solids concentrations ranging from 30 to 194,000 mg/L, which is not the order in which the runs were made. Ninety-five percent or more of the -50+80 mesh gold was caught on the first riffle section, and the amount lost from the box was 0.6 percent or less during these runs. Therefore, it was concluded that even concentrations of total suspended solids as high as 200,000 mg/L do not cause significant gold loss at the run durations and water duties of the tests.

CLAY AND VISCOSITY

Clay concentrations and viscosity levels in the sluice wash water were then hypothesized to be a possible cause of gold migration and loss.

The types of clay in effluent samples were determined by X-ray diffraction to be kaolinite, illite, chlorite, and lepidolite. The concentration of clay-sized particles was also determined. The upper limit for the size of a clay particle is generally referenced as between 0.002 and 0.005 mm. For the pilot studies, 0.002 mm was used as the upper limit of clay-sized particles, which was determined by using a hydrometer. The concentration of clay-sized particles for each run was calculated by multiplying the percentage of clay-sized particles by the total suspended-solids concentration. The concentration of clay-sized particles in the 12 test runs ranged from zero to about 40,000 mg/L. Figure 4 displays a low correlation between the concentration of clay-sized particles and gold migration. Because of this situation, additional work was done to assess the affects of using clay as opposed to simply clay-sized particles and to assess the affects of viscosity.

A series of kaolin, illite, and bentonite clay-water solutions were made and viscosity measured on each solution using a Fann Viscometer. Figure 5 displays the results of these measurements (Peterson, 1984) along theoretical curves for bentonite, subbentonite, and native clay (Magcobar, 1977). Of the three clays tested, bentonite is the most expansive and displays the highest viscosity levels at the lowest solids concentration. Additional pilot test runs were then conducted using bentonite and kaolin to determine whether true

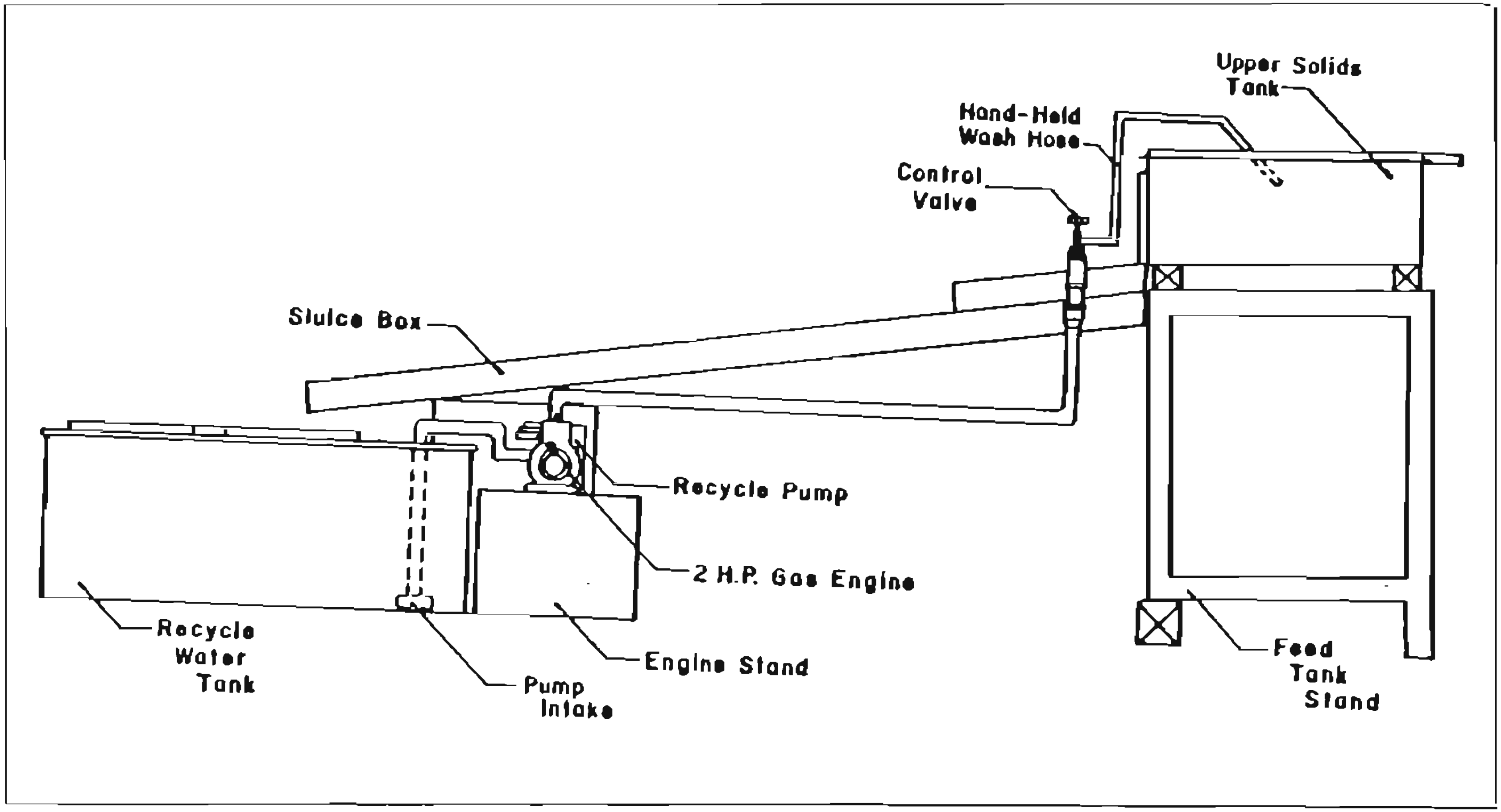

Figure 1. Pilot-test recycle facility (side view), scale: 1/2 in. = 1 ft.

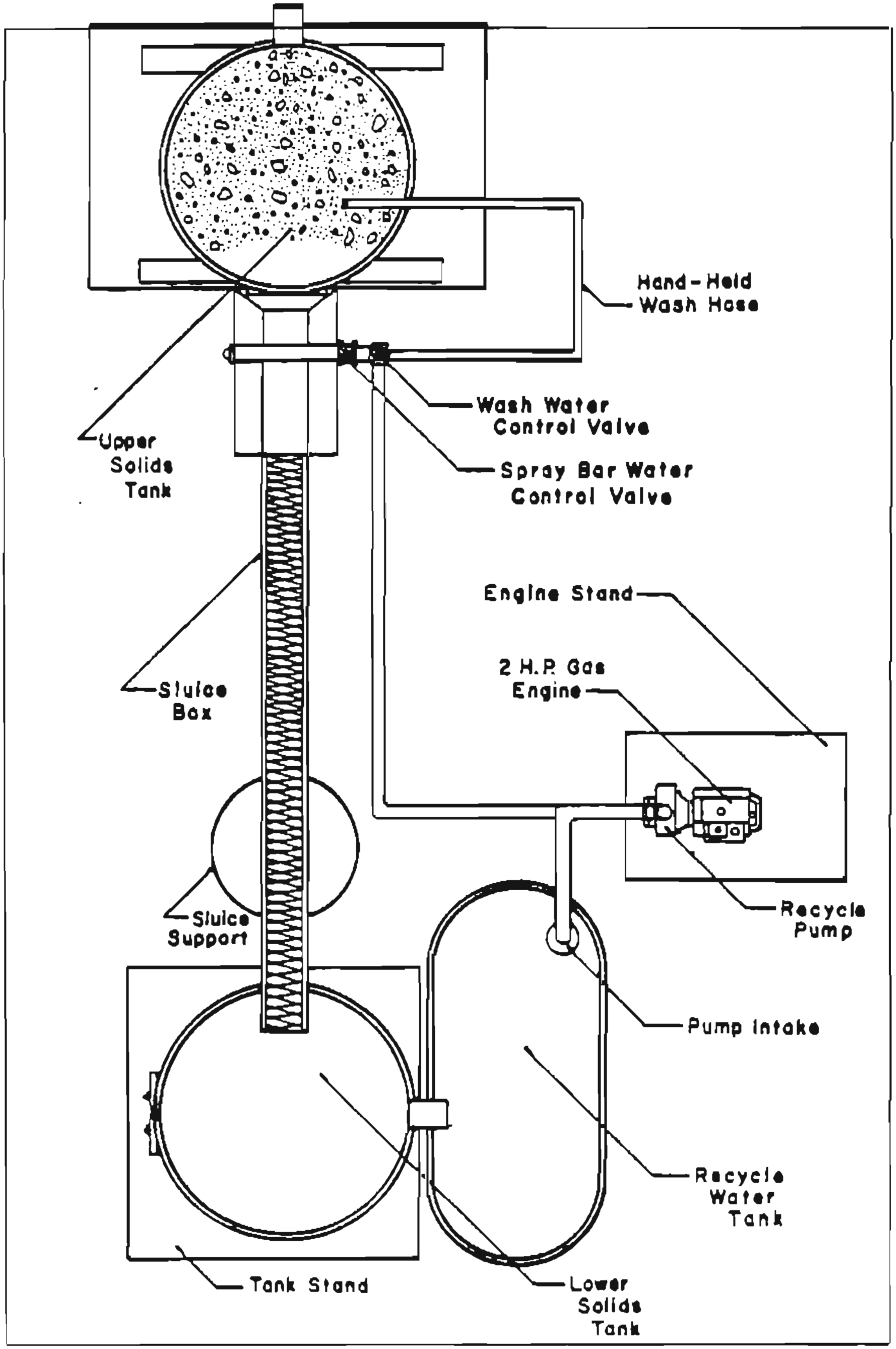

Figure 2. Pilot test, recycle facility (plan view), scale: ½ in. = 1 ft.

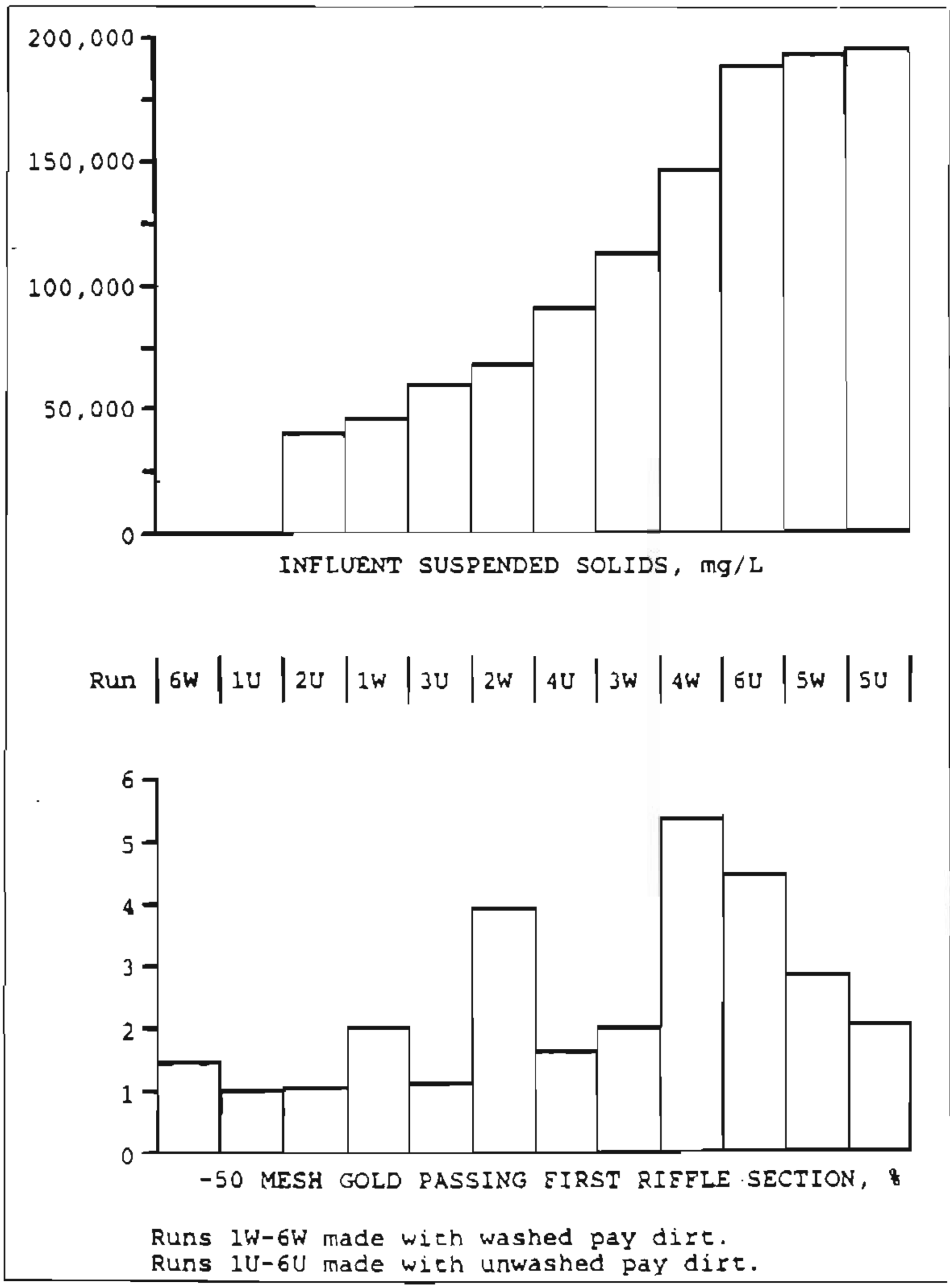

Figure 3. Influent suspended solids vs gold migration.

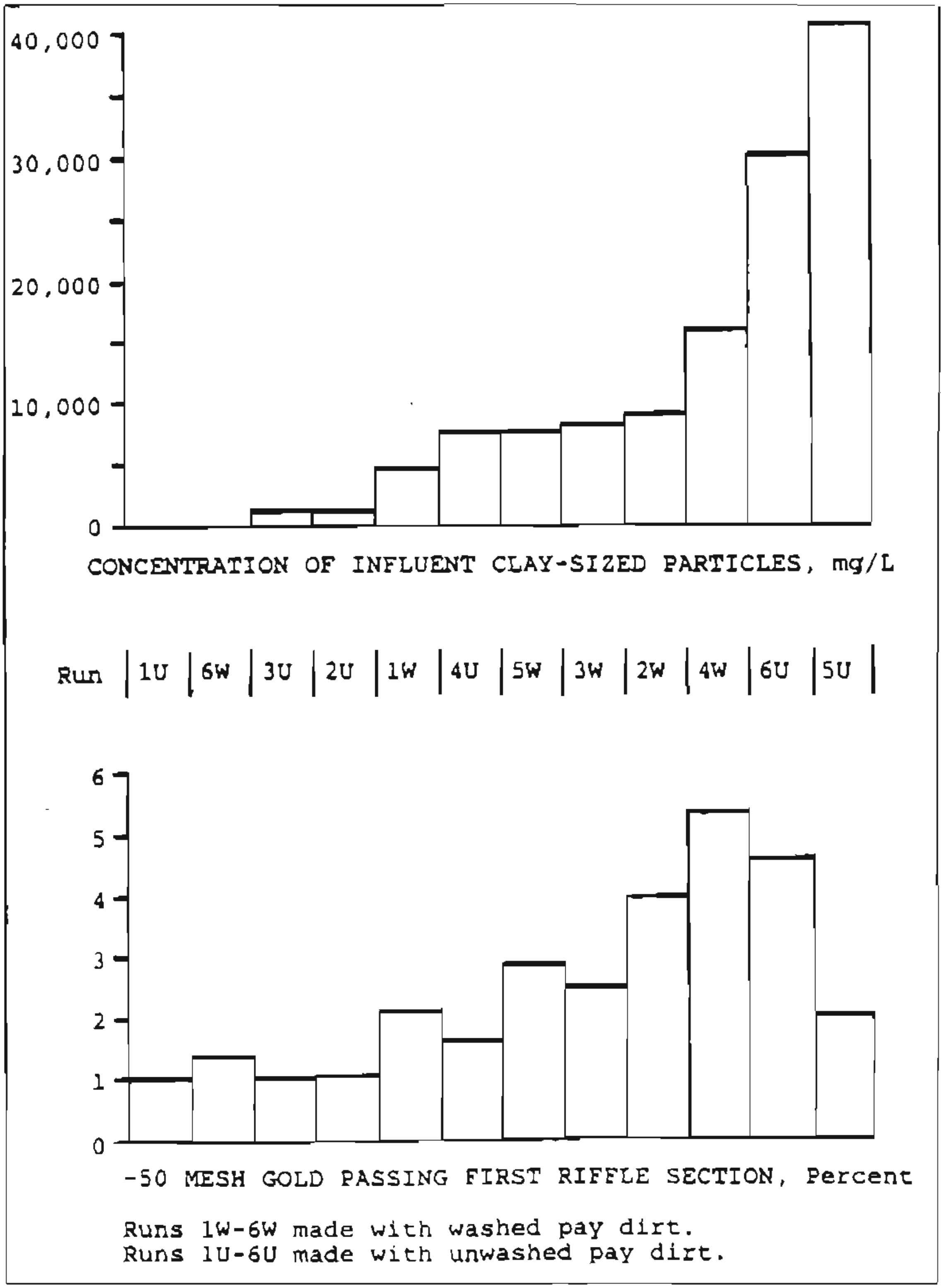

Figure 4. Influent clay-sized particles vs gold migration.

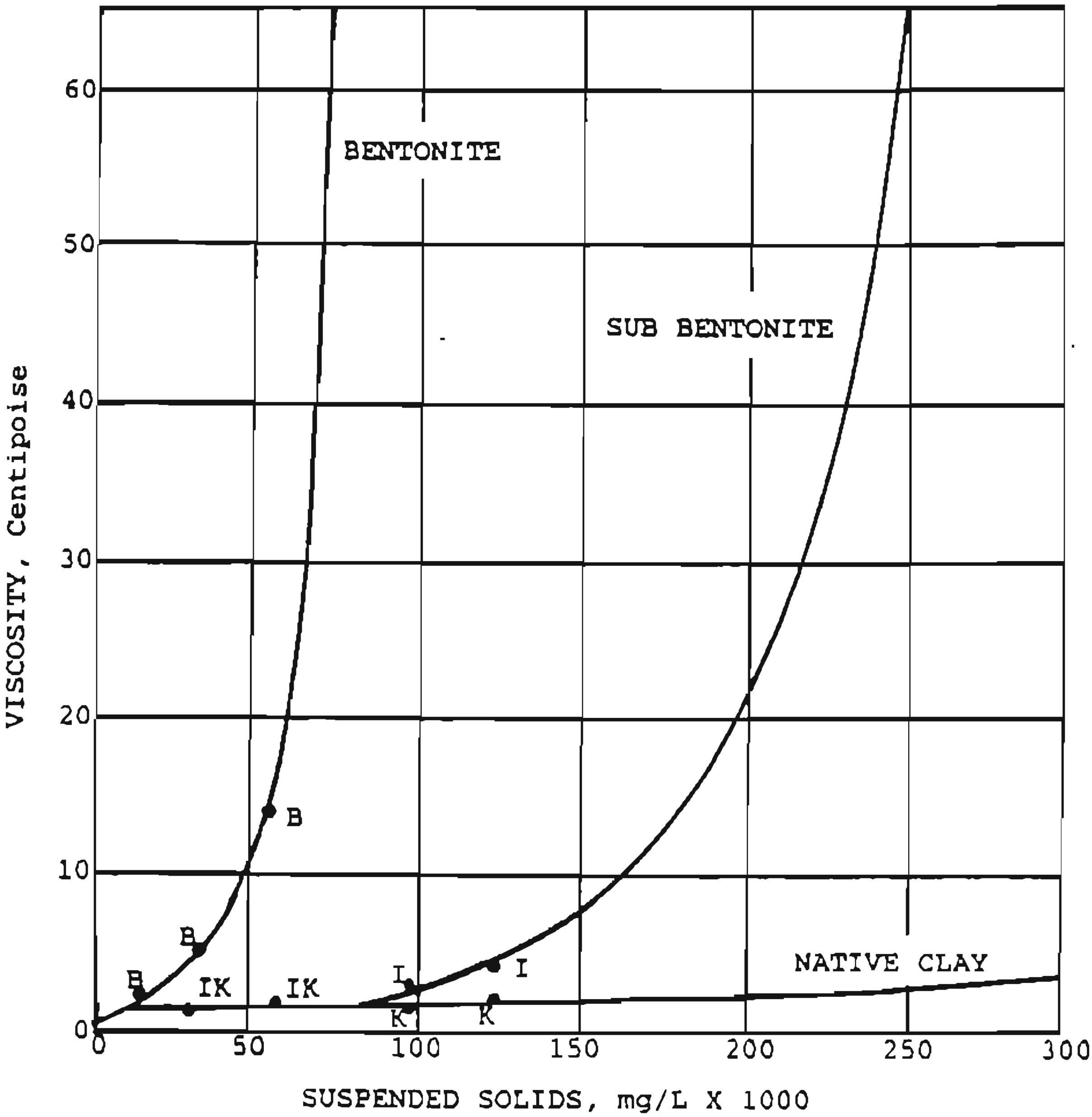

Figure 5. Viscosity vs suspended solids for different clays. Measured levels of viscosity for bentonite (B), illite (I), and kaolin (K) plotted on theoretical curves.

clays, rather than simply clay-sized particles, would cause gold migration. The kaolin and bentonite runs were made with about 100,000 mg/L total suspended solids, but the concentration of true clay was only 32,000 mg/L for the kaolin run and 60,000 mg/L for the bentonite run. The rest of the total suspended solids in these runs was made up of nonclay particles less than 0.002 mm. One additional run was made using Drispac, a viscosity builder used by the drilling industry, to test the effect of viscosity without clay.

Viscosity at the run temperature during the kaolin run was 3.2 centipoise (cp), 14 cp for bentonite, and 35 cp for the Drispac run. Gold migration of the -50 mesh gold in the sluice box for these three runs appears in figure 6. The difference in gold migration is readily apparent in all four riffle sections. Of the total -50 mesh gold recovered in the kaolin run, 98.9 percent was caught in the first riffle section, 1.1 percent in the second, and a trace in the third, with less than 0.1 percent in the fourth riffle section. These percentages are calculated based on the total amount of gold caught in the box because losses in these runs were very low. Drispac resulted in only 12.3 percent of the gold caught in the first riffle section, another 12.3 percent in the second, 11.1 percent in the third, and 9.5 percent in the fourth riffle section; 54.8 percent of the gold used passed through the box and was located in the solids settling tank, the recycle tank, the hose from the recycle pump to the upper solids feed tank, and in the upper feed tank. From these data, it appears that -50 mesh gold was just beginning to migrate in the bentonite run (viscosity, 14 cp) and moved significantly in the Drispac run, where viscosity was 35 cp. Therefore, it appears that a viscosity of about 14 cp or slightly higher will cause gold migration and loss.

Does this mean that viscosity at operating mines in Alaska can cause gold migration and loss? Available evidence suggests this is not the case. Figure 7 displays the range of temperatures and viscosities observed in sluice-effluent samples from six mines and shows a maximum viscosity of 5 cp, or about one-third the viscosity level hypothesized to cause gold migration. However, 'makeup' water diluted the recycle water at the mines used for this analysis.

Theory suggests that clay particles would concentrate in a zero-discharge, high-rate-recycle pond as the mining season progresses. Even if the clay concentration does increase, the type of clay will be an important contributing factor controlling the increase of viscosity. It appears from figure 5 that only bentonite is expansive enough to cause a significant increase in viscosity, and the concentration of bentonite would have to exceed approximately 60,000 mg/L to cause gold migration. This situation is unlikely at operating mines because: 1) the maximum concentration of clay-sized particles measured in a sluice effluent so far is only 15,400 mg/L; 2) bentonite has not been found at any of the sites studied; and 3) the temperature of recycle water would increase during the season, thereby offsetting some of the potential increase in viscosity.

CONCLUSIONS

On the basis of these two pilot tests, high total suspended-solids concentrations, clay, or viscosity are not the major causes of gold migration and loss. However, this conclusion is qualified because the pilot tests were operated under optimum conditions for a number of factors affecting gold recovery. According to Shannon & Wilson (1985b), gold recovery in a sluice depends mostly on the characteristics of three factors:

The gold: Its particle size and shape, porosity, and hydrophobicity.

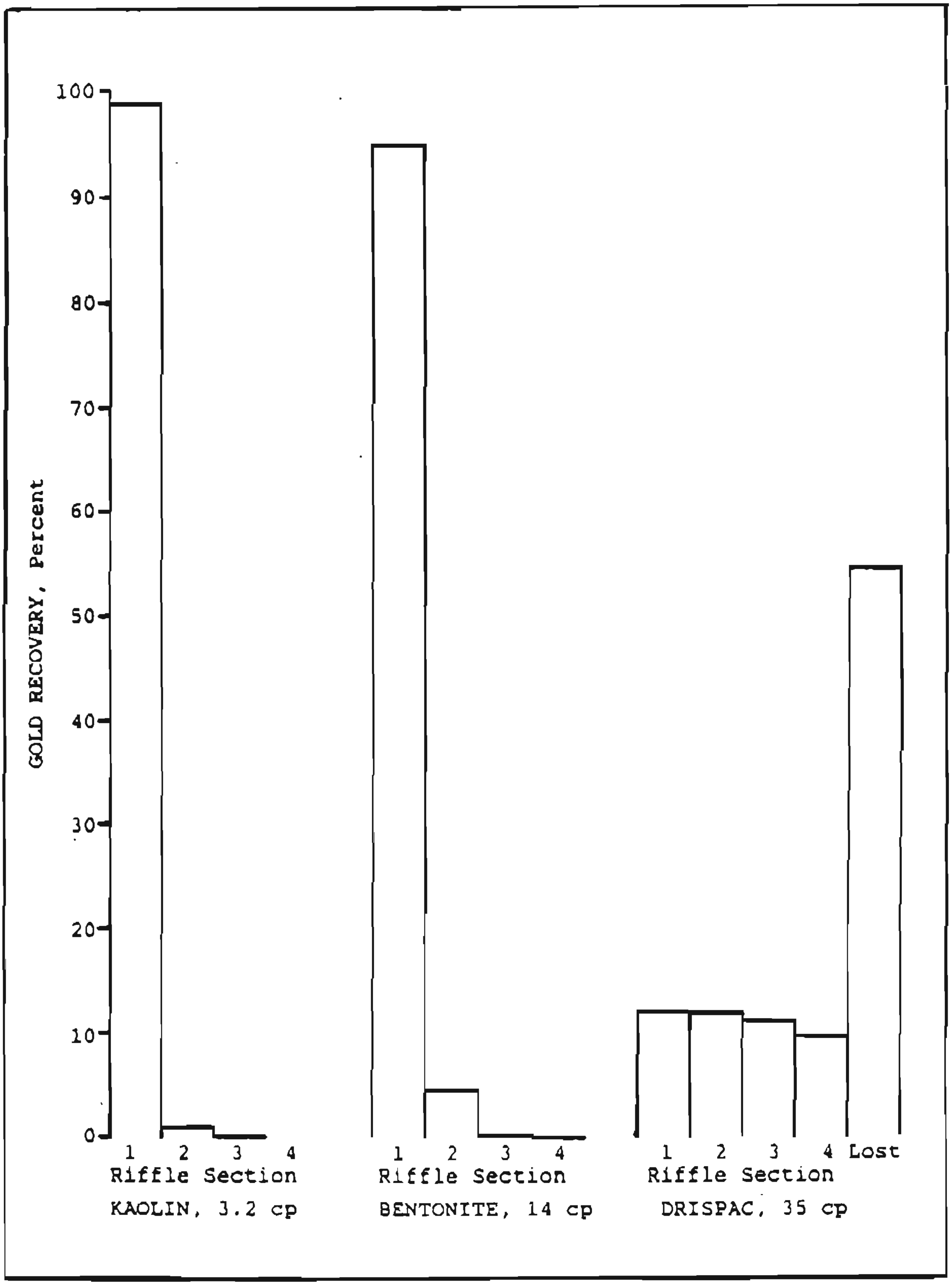

Figure 6. Gold migration vs viscosity.

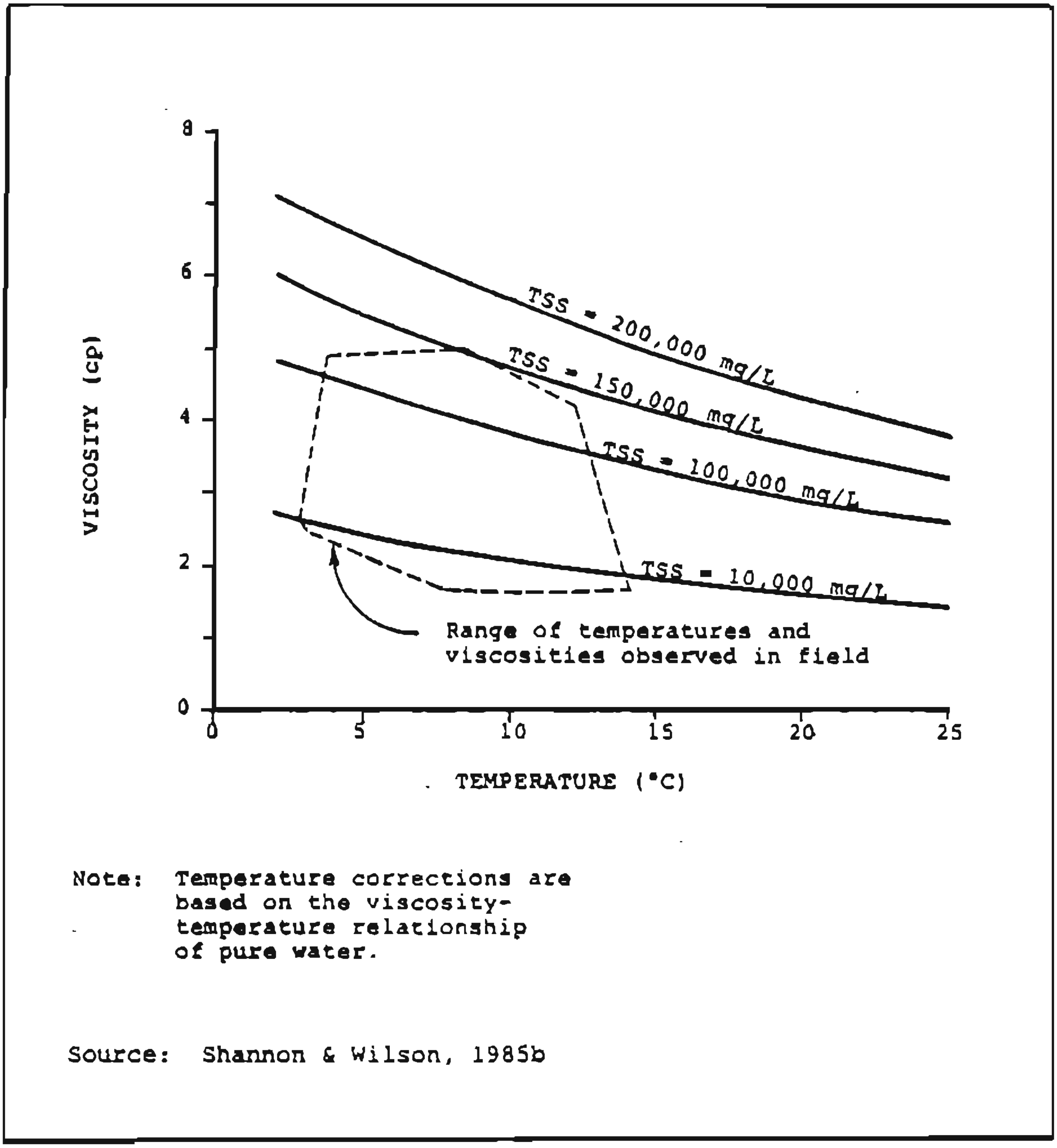

Figure 7. Viscosity relationships, sluice-effluent samples.

The sluice: Its slope, length, width, riffles, and mat; paydirt feed size and rate; sluice loading; riffle packing; and time between cleanups.

The wash water: Its washing effectiveness; the water duty; steady flow; and viscosity.

The pilot tests were conducted at nearly optimum conditions for most of the sluice and wash-water factors. For example, figure 8 displays sluice performance versus water duty for three ranges of gold particle sizes adapted from information presented by Poling and Hamilton (1987). The water duties used during pilot testing ranged from 0.19 to 0.56 and averaged 0.24 $yd^3/1000$ gal wash water. It was readily apparent that water duties used during pilot testing provided an optimum condition for maximum gold recovery. These pilot-test water duties are lower than water duties found at operating mines. The range of the sluice-water duties measured at mine sites during the Placer Mining Demonstration Grant Project ranged from 0.36 to 1.2 $yd^3/1,000$ gal, comparable to bank-water duties of 0.4 to 1.7 $yd^3/1,000$ gal (Peterson and others, 1987). Pilot studies were also conducted with expanded metal riffles, which provide better recovery than Hungarian riffles (fig. 9).

Finally, the time between cleanups of the pilot tests was short, ranging from 14 to 45 min. Figure 10 displays gold recovery versus time between cleanups based on data represented by Zamyatin and others (1975). For these data, the gold loss ranges from 4.7 to 8 percent at the 20-hr cleanup compared to the 2-hr cleanup. Because of the short time between cleanups (or short-run durations), there was no riffle packing in the pilot tests.

In summary, the pilot-scale sluice-box tests were conducted under nearly optimum conditions for all the sluice- and wash-water factors (except riffle packing), that affect gold recovery in a sluice box. The major conclusion of this work is that total suspended solids, clay, and viscosity do not directly cause gold migration and loss by themselves. However, these parameters may play a role in riffle packing during longer run durations or those with higher water duties.

Another series of pilot-test runs was made in 1986 using longer run durations and a test-water duty of 0.5 $yd^3/1,000$ gal (Peterson and others, 1986).. The major purpose of this testing was to assess the effects of riffle packing on fine-gold recovery (see p. 75).

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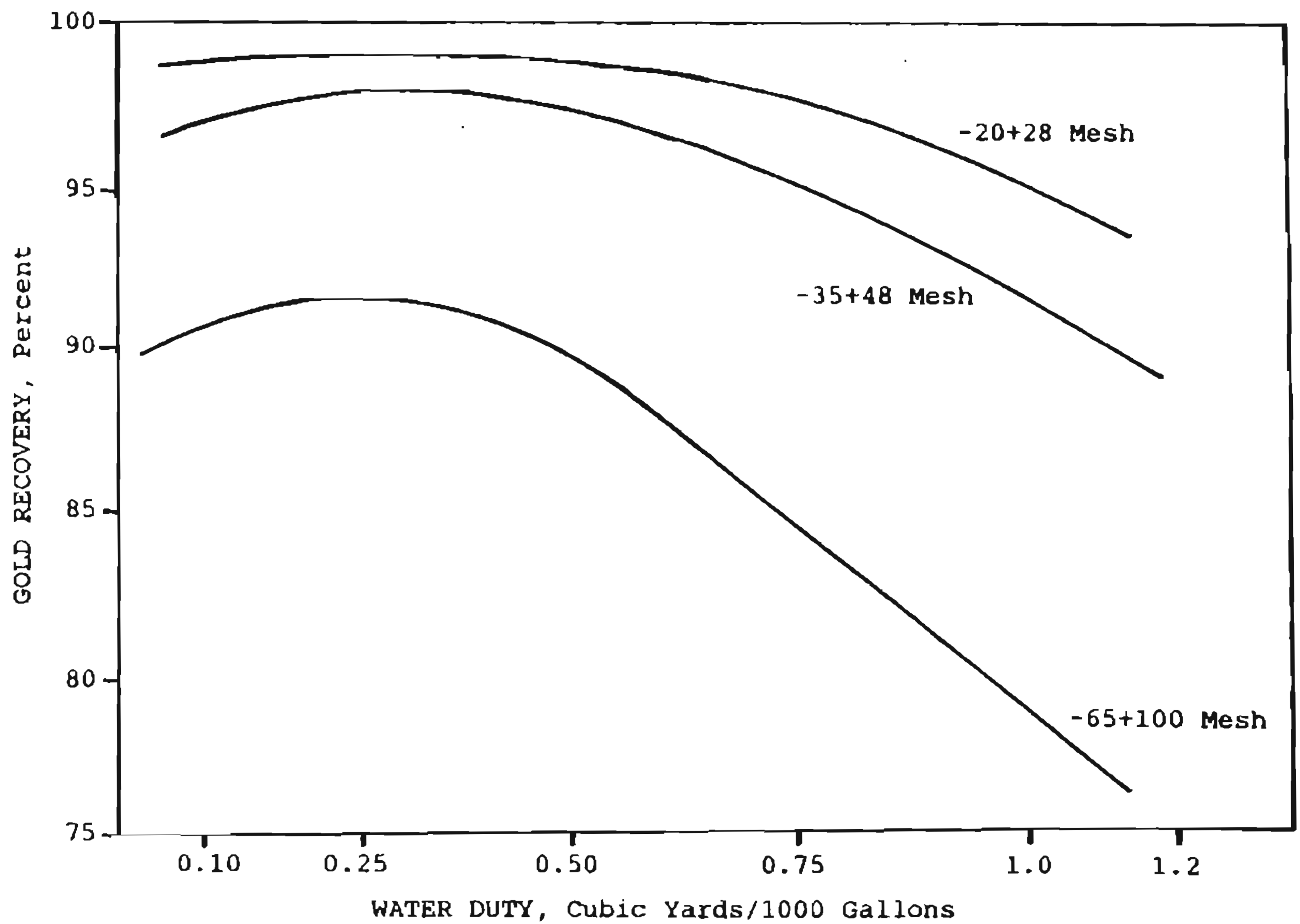
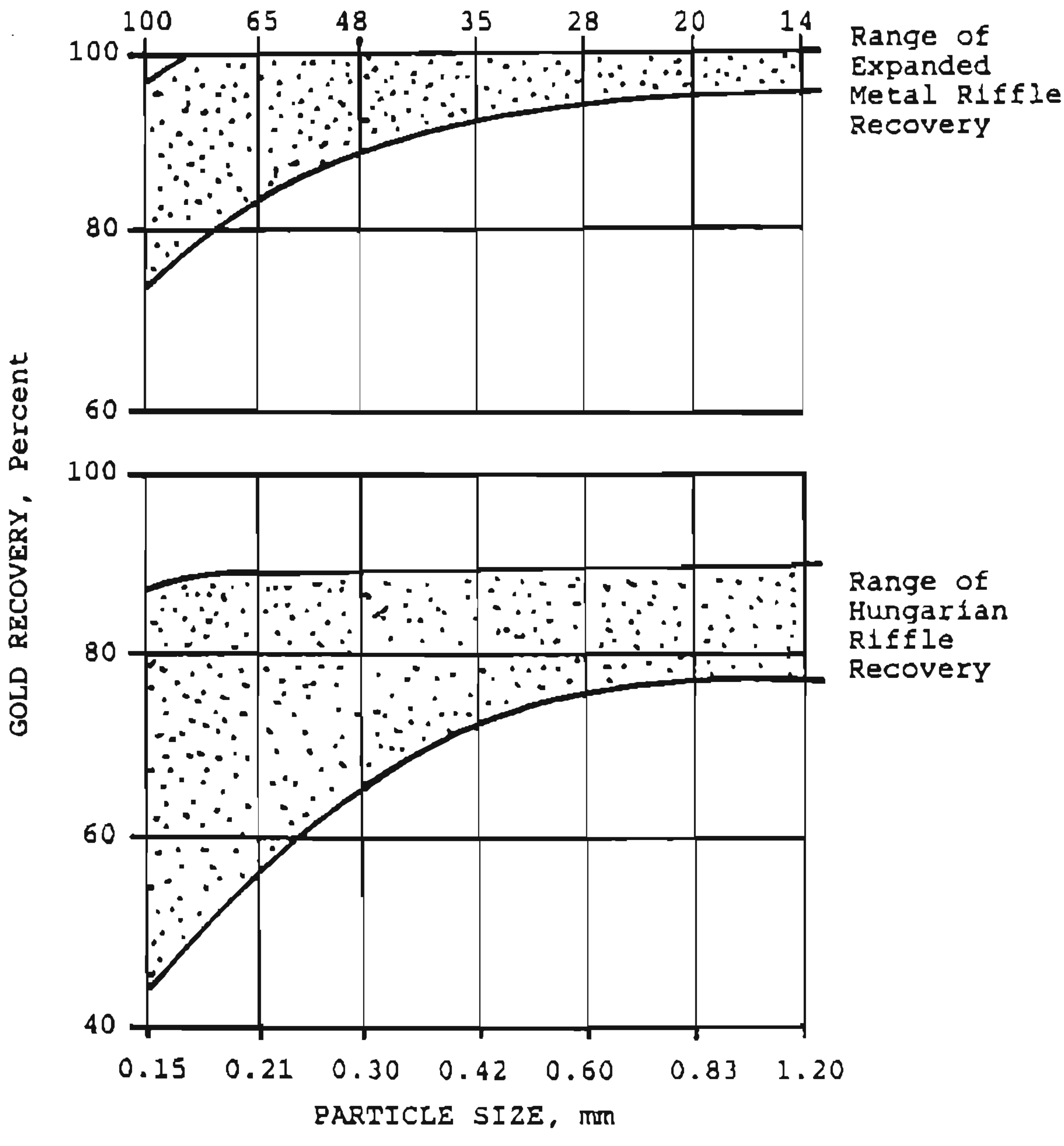


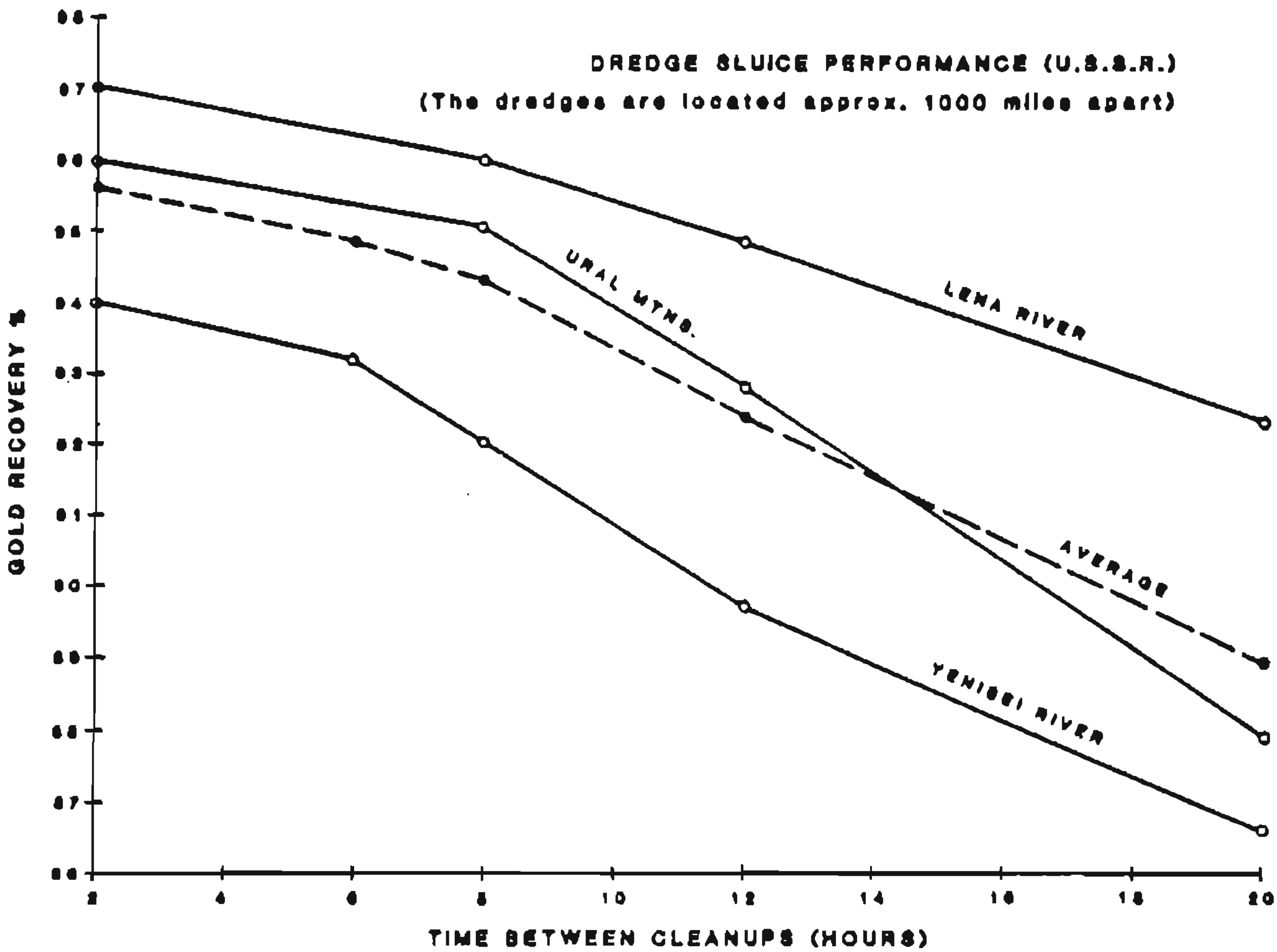
Adapted from Poling and Hamilton, 1987.

Figure 8. Gold recovery vs. water duty.



Source: Poling and Hamilton, 1987.

Figure 9. Gold recovery vs particle size for expanded metal and hungarian riffles.



Source: Zamyatin and others, 1975.

Figure 10. Gold recovery vs time between cleanup.

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RIFFLE PACKING, SUSPENDED SOLIDS, AND GOLD LOSS IN A SLUICE BOX

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INTRODUCTION

Virtually all placer miners have experienced riffle packing and its dreaded consequence, gold loss. Even those miners who use jigs or other recovery systems face conditions not unlike riffle packing in a sluice box (Tsigonis Engineering & Environmental Services, 1985). Although riffle packing has been affecting gold recovery for generations, ever since placer mining began several thousand years ago, little quantitative information exists about it because it is so hard to measure.

In 1984, testing with a pilot-scale sluice box indicated that total suspended solids (TSS), concentration of clay-sized particles, and viscosity are not major causes of gold migration or loss from sluice boxes at Alaska placer mines (Shannon & Wilson, 1985; Peterson and others, 1984).

In 1986, Larry Peterson, Gary Nichols, and the author ran tests to determine the effect that varying levels of total suspended solids in the sluice feed water had on riffle packing and gold recovery in a pilot-scale sluice box (Peterson and others, 1986).

The guidance of Willis Umholtz, EPA's project manager, is gratefully acknowledged. The invaluable assistance of Ron Roman, who not only allowed us to set up our test facility at his mine site on Fish Creek near Fairbanks, but provided us with the paydirt we used in testing, is also very gratefully acknowledged.

TEST FACILITY

A schematic of the test facility is shown in figure 1. About 100 yd^3 of paydirt was stockpiled and dry screened as needed through a 3/4-in.-deep vibrating screen. Grain-size analyses (figs. 2 and 3) of the screened material from each test run show that the paydirt was relatively uniform.

Minus 3/4-in.-mesh paydirt was fed from the hopper onto a slick plate, where it was washed into a hydraulic lift made from a 2½-in.-diam Keene suction-dredge nozzle. The hydraulic lift carried the paydirt to the head of the sluice.

The sluice was the same one used for the 1984 pilot-scale testing. It is 8 ft long and 6 in. wide and was set at a 1¼-in./ft drop, which is a 10-percent slope, for all the testing.

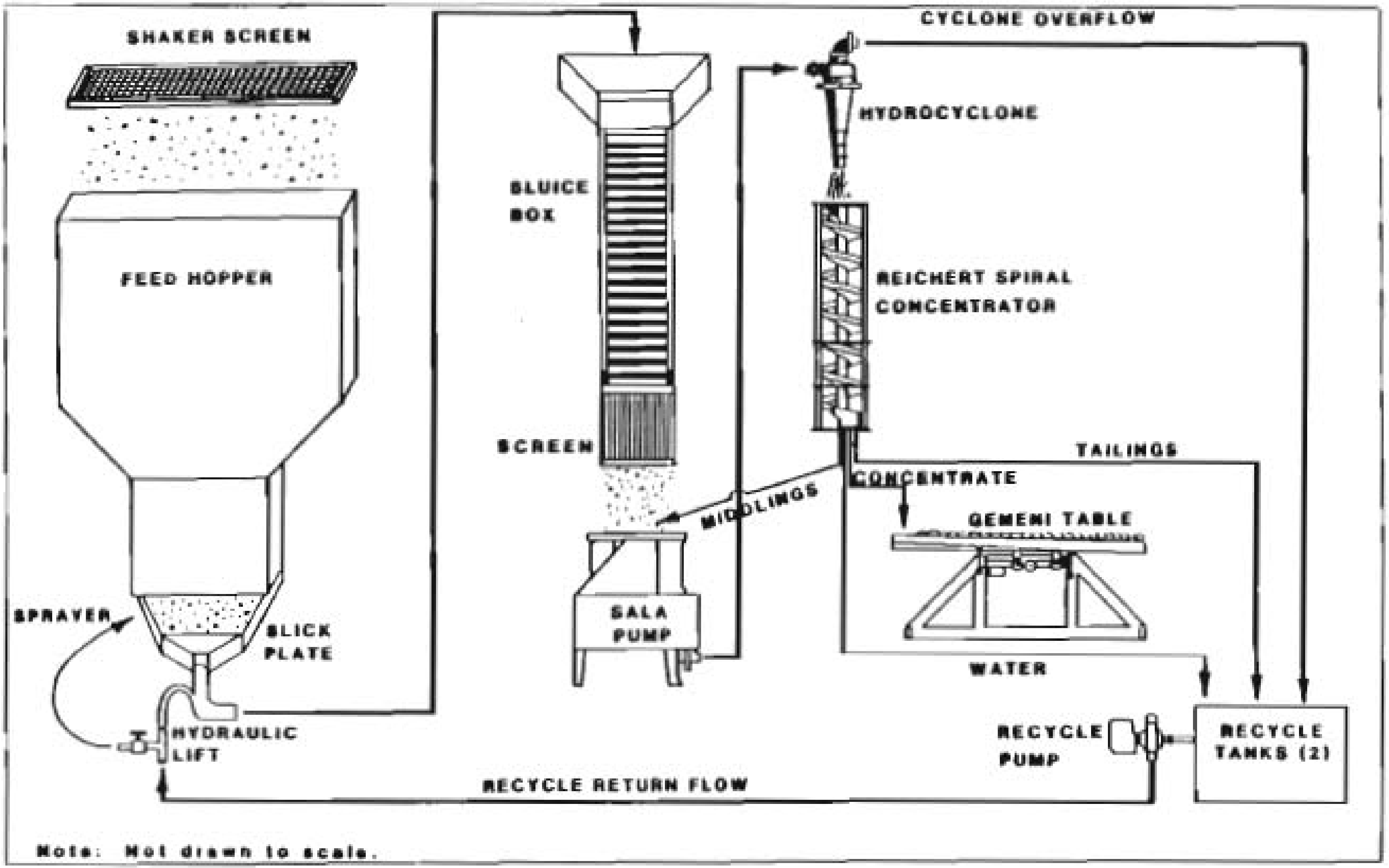

Figure 1. Recycle flow schematic.

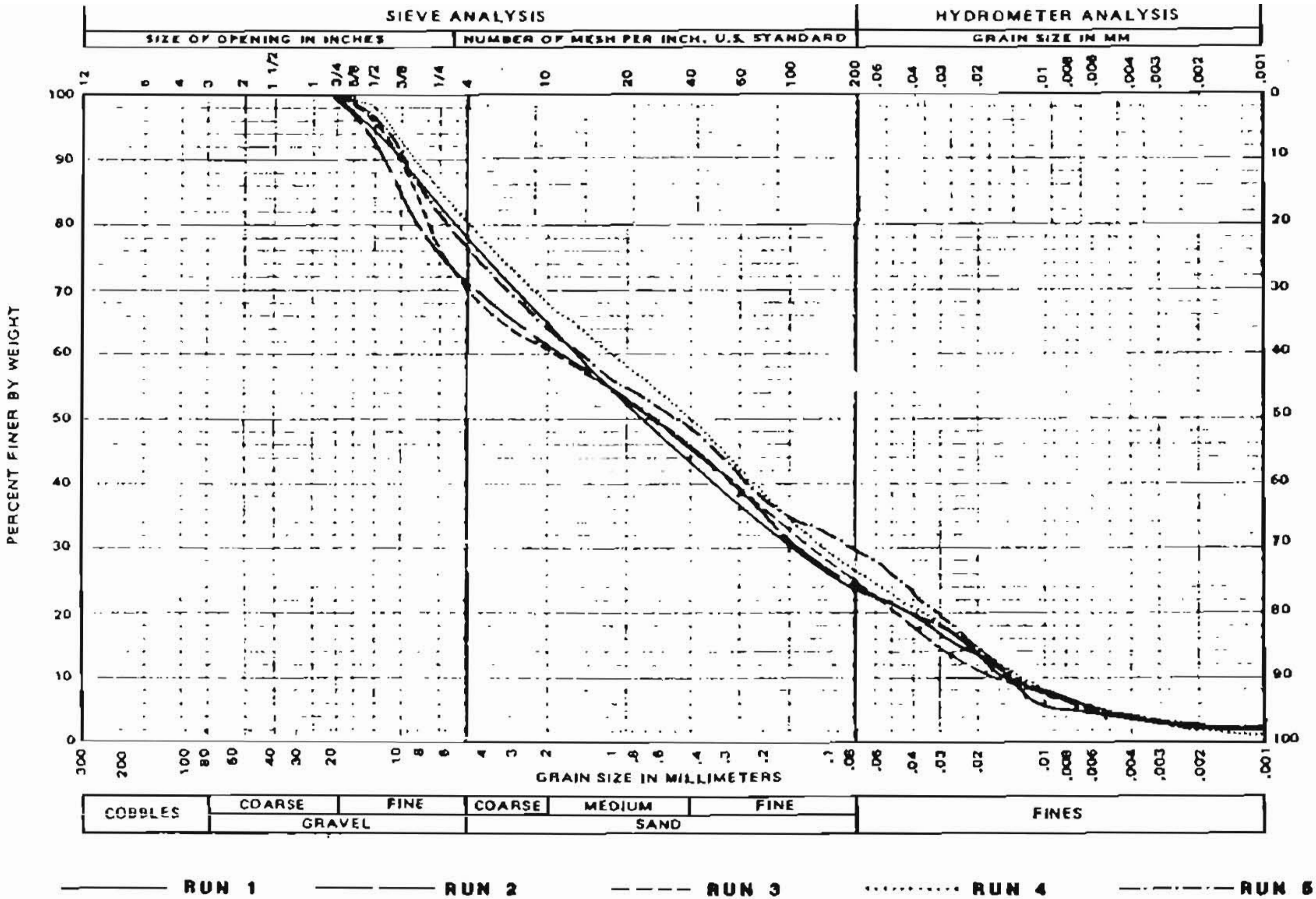

Figure 2. Paydirt grain-size distribution.

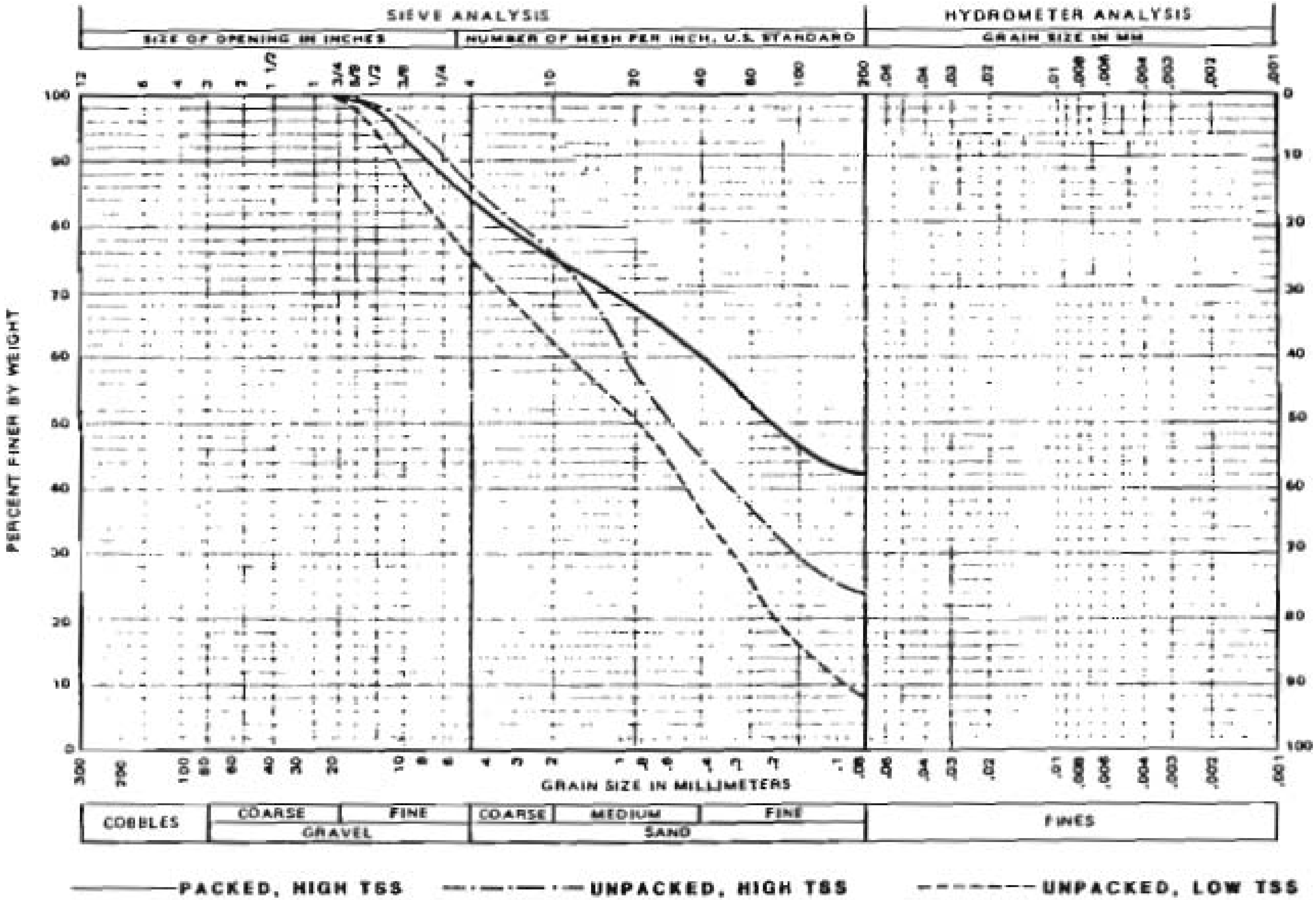

Figure 3. Grain-size distribution of packed and unpacked riffles.

The riffles were constructed in three sections, and cleanups were done by section. Riffle 1 covered the top one-sixth of the box, riffle 2 covered the next one-third, and riffle 3 covered the bottom half of the box. In 1984, we used four equal sections. To allow a better determination of when gold begins moving down the box, we changed the riffle configuration.

Water was supplied to the hydraulic lift by a 10-hp, gasoline-powered water pump at a rate of 40 gpm. The first three test runs were made with 100-percent recycled water, run 4 was made with partially recycled water, and run 5 was made with clear water and no recycle.

Because riffle packing causes gold loss, gold loss was used as a measure of riffle packing. Gold loss was measured by means of a secondary recovery system that processed sluice tailings.

At the bottom of the sluice was a 1-ft-long section of wedge-wire screen. The wedge-wire ran parallel to the flow; spacing between the wires was 2 mm. Most of the water and minus 10-mesh material from the sluice fell through the screen and into a 1½-in.-diam Sala slurry pump. The pump lifted the material to a hydrocyclone situated above a single-start Reichert Mark VII spiral (fig. 4). Underflow from the cyclone flowed down the spiral while overflow went back into the recycle tanks.

Concentrate from the spiral flowed onto a Gemini shaking table, where gold lost by the sluice was recovered. Middlings from the spiral were returned to the Sala pump to maintain the proper flow rate and pulp density for the spiral. The water and tailings splits flowed back into the recycle tanks.

This system provided continuous monitoring of gold loss and indicated, within a few minutes, when the riffles had become packed to the extent that gold was being carried through the box.

TEST RUNS

Each test run was made with a fresh portion of screened paydirt. Water duty was close to 0.5 yd^3 of screened paydirt per thousand gallons of water during all five runs.

Although gold-bearing paydirt was used for testing, we needed to be sure there was a statistically significant quantity of gold in the sluice box. Therefore, fine gold was added to the paydirt as it passed across the slick plate. Corey's shape factor of the salted gold ranged from 0.12 (which is quite low) to 0.42, with a mean of 0.26.

The gold was divided into five equal parts of approximately equal weights of 50- to 70-mesh and 70- to 100-mesh. All gold was processed across the Gemini table prior to sluice testing; only those particles recovered by the table were used in the subsequent tests. Gold was added in increments to the paydirt during each run. About 2.5 g gold was added every 20 min throughout each run.

Figure 4. Pilot-test plant for fine-gold recovery and riffle packing. For each test run, the sample gold was removed from the sluice concentrate and tailings. The Reichert MKVII spiral and Gemini table were used to measure fine-gold recovery and loss.

After each run, cleanup was done by riffle section. Sluice concentrate from each riffle was washed into a 5-gal bucket, screened into size ranges of plus ½-in., ½-in. to 10-mesh, 10- to 20-mesh, and minus 20-mesh. Each of the plus 20-mesh-size fractions was panned and the minus 20-mesh fraction was run across the Gemini table to separate the gold.

The suspended-solids level for each run was adjusted by either adding settling-pond fines from Ron Roman's lower settling ponds or by diluting the solids level from the previous run with clean water.

WATER QUALITY

Table 1 summarizes the water-quality data for all five runs. For run 1, influent-suspended solids averaged 249,000 mg/L and expanded metal riffles were used; the run lasted 315 min, when we noticed a marked increase in the

Table 1. Pilot-test water-quality data for composite samples.

Parameter*	Run 1	2	3	4	5
			Sluice influent		
Suspended solids	249,000	285,000	421,000	62,300	348
Turbidity	59,000	74,000	76,000	19,000	17
Settleable solids	280	400	420	120	<0.1
Specific gravity	1.155	1.178	1.282	1.040	1.000
Viscosity at run temp.	2.8	3.5	4.1	1.8	1.0
Run duration	315	315	60	315	315
Screened-water duty	0.5	0.5	0.5	0.5	0.5
Bank yard water duty	0.6	0.6	0.6	0.6	0.6
			Sluice effluent		
Suspended solids	292,000	313,000	469,000	95,400	29,700
Turbidity	67,000	80,000	88,000	21,000	4,000
Settleable solids	350	440	440	130	36
Specific gravity	1,182	1.200	1.296	1.062	1.020
Viscosity at run temp.	3.2	3.5	6.1	2.0	1.5

*Units:
- Suspended solids - mg/L
- Turbidity - NTU
- Settleable solids - ml/L
- Specific gravity - gm/cc at 20 c
- Viscosity - cp (centipoise)-gm mass/cm sec.
- Run duration - Min
- Water duty - yd^3/1000 gal (cubic yards of paydirt sluiced using 1000 gallons of water)

amount of gold showing up on the Gemini table, indicating that the riffles were packed.

After run 1 the expanded-metal riffles were replaced with Hungarian riffles made from 1½-in. angle iron. This was done for four reasons:

1) To compare packing between expanded-metal and Hungarian riffles

2) Because Hungarian riffles reportedly pack faster than expanded-metal riffles

3) Hungarian riffles are commonly used

4) We noted during run 1 that not as much concentrate accumulated in the expanded-metal riffles where the metal was not perpendicular to the direction of flow.

For run 2, influent TSS averaged 285,000 mg/L, settleable solids averaged 400 ml/L, and run duration was 315 min.

For run 3, the recycle water was as dirty as we could get it. Influent TSS were 421,000 mg/L, settleable solids were 420 ml/L, and run duration was 60 min.

For run 4, partial recycle was used, and influent TSS were reduced to 62,300 mg/L, settleable solids were 120 ml/L, and run duration was 315 min.

Run 5 was the clear-water run with no recycle. Influent TSS were 348 mg/L, settleable solids were zero, and run duration was 315 min.

Besides these five runs made to assess the effect of riffle packing on gold loss, three additional runs were made to measure the amount of material retained in the riffles and its grain-size distribution for packed and unpacked conditions.

RESULTS

Gold Recovery

Table 2 summarizes gold recovery by percent. During the five test runs, gold captured on the Gemini table was placed into separate containers, depending on when it showed up. For all but run 3, the gold caught during the last 25 min was kept separate from all gold caught during the first 290 min of the run. For run 3, which lasted 60 min, the gold caught on the Gemini table was divided into a 30-min segment and two 15-min segments.

Runs 1 and 2 were the same except for the type of riffles used. For run 1, with expanded metal riffles, a continuous slow rate of gold loss was observed on the Gemini table, whereas for run 2, practically all the gold loss occurred during the last 40 to 50 min. Apparently, expanded-metal riffles do not pack as completely as Hungarian riffles do; instead, they continuously allow a small amount of gold to pass through the sluice. The expanded-metal riffles were aligned in one direction only. Alternating the alignment of these riffles might reduce the opportunity for gold to pass through the box. Also, it appears that once the Hungarian riffles pack, they lose more gold than do the expanded-metal ones.

Run 2 lasted 315 min, whereas run 3 lasted only 60 min. Significantly more gold was captured in the first riffle section in run 3 than in run 2, despite a higher influent TSS average (421,000 mg/L to 285,000 mg/L). This indicates that neither the high TSS concentration of run 3 nor its viscosity (4.1 cp in the recycle water) adversely affected gold recovery---as long as

Table 2. Percent gold recovery.

	Riffle Section			Gemeni Table		
	1	2	3	A	B	C
Run 1						
+50	2.71	0.49	0.02	0.07	0.02	---
-50+70	31.66	12.52	0.58	0.12	0.05	---
-70+100	28.76	16.91	2.09	0.12	0.11	---
-100	1.85	1.52	0.35	0.03	0.02	---
Total	64.98	32.44	3.04	0.34	0.20	---
Run 2						
+50	0.89	0.28	0.01	0.02	0.02	---
-50+70	34.52	11.31	0.52	0.11	0.20	---
-70+100	30.24	16.52	1.83	0.17	0.42	---
-100	1.60	1.01	0.22	0.04	0.07	---
Total	67.25	29.12	2.58	0.32	0.72	---
Run 3						
+50	3.05	0.56	0.03	0.00	0.02	0.01
-50+70	37.81	6.08	0.34	<0.01	0.02	0.05
-70+100	33.91	12.99	1.69	<0.01	0.02	0.08
-100	1.83	1.19	0.27	0.00	<0.01	0.03
Total	76.60	21.82	2.33	0.01	0.06	0.17
Run 4						
+50	2.49	0.66	0.01	0.01	0.00	---
-50+70	39.83	8.31	0.03	0.01	<0.01	---
-70+100	31.78	13.97	0.13	0.02	<0.01	---
-100	1.67	1.06	0.02	<0.01	<0.01	---
Total	75.77	24.00	0.19	0.05	<0.01	---
Run 5						
+50	3.33	0.05	<0.01	<0.01	<0.01	---
-50+70	42.10	2.62	0.02	0.01	<0.01	---
-70+100	42.61	6.14	0.06	0.01	<0.01	---
-100	2.48	0.52	<0.01	<0.01	<0.01	---
Total	90.52	9.33	0.09	0.04	0.02	---

Note: Gemeni table designation A, B, and C are:
For runs 1, 2, 4, and 5, A is the first 4 hr, 15 min of operation and B is the last 25 min of operation.
For run 3, A is the first 30 min of operation, B is for 31 to 45 min, and C is for 46 to 60 min.

the riffles were not packed. However, the riffles began packing very quickly during run 3, as shown by the rapid increase in gold loss in just 60 min. The total gold loss in run 3 might have surpassed that of run 2 in another 15 min or so.

The primary difference between runs 2, 4, and 5 was influent TSS. Influent TSS concentrations averaged 285,000 mg/L for run 2, 62,300 mg/L for run 4, and 348 mg/L for run 5. For the test sluice used in this study, riffle packing did not occur in 315 min at an influent TSS of 62,300 mg/L but did occur at 285,000 mg/L, as shown by the gold loss of 0.72 percent during the last 25 min of run 2. These findings indicate that gold migration due to riffle packing is a real phenomenon and that, under conditions similar to this study, riffles become packed in about 5 hr at some influent TSS exceeding 62,300 mg/L. The specific influent TSS causing riffle packing and gold loss under full-scale mining conditions is unknown, and will probably vary with paydirt composition, sluice conditions, and time between cleanups, as it did here.

In runs 4 and 5, the major difference in gold recovery was in the distribution of gold in the sluice. For run 4, 75.8 percent of the gold remained in riffle 1; in run 5, 90.5 percent of the gold was found. These data clearly indicate that an influent TSS of 62,300 mg/L caused more gold to migrate down the sluice than clear water with an influent TSS of 348 mg/L. Again, the specific influent TSS concentration causing gold migration under full-scale mining conditions may lie within the range determined from this study, but will vary with paydirt composition, sluice conditions, and run duration.

RIFFLE PACKING

Three additional runs were made to measure the amount of material retained in the riffles and its grain-size distribution in both packed and unpacked conditions. One of those runs was done at high TSS and lasted 315 min, which is the length of time required for packing to occur at this solids level. The grain-size distribution curve for this run is shown in figure 3 by the solid line. The second run was made at the same high TSS but only lasted 60 min (dash-dot-dash line in fig. 3); the third (clear-water) run ran for 315 min (dashed line).

At the end of each run, riffle material was washed into buckets, sized, and weighed. Multiplying the total weight of material in the sluice by the percentages of plus 100-mesh material shows that the weight of material larger than 100-mesh was the same for both the packed and the unpacked high-TSS runs. The difference between packed and unpacked riffles appears to be the amount of material smaller than 100-mesh that was trapped in the riffles. In fact, twice as much material finer than 100-mesh was found in the riffles after 60 min sluicing with high-TSS water than after 315 min of sluicing with clear water. Three times as much material finer than 100-mesh was found in the riffles after sluicing for the same time (315 min) with high-TSS water rather than with clear water. Another important observation is that, when using water with very high TSS, the rate of accumulation of fine material is high initially and slows as sluicing continues. This means that, to avoid riffle packing, cleanup frequency must increase quickly as the TSS of the wash water increases.

LIMITATIONS

Pilot-scale studies have the advantages that test conditions can be more easily measured and controlled and testing can be done less expensively than under full-scale conditions. However, judgement must be used when applying pilot-test results to full-scale operations (figs. 5 and 6).

CONCLUSIONS

1. Using recycle water does affect gold recovery, but a well-designed and operated sluice box can capture 99 percent of the free gold down to at least 100-mesh, even when using very, very dirty water (421,000 mg/L TSS).

2. When using recycle water in the quality commonly found at Alaska placer mines, properly designed and operated sluice boxes can capture well over 99 percent of free gold down to at least 100-mesh.

3. Gold migration in a sluice box can be expected to occur before gold loss. Small-sized gold particles migrate more than larger particles.

4. Expanded-metal riffles appear to have a continuous, low-rate gold loss compared to Hungarian riffles although it may be possible to reduce this loss by alternating the direction of adjacent sections of expanded metal riffles.

5. The accumulation of fine material (minus 100-mesh) caused riffle packing in this study. The actual size of material that causes riffle packing in full-scale mining operations will vary from mine to mine. Frequent cleanups or frequent use of a hose to loosen and wash this material from the riffles may lessen gold loss caused by riffle packing.

Readers desiring further information on this testing program are encouraged to review Peterson and others, 1986.

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Figure 5. Typical Alaskan sluice plant at Eagle Creek. The plant includes a dozer trap and hopper, conveyor belt, vibrator classifier, and three-section sluice.

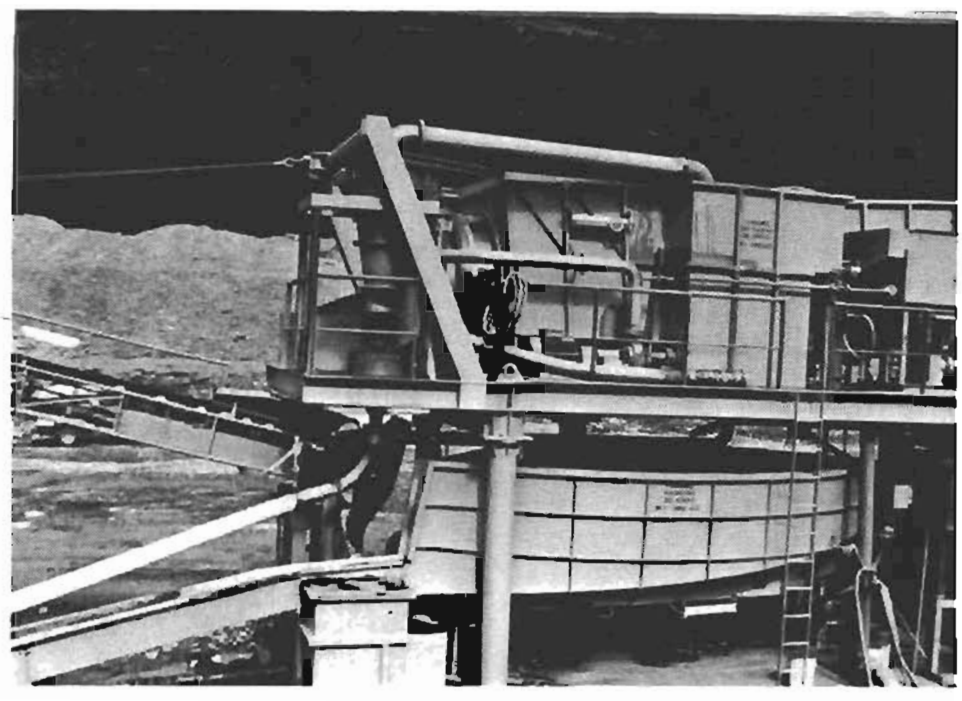

Figure 6. An IHC jig plant. Jig plants may be susceptible to packing.

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EXPERIMENTAL UNDERGROUND PLACER MINING ON WILBUR CREEK

by

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ABSTRACT

A modern underground placer mine on Wilbur Creek, Livengood, Alaska, has been in operation since October 1986.

This paper describes some of the aspects of placer mining in permafrost based on data available from Siberia and Yukon Territory and experience on Wilbur Creek. In particular, selection of mining method and stability of openings in very temperature sensitive ground are discussed. Technical and economic comparisons between surface stripping (hydraulicking) and underground mining (drifting) are provided.

The volume of muck, gravel, and bedrock that has to be mined on Wilbur Creek to produce unit quantity of gold was found to be 46.9 times greater for a surface mine than for drifting. The volume of silt and clay to be handled is over 108 times larger. The preliminary cost of underground mining (at \$20/hr labor and excluding access, exploration, insurance, and processing costs) on Wilbur Creek was \$11.30 yd^3 at 41-percent breakdown time and \$8.05 yd^3 for zero breakdown time.

INTRODUCTION

Wilbur Creek placer property has been mined since the early years of this century, first by drift mining and later by hydraulicking. The creek has produced several thousand ounces of gold. The deposit is a bench that lies west of the present channel of Wilbur Creek. The auriferous gravel and bedrock are covered with 65 to 135 ft of frozen overburden muck (fig. 1). Reserves, estimated at 313,000 yd^3, comprise a relatively high-grade deposit 2.5 mi long and 80 ft wide. The pay section consists of 5 ft of gravel and 3 ft bedrock. Although the auriferious gravels average 200 ft wide, an

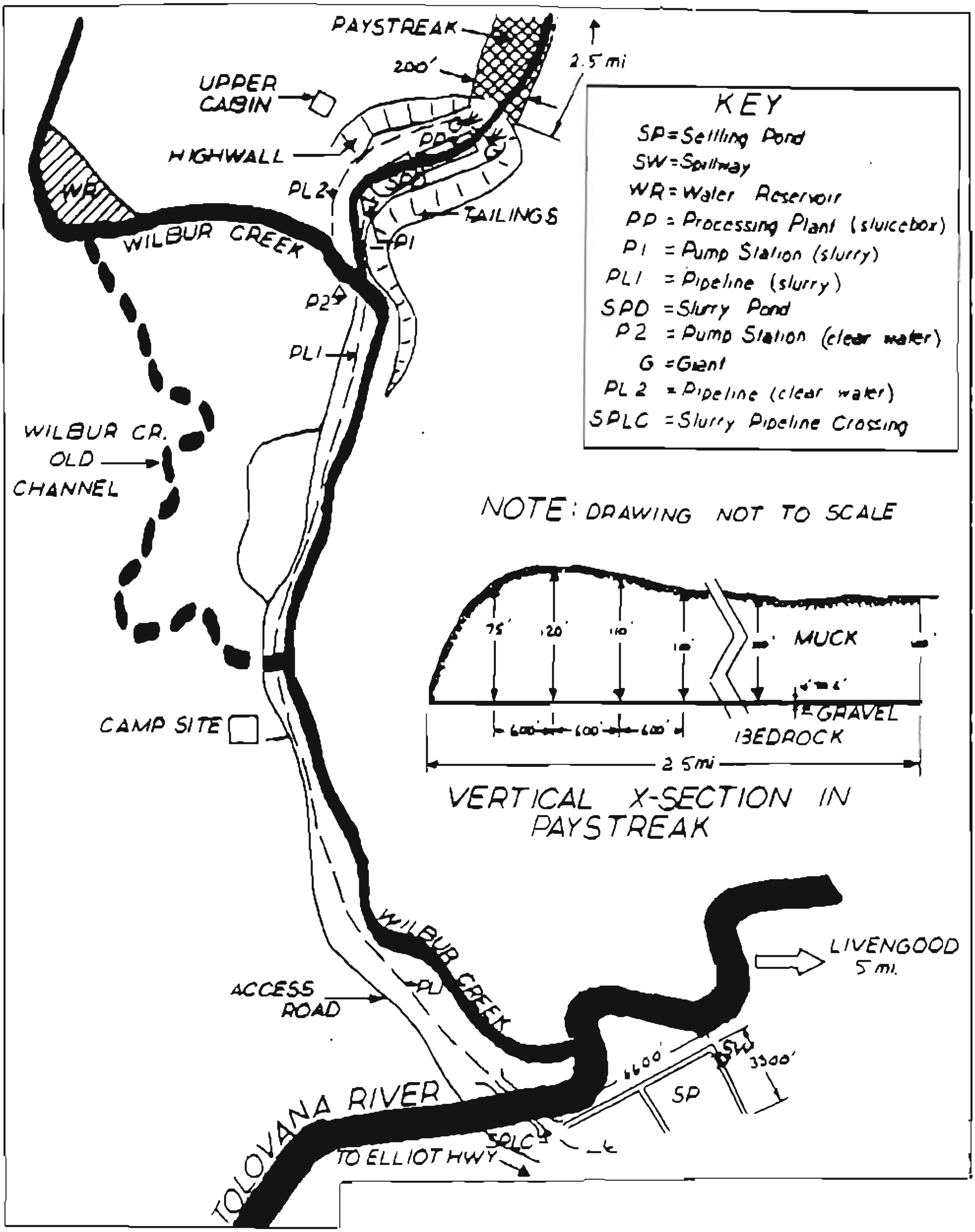

Figure 1. General outline of the Wilbur Creek Mine deposit and conventional technology (hydraulicking).

80-ft-wide high-grade zone is estimated to contain about 80 percent of the gold (based on data of previous years).

The current owner, Stanley C. Rybachek, has mined this property since 1961 using hydraulicking. Increasing thickness of the overburden, stringent enforcement of the 1972 Clean Water Act, and limited space available in the narrow cut produced by hydraulicking made the compliance with settleable solids (SS) and turbidity standards impossible. Attempts to construct settling ponds in a diverted channel of Wilbur Creek failed each time Wilbur Creek reached a flood stage.

Obviously, new technology was needed. Serious consideration was given to underground mining, which would allow selective mining of a high-grade part of the deposit, reduce water use, and limit surface disturbance. At the same time, selective mining of the gold-bearing section of the deposit would minimize the volume of silt (from overburden muck) to be processed through the sluice box. This in turn would have a positive effect on fine-gold recovery.

State grants from the Alaska DNR and DEC for developing new methods and technologies for placer mining enabled the Wilbur Creek project, comprising underground mining and 100 percent recirculation of water, to be started. The project includes all tire-mounted equipment for drilling, loading of ore, and hauling, and the use of explosives to break frozen ground. All procedures are in compliance with MSHA safety regulations. Because of a delicate heat balance of frozen ground and its pronounced influence on stability of underground openings, mining can be conducted only when the outside air temperature is low.

Financial compromises had to be made for selecting some of the mine components, particularly the drilling equipment. Generally, used equipment had to be acquired. Frequent breakdowns of the used equipment made evaluation more difficult.

The data presented here covers the period from the startup of the operation last fall (1986) through December 24, 1986. Until October 20, work concentrated on preparation of the stockpile site, construction of service and living quarters, upgrading of the surface haulage road, and excavating the portal site. A 14-ft-diam by 30-ft-long culvert portal was installed on October 23. Because of the brief period when data on underground mining were collected, the data have to be viewed as incomplete. (In fact, complete data on stability of the underground openings will become available only after several years of operation, when seasonal and yearly variations in climatic conditions can be included.)

Earlier studies (Skudrzyk, 1968a,b) state that there is not enough space close to the mine for the existing settling ponds to handle settleable solids from conventional hydraulicking. The effluent would have to be pumped to flat terrain across the Tolovana River. (Cost comparisons between hydraulicking and underground mining begin on p. 114.)

UNDERGROUND PLACER MINING IN PERMAFROST, GENERAL CONSIDERATIONS

Stripping (hydraulicking) vs Underground Mining

The major considerations in determining whether a deposit should be mined using surface or underground methods are depth and size of a placer deposit, temperature of the ground, strength of frozen ground, and the subsurface hydrology. In addition, environmental requirements such as water-quality control and surface disturbance (reclamation) may dictate a specific type of surface mining (for example, ripping or blasting of frozen muck, hauling it away and revegetating it or bringing it back after mining of gravels for reclamation) or even an underground method.

Dredging, though economically feasible, is not considered here.

Depth and Size of Placer Deposit

Commonly, an auriferous alluvial deposit has a shape of a paystreak with one dimension much larger than the remaining two (fig. 2). With a stable slope angle for frozen muck covered with a layer of vegetation of about 30°, the volume stripping ratios (volume of overburden removed per unit volume of mining section) are significantly higher than those based on a ratio of thickness of overburden to thickness of mining section. (For a 10-ft-thick mining section with a 100-ft-thick overburden, the volume stripping ratio is 20.5 for a 200-ft-wide paystreak and 31 for a 100-ft-wide paystreak.) Because the overburden per unit volume usually contains much more silt and clay than the mining section, switching to underground mining significantly reduces the volume of silt and clay handled, typically by as much as 100 times.

In terms of ground disturbance, underground mining produces none except for surface subsidence, which would come into play only when undermining a highway or a river. Neither case is likely in Alaska.

Temperature of the Ground

Continuity of the permafrost in the mining section and a few feet below it is a 'must' requirement for an underground operation. Figure 3 (Hartman and Johnson, 1984) indicates the general continuity of permafrost in Alaska. Even if a placer deposit is in a continuous permafrost zone, the ground temperature should be checked during exploratory drilling. Though refreezing of discontinuous permafrost is technically feasible, it would obviously increase the cost of mining.

Several cases have been reported of an underground operation encountering a catastropic inflow of water because of warm ground, or working during warm season, or both. Working underground at temperatures only slightly below freezing is also dangerous because of the reduced strength of the frozen ground (fig. 4, Skudrzyk and others, 1987).

Therefore, you must obtain information on temperature distribution in the placer deposit (for example, down-hole temperature measurements during exploration). A string of thermometers (accurate to ±0.1°F) properly housed

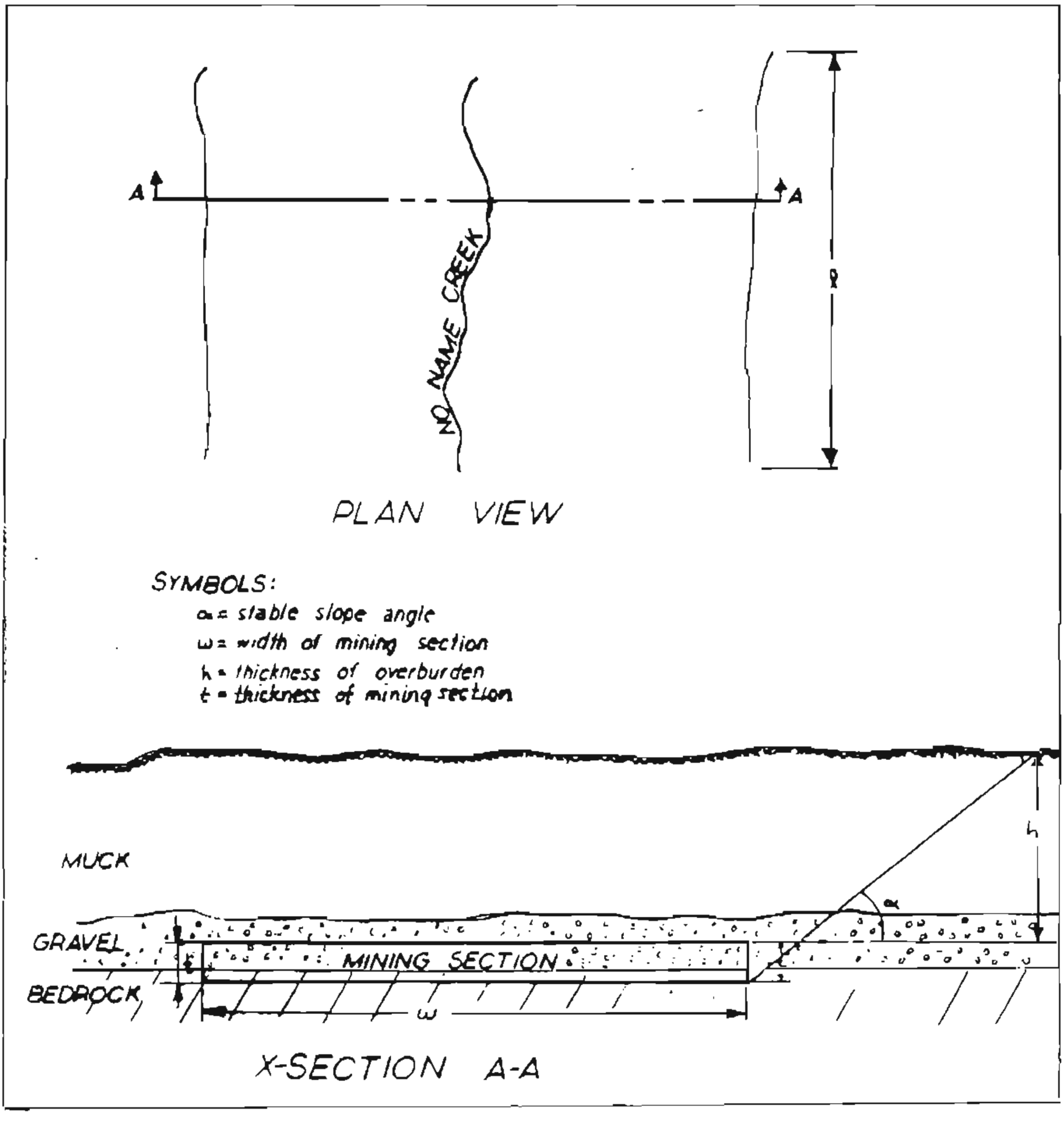

Figure 2. Simplified geometry of an alluvial deposit.

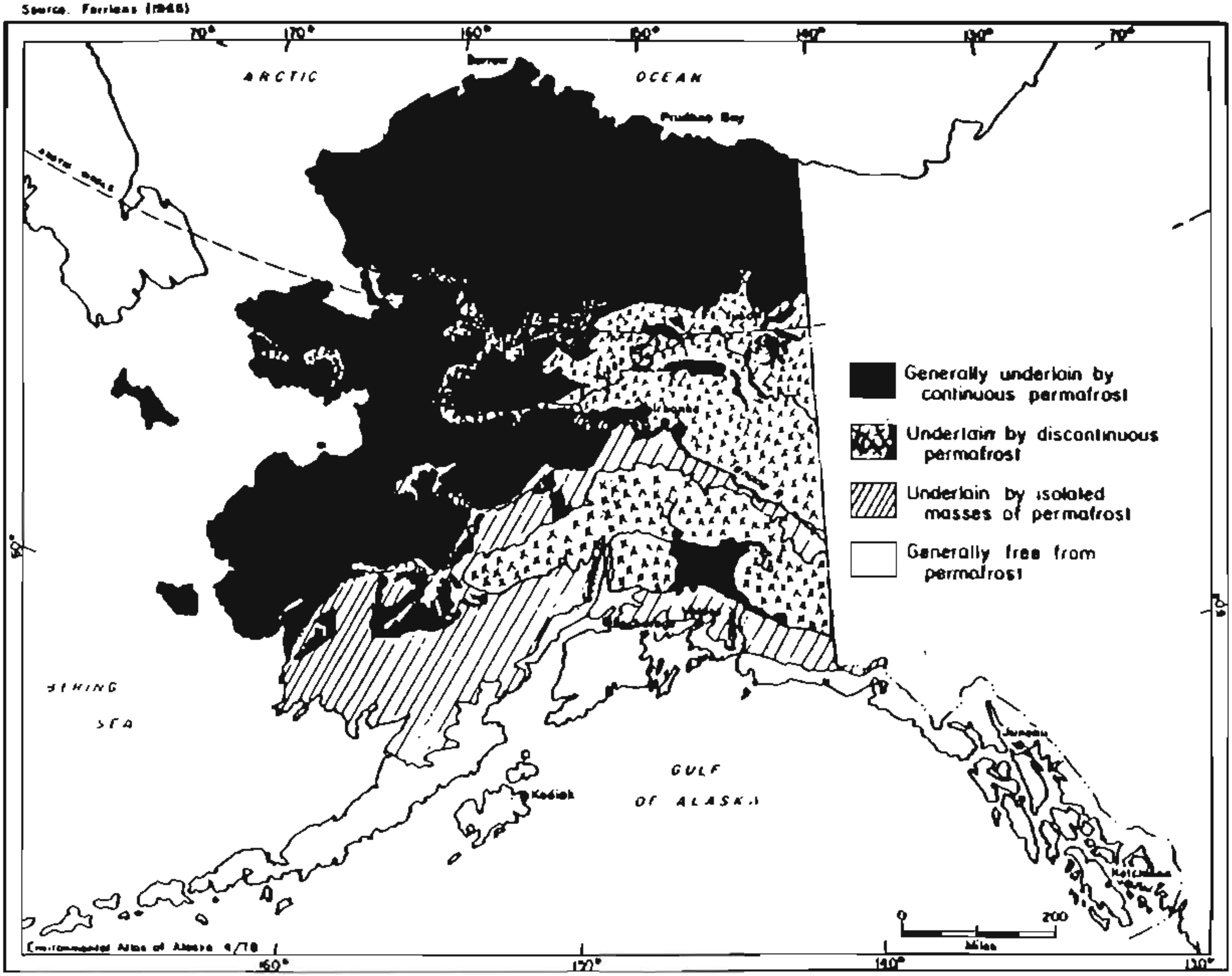

Figure 3. Permafrost in Alaska (from Hartman and Johnson, 1984).

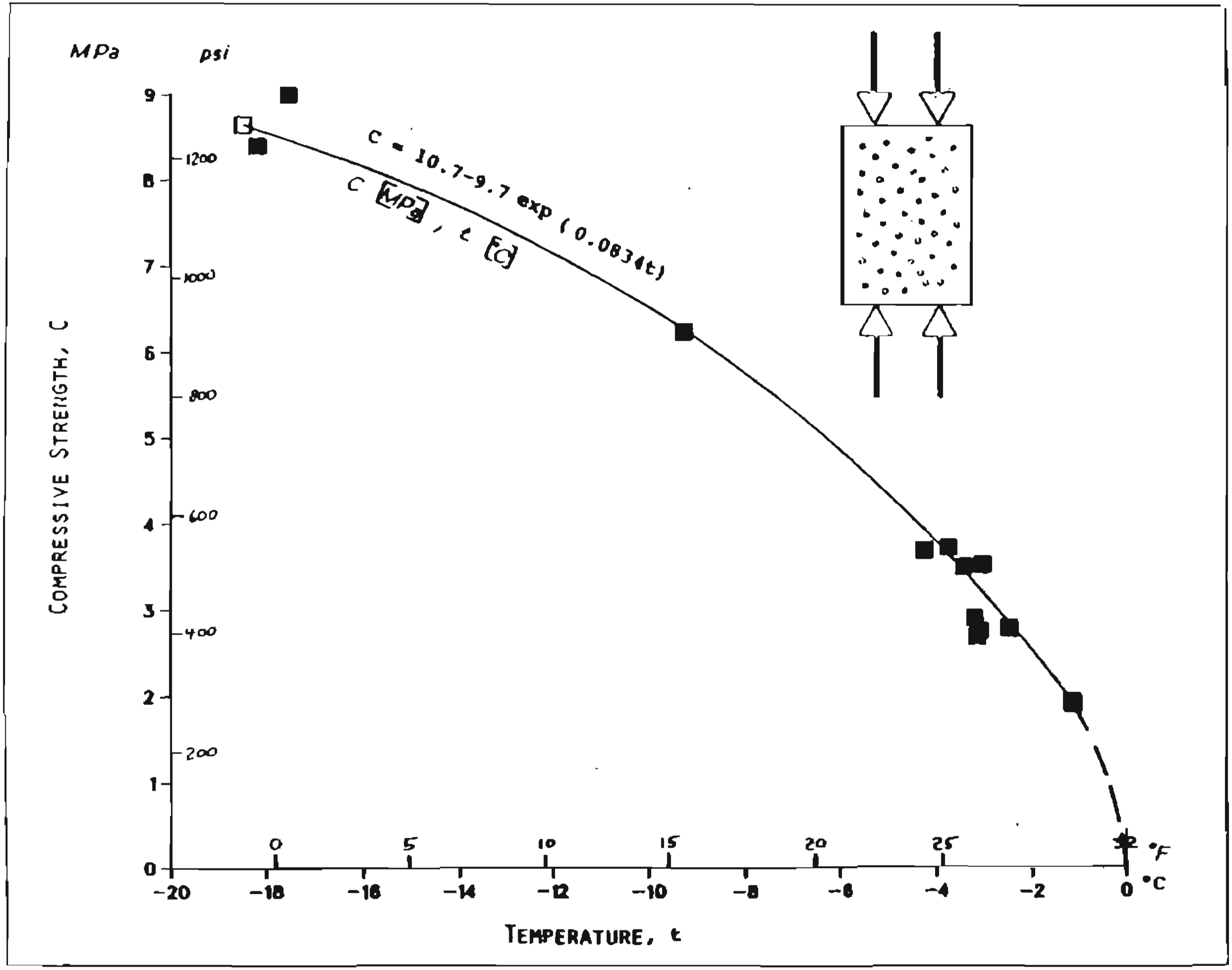

Figure 4. Compressive strength of frozen ground vs. temperature (from Skudrzyk and others, 1984).

and insulated in a temperature probe could be lowered into each of the exploration holes (fig. 5) to obtain the ground-temperature profile. Measured changes of temperature with depth would allow one to estimate the proximity to thawed zones. Presence of thickly weathered and well-decomposed (high-silt content) bedrock would usually improve the situation, because weathered bedrock is a poor heat conductor.

Measuring the temperature of each drill hole during exploration should be a standard procedure when drilling any deep placer deposit.

Strength of Frozen Ground

The strength of frozen ground depends largely on the temperature and ice content. Ice as a cementing material is much weaker than the gravel grains themselves and determines the strength of the frozen gravel. Data in figure 6 (Skudrzyk and others, 1987) are for standard testing conditions (6-in.-diam, 12-in.-high samples tested at 27°F with a 10$\mu\varepsilon$/sec [1$\mu\varepsilon$=10^{-6} in./in.] strain rate). With decreasing ice content (below 20 percent by dry weight) the strength of frozen gravel decreases. At close to zero-percent ice content, the strength is obviously very low. Such a material---be it gravel, sand, silt or a combination thereof---is called dryfrost, which, when encountered in underground openings, may cause severe stability problems.

Frozen-ground strength also depends on the rate of loading. Figure 7 (Skudrzyk and others, 1987) shows the strength of frozen gravel at 27°F when each sample was loaded at different strain rates (12-in.-long samples were contracted by 0.25 in. in 7 sec to 2.5 hr). Frozen gravel loaded at faster rates is stronger (fig. 7). This loading-rate-to-strength relationship is particularly important in determining the long-term strength and when selecting explosives. Explosives with lower detonation velocities (ANFO and similar types) have proved to be more effective in breaking frozen ground than dynamites and other high-velocity explosives.

Subsurface Hydrology

Shallow permafrost (with its lower boundary slightly below bedrock) and obviously discontinuous permafrost will be difficult, if not impossible, to mine underground if pressure of the subsurface water in the thawed zone is high. Driving openings close to thawed ground with high pore-water pressure may cause failure and inflow of water into the mine. Taking temperature measurements in holes drilled both ahead of and in the floor of existing advancing underground openings could warn of 'warm' permafrost zones.

PLACER MINING EXPERIENCE IN SIBERIA

More and more information is becoming available on mining of gold, tin, and tungsten placers in Siberia. Last year, during the Eighth Annual Placer Mining Conference, a paper titled 'Underground Placer Mining Practices in Siberia' was presented (Skudrzyk and Barker, in press); it outlined some of the placer geology and mine-development practices in Siberia. A tragic history of gold mining in Siberia was recently provided by Cieslíwicz (1987).

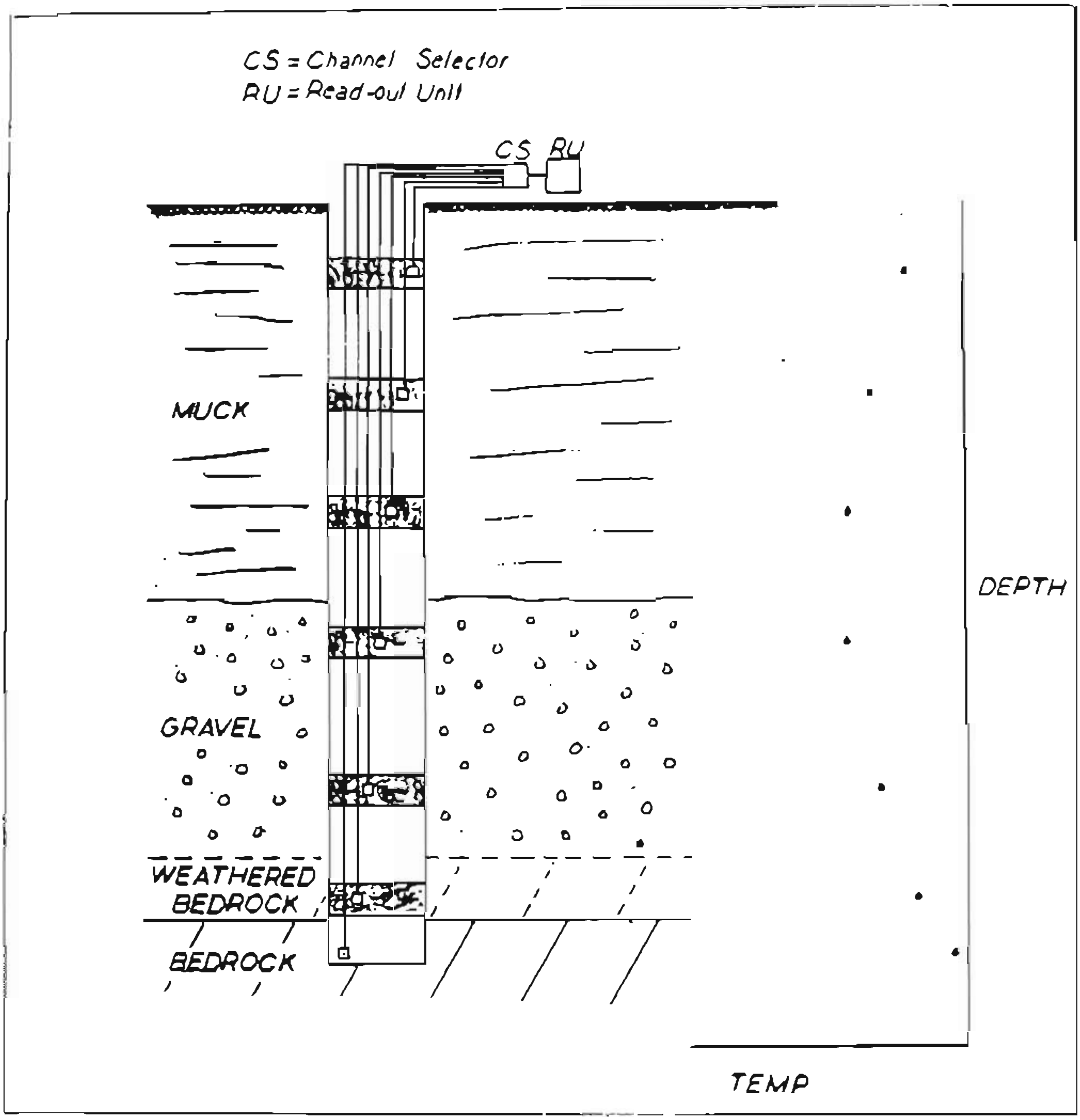

Figure 5. Temperature measurement in an exploration borehole.

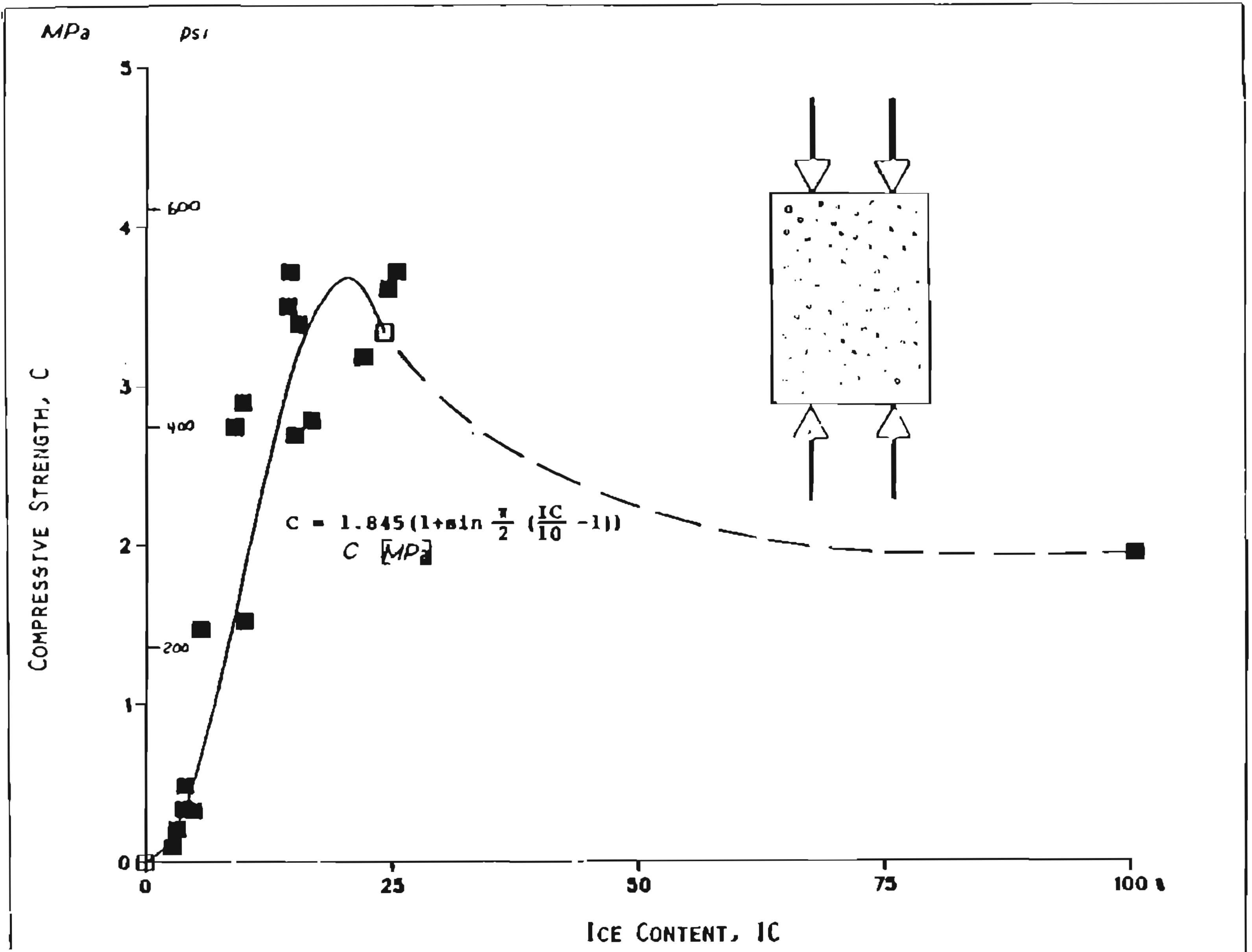

Figure 6. Strength of frozen gravel as a function of sample ice content (Skudrzyk and others, 1987).

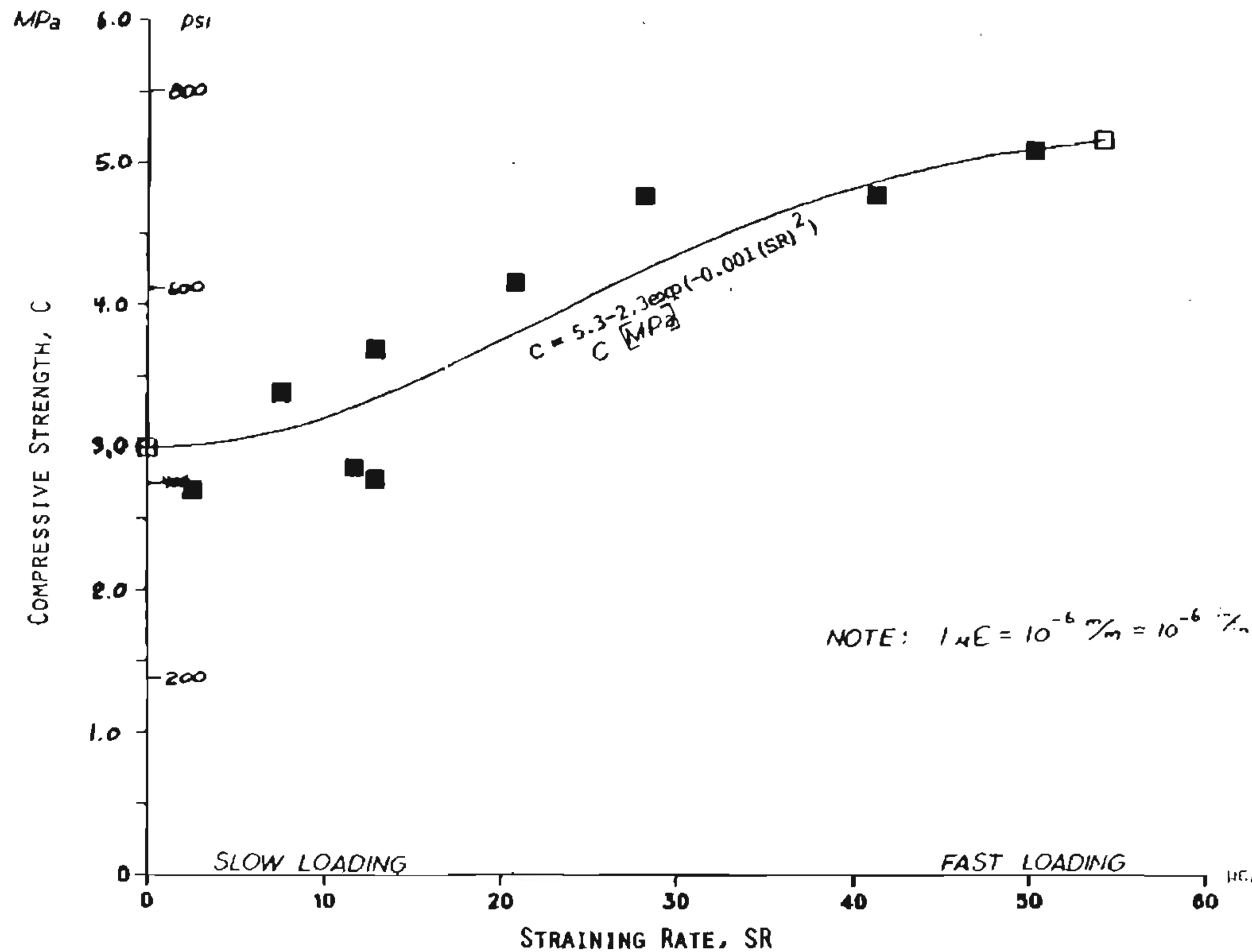

Figure 7. Strength of frozen gravel as a function of straining rate (from Skudrzyk and others, 1987).

In this volume, Robert Ziegler discusses Siberian underground drilling and blasting practices in frozen ground (p. 235).

Up to 10 million troy oz/yr of gold may be coming from placer deposits in the USSR, most of them presently mined using underground methods. Government-funded research has been conducted for several years, and some publications are available (Emelanov and others, 1982). With up to 1,000 yd^3/day of auriferous gravel production from some of the largest mines, much effort has been devoted to improving ventilation and stability of the underground mine. Underground layouts of extraction openings were discussed last year (Skudrzyk and Barker, in press). Selection of mining method, size of extraction openings, and support is done based on a classification system show in table 1. Stability evaluation and roof-control measures are also based on this classification system and depth of the deposits (tables 2 and 3 and fig. 8).

CANADIAN EXPERIENCE IN UNDERGROUND MINING OF FROZEN PLACERS

Data from Main Street Mining, Ltd., Yukon Territory is summarized below:

- Paystreak has been previously outlined by drilling.
- Paystreak dimensions: 300-ft-across, 1,500-ft-long, 10-ft mining section.
- Horizontal drift access to operation.
- Crew size: 2-LHD Operators
2-Drillers
1-Mechanic
1-Cook
- Drift dimensions: 18- x 12- x 10-ft high.
- Production rate: Two 18- x 12- x 10-ft rounds/shift.
- Haulage equipment: Two EIMCO-912D scooptrams (LHD's) 2.25-yd^3 buckets; 'material up to watermelon size is easily handled.'
- Drilling equipment: Two Air track drills.
- Blasting: 2-in.-diam holes;
13 holes/round; AMEX ANFO explosive.
- In 4-mo season, they drifted 1,700 ft.

The Main Street mining operation may have been the first modern underground placer mine in the western hemisphere.

UNDERGROUND MINING ON WILBUR CREEK

Brief Geology of the Wilbur Creek Placer and its Basic Properties

The Wilbur Creek placer is a bench deposit along Wilbur Creek, one of the Tolovana River tributaries, in the Livengood area. Bedrock locally exposed by recent surface and underground operations consists of thin, vertical beds of graywacke striking northeast.

Table 1. Classification of roof strata for underground mining of frozen placers (Emelanov and others, 1982).

Class of stability of frozen ground	Composition of immediate and main roof, up to 15 m thickness	Permafrost temperature (°C)	Total ice content (%)	Texture (structure)
I. Highly stable	1. Alluvial deposits consisting of gravels, cobbles, and rare boulders with matrix of sand, silt, and clay. Complete saturation of pores by ice. Matrix-material content 25 to 50%. Stratified with single homogeneous stratum at least 10 m thick.	below -6	below 25	massive
	2. Alluvial and lake-alluvial deposits similar in composition and thickness to the above.	-6 to -3	below 25	massive
	3. Sandy deposits with cobbles and gravels up to 30%. Ice-saturated pores.	below -3	below 25	massive
II. Stable	1. Homogenous silty and clayey deposits.	below -4	25 to 50	massive, stratified
	2. Alluvial, lake-alluvial, glacial, and shallow-marine sediments of interbedded layers of large-grain and clay-size materials with poorly shown stratification with thickness of homogeneous layers 5 to 10 m.	below -3	below 25 (for large-grain materials), 25-50 for clay-size materials	massive, stratified
III. Medium stable	1. Alluvial and lake-alluvial deposits, composition as in I.1.	-3 to -2	below 25	massive
	2. Sandy deposits with cobbles and gravels up to 30%.	-3 to -2	25 to 50	stratified
	3. Homogeneous silty and clayey sediments.	-4 to -3	25 to 50	stratified, netlike

Table 1. (con.)

Class of stability of frozen ground	Composition of immediate and main roof, up to 15 m thickness	Permafrost temperature (°C)	Total ice content (%)	Texture (structure)
	4. Interbedded layers of large-grain to clay-size materials with horizontal stratification. Layers up to 2 m thick.	-2 to -1	below 25 for large-grain materials), 25 to 50 for small-grain materials	massive, porous layered, netlike
	5. Ground ice and clear ice.	below -6	above 60	ataxic
IV. Poorly stable	1. Alluvial and lake-alluvial deposits similar in composition to I.1.	-2 to -1	below 25	massive
	2. Sandy deposits with gravels up to 30%.	-2 to -1	25 - 50	stratified
	3. Homogeneous silty and clayey sediments.	-3 to -1.5	25 - 50	stratified, netlike
	4. Interbedded layers of large-grain to clay-size materials with distinct horizontial stratification; layers up to 2 m thick.	-2 to -1	25 - 50	netlike, massive, porous, stratified
	5. Eluvial-solifluction silty formations, loess-like clays of lake-swamp and marine-lagoon genesis.	-3 to 1.5	25 - 50	netlike
	6. High ice content, saline (above 0.25%); clay formations of lake-swamp, offshore marine, and lagoon genesis.	-6 to -3	above 50	ataxic
	7. Ground ice and clear ice.	-6 to -3	above 60	ataxic

Table 1. (con.)

Class of stability of frozen ground	Composition of immediate and main roof, up to 15 m thickness	Permafrost temperature (°C)	Total ice content (%)	Texture (structure)
V. Unstable	1. Plastic, frozen alluvial materials of any grain-size distribution with silty and clayey matrix.	above -1.5	below 50	any
	2. As above, with sandy matrix.	above -1	below 50	any
	3. Unconsolidated, poorly cemented by ice materials.	any	below 3	massive, spongy
	4. Ground ice and clear ice.	above -3	above 60	ataxic

Table 2. Stability of extraction openings.

Depth of opening (M)	Stress (MPa)	Probability of loss of stability for a given class of frozen-ground strength (%)				
		I (2.5)	II (2)	III (1.6)	IV (1.3)	V (1)
20	0.57	0.01	0.03	0.02	0.3	1.6
30	0.66	0.03	0.2	1.1	4.5	24.5
50	1.43	1.6	7.6	29.8	100	100
100	2.66	20	70	100	100	100
150	4.3	100	100	100	100	100

Unstable for probabilities >20 percent.
In parentheses: long-term strength of frozen ground (MPa)

Table 3. Thickness of overburden and span of extraction openings.

Stability class	Overburden thickness (m)		Stable span of extraction openings (m)			
	Monolitic roof	Stratified roof	Single opening		Multiple openings	
			Monolitic	Stratified	Monolitic	Stratified
I	14-20	13-18	35-45	30-40	26-37	24-33
II	12-16	10-12	25-35	23-27	22-30	21-24
III	10-13	7-9	20-25	15-20	19-24	13-17
IV	7-11	4-6	10-15	8-12	8-12	7-11
V	4-8	2-4	6-10	5-8	5-7	4-6

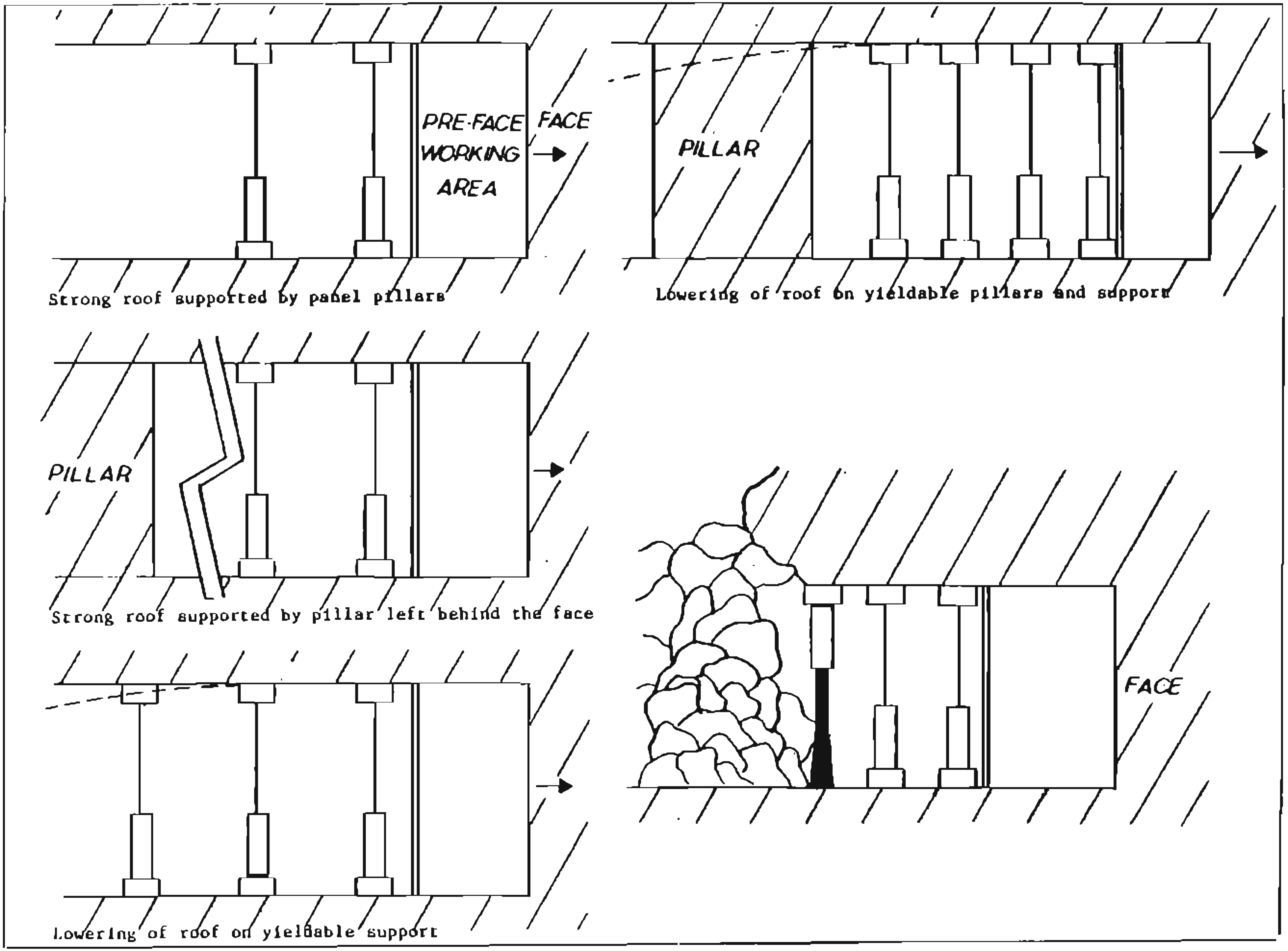

Figure 8. Schemes of roof control in frozen placers.

The placer gravels are 2 to 8 ft thick and well sorted, with no cobbles and larger particles and less than 11-percent silt and clay content. Overburden, which is 65 to 135 ft thick and consists of wind-blown silt and organic material, contains abundant ground ice as vertical and horizontal sheets, wedges, and irregular masses. The ice constitutes about 20 percent of the volume of the overburden and ranges from clear, with many trapped air bubbles, to laminated with silt. Its specific gravity is as low as 0.94. The remaining 80 percent is frozen silt (specific gravity, 1.52), which contains 40-percent interstitial ice by weight. Overburden stratification is evident at greater depth with inclusions of organic material and volcanic ash. When hydraulicking, a 30° stable slope is typical for overburden material covered with new vegetation. Ice content with respect to weight of frozen material is 8 percent for bedrock and gravels and 47 percent for overburden muck. Silt and clay content of gravel and bedrock is 10.59 percent. The average bulk (bank) unit weight is 130 lb/ft^3 for frozen intact bedrock and gravel and 87.2/ft^3 for frozen muck. Unit weight of broken frozen gravels and bedrock is 81.25 lb/ft^3 (swell factor of 1.6), 100 lb/ft^3 for thawed gravels, and 74.1 lb/ft^3 for thawed loose silt and clay.

Definition of the New Underground Placer Technology

Environmental restrictions concerning treatment of effluent from the placer operation increased the cost of the mining operation and dictated the change to underground mining. The underground operation (fig. 9) consisted of a modified room and pillar system, drilling (with a single-boom rubber-tired jumbo), and blasting (with ANFO); the rubber-tired mucking season was about equal to the average seasonal production possible in a surface mine. Thus, mining a block 80 ft by 500 ft by 9 ft each season (winter only) would result in a mine life of 26 yr.

The 14-ft-diam and 30-ft-long culvert portal was placed horizontally into the northeastern corner of the deposit. The initially planned location for the portal was within the paystreak, but because of a recent landslide, it had to be moved to the present location.

A development opening 24 ft wide and 9 ft high (4 ft in gravels and 5 ft in bedrock) was then driven toward the high-grade channel of the paystreak. The channel, delineated by mining in previous years, is about 80 ft wide.

The deposit is developed with a main 24- by 9-ft drift from which crosscuts are driven at 48-ft intervals. Pillars 24 ft by 48 ft are left along the drift (fig. 9). The pillar design formula for frozen gravels suggested recently by Skudrzyk (1987) was used in selection of pillar dimensions:

$$C_p = (1.63-1.1 \exp (0.107v))*(3.3-3 \exp (0.834t))*$$

$$*0.57(1+\sin \Pi/2)*(IC/10-1)C_o/(SF*CRF)$$

where

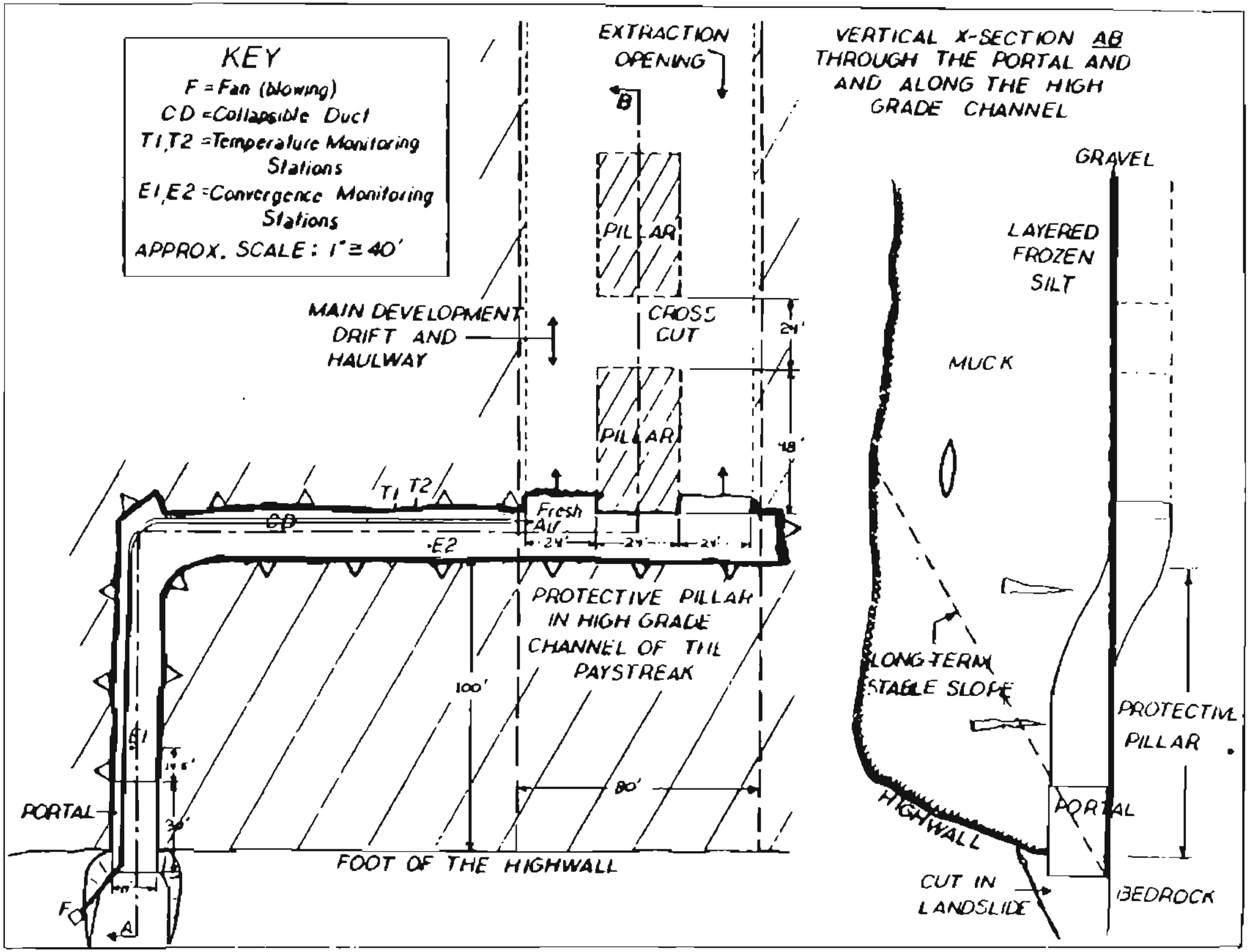

Figure 9. General outline of the underground Wilbur Creek Mine.

C_p = frozen gravel pillar strength (MPa)
V = volume of the pillar (m^3)
t = temperature of the pillar ($^{\circ}C$)
IC = ice content of the pillar (%)
C_o = compressive strength of a standard sample (MPa)
SF = safety factor
CRF = closure rate factor related to acceptable contraction of the pillar.

The extraction opening, now planned to be 30 ft wide, may be expanded to up to 60 ft by moving the development drift to the east if a wider high-grade channel is encountered. The extraction ratio for the 80-ft-wide channel, now 80 percent, will be increased by thinning out the pillars to 16 ft wide and possibly further in a few years, if data on ground temperature and convergence warrant it. Also, rooms to the east of the main drift can be excavated if pockets of high-grade ore are found there.

All ground fragmentation is done by drilling and blasting. A single-boom jumbo ATH12 SECOMA with a hydraulic drifter RPH 200 (rotary and rotary percussive drilling) is used to drill 8½-ft-long, 1-3/4 in.-diam blast holes. The hydraulic jumbo has an articulated chassis with four-wheel drive driven by a four-cylinder-Deutz diesel rated 52 hp at 2,300 rpm. The RPH 200 drifter is designed to drill small-diameter holes (production holes 1¼ to 1-3/4 in. and cut holes up to 3½ in.) with water flushing. A hydraulically operated drifter generates no mist, and noise level is low (105 dB at 3 ft). Drilling rates in frozen ground are very high (5 to 10 ft/min under good conditions).

The blasting-round design is a subject of ongoing experiments. Initial trials with a V cut did not produce satisfactory results, mostly because of dead pressing of the ANFO (ammonium nitrate and fuel oil). Later, a modified burn cut was used with acceptable results. Now, based on extensive Russian studies and practices in eastern Siberia, experiments are being conducted with a cylindrical cut and its modification. ANFO explosives, primed with a stick of TOVEX 220 (water-gel explosive) and electric caps are used. Both millisec (MS) and long-period (LP) ACUDET caps have been used. ANFO is loaded to blast holes with a compressed-air ANFO loader. No plastic sleeves or other means are used to protect ANFO from absorption of water. Generally, this blasting technique produced satisfactory results. The powder factor has been reduced from an initial 4 lb/yd^3 to less than 3 lb/yd^3 (bank volume), with further reduction possible. Further experiments are planned to reduce overbreak in the roof (by using lighter loads of ANFO, T-1 water gel, or a heavy detonation cord), which would also lower the powder factor. Fragmentation is good, and there has been no significant time loss associated with floor cleanup.

Mucking and primary haulage are done with a 2-yd^3 Wagner ST-2D scooptram, which loads the ore, now outside the mine, into a 13-yd^3 highway truck. The truck delivers gravels to the stockpile area (fig. 10). As the underground operation advances, the scooptram-truck reloading station will be moved underground to keep the primary haulage distance within economic limits.

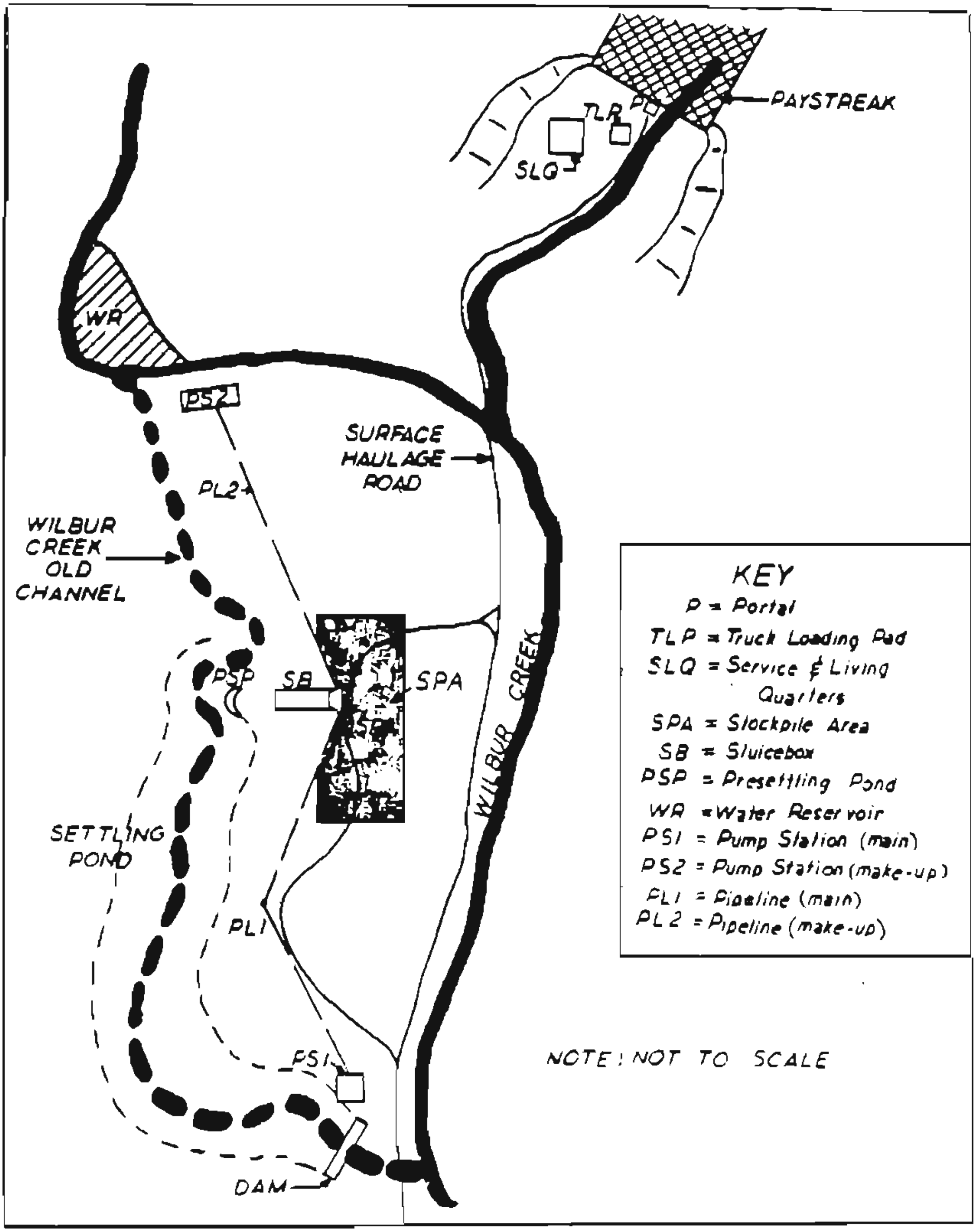

Figure 10. Surface facilities of the underground Wilbur Creek Mine.

Production Cycle

The present production cycle consists of drilling, loading explosives, blasting, ventilating the fumes, mucking, and haulage for a typical round of 24 ft by 9 ft by 8 ft. Detailed data on the unit operations are being collected and will be reported in the final report. A typical cycle time is 8 hr, (2 hr, 25 min of drilling time; 1 hr, 30 min of cleaning holes and explosives loading; 30 min of ventilation; and 3 hr, 20 min of mucking and haulage).

With increasing experience, improvements in technology, and increasing haulage distance, the unit-operation times will change.

Ground Control and Air-quality Monitoring

Ground Control

With the lack of data on stability of mining openings in warm permafrost in Alaska, ground control is a major factor in assuring safe and flawless operation of the underground placer mine. The stability of openings will be determined by the steep high-wall exposed after a recent landslide; much heat can be exchanged, such as heat dissipated by relatively high-powered diesel equipment (scooptram 77 hp, jumbo drill 52 hp, and, in the future, a 13-yd^3 truck), seasonal and daily changes in temperature of air ventilated during the mining season through the mine, strength of the pillars' gravel and bedrock and roof's muck, and the layout of openings. Because of the lack of data on the resultant underground temperature and strength of gravels, a conservative approach was assumed. Pillar dimensions and span of openings have been overdesigned to assure safety, though still offering a workable solution. Temperature and convergence monitoring equipment has been installed, and data are being collected. As the mine workings expand, additional monitoring stations will be installed at the convergence of tunnel openings. Temperature data will be analyzed after next summer, but it will only reflect this year's winter and summer climatic conditions, which may be far from the average. The thermal balance of the underground permafrost may also be influenced by subsurface aquifers and possible flow of water into the mine from the thawing landslide.

As mentioned, vertical and horizontal convergence of opening is monitored at strategic locations in the mine, but although this provides important information on safety, it cannot eliminate some of the factors of potential danger to the stability of the mine.

Underground Environmental Considerations

The mine is connected to a forcing fan and flexible duct system for venting.

Ventilation is necessary to dilute the toxic and noxious gases from blasting and diesel exhaust. The following calculations point out the minimum quantity of air and power of a fan required for the current production plan. The air quality is based on diesel-exhaust dilution. This can be calculated from the manufacturer's specifications. For a mine with a single ST-2D scooptram, the following factors apply:

Deutz diesel (4 cycle), model F6L-912W

Maximum power at 2,300 rpm - 81 hp

MSHA schedule 24-power rating at 2,300 rpm - 77 hp

Maximum torque at 1,500 rpm - 191 ft-lb

Bore and stroke - 3.94 by 4.7 in.

Displacement - 345 $in.^3$

Cylinders - 6

Cooling - air

On basis of the State of Alaska Mines Safety and Conservation Regulation (1975)

$$\text{Air quantity} = \frac{100\ ft^3}{min\ BHP} \times 77\ BHP$$

$$+\ 2\ men\ \frac{(100\ ft^3/min)}{man}$$

Required quantity = 8,000 cfm

The law also stipulates that concentration of the toxic gases should not exceed the following TWA-TLV limits:

Carbon monoxide	- CO	>100 ppm
Carbon dioxide	- CO_2	>5000 ppm
Oxides of nitrogen	- NO_x	>25 ppm
Aldehydes	-	>10 ppm

Because the load factor of the scooptram is only ((3.3/12) x 100) 27.5 percent during an average shift, the calculated TWA-TLV exposure is below the stipulated limits.

Quantity of Air Based on Dilution Requirement

In an average round of blasting with both headings advancing, it is assumed that, at the most, 45 holes are detonated. Each hole contains 10 lb ANFO (94-percent ammonium nitrate and 6-percent fuel oil). Under hydroscopic conditions (wet holes), as is the case here, the volume of CO liberated is 0.59 ft^3/lb of ANFO, and the volume of NO_2 liberated is about 0.08 ft^3/lb of explosive. On the basis of these data, the volume of CO liberated (the limiting case) is:

$$45 \times 0.59 \times 10 = 266 \text{ ft}^3 \text{ of CO}.$$

Initial volume of the opening considered = 300 by 24 by 9 ft = 64,800 ft^3

Concentration of CO = $\frac{266}{64,800}$ = 0.41 percent by volume

TLV for CO = 100 ppm - 0.01 percent by volume.

Assuming a standup time of 30 min, the quantity of air required to dilute the concentration of CO to safe limits is:

$$-\text{Log}_{10} \frac{0.01}{0.41} = \frac{Q}{2.303} \times \frac{30}{64,800}$$

Q=8,020 CFM

On the basis of these calculations, it appears that if the air quantity provided satisfies the diesel stipulations, it will be enough to dilute the blasting fumes within a reasonable standup time.

Head-loss Calculation

The flexible duct is 24 in. in diameter and, based on manufacturer's data, the expected head loss for a quantity of 8,000 cfm is about 1.5 in. of water gauge per 100 ft of tubing. The total length of the duct, including coupling and bend losses, is about 387 ft. Therefore, head loss in the intake side is:

$$387 \times \frac{1.5}{100} = 5.80 \text{ of water gauge}$$

Head loss calculated on the return side is negligible. Therefore, the minimum fan horsepower for the present configuration is:

$$\text{Hp} = \frac{(5.80)\ (8000)}{6250} = 7.3 \text{ hp}$$

The horsepower of the current fan is around 10 hp and therefore appears to be adequate for now. However, a larger fan will be needed as the mine advances or as the requirement for air quantity increases because of the increased load factor of the scooptram or introduction of other haulage equipment (such as highway trucks) into the system.

Gold Recovery and Effluent Handling

In addition to the obvious reduction of water use in groundbreaking, surface area, amount of ground disturbed, and volume of silt and clay handled, underground mining of a frozen auriferous placer has several advantages in terms of gold recovery. Assuming that gold content of the mined gravels is monitored on a regular basis, the deposit can be mined selectively, taking the valuable ground only and leaving in place the low-grade spots. Layout of pillars and spacing between them can be adjusted to maintain a high extraction ratio (above 80 percent, or even approaching 100 percent) with construction of artificial pillars made of moist gravel tailings brought back underground (Skudrzyk and others, 1987) or by leaving pillars in the low-grade ground. In case of hydraulicking, the entire width of the mining section has to be mined.

Under favorable conditions (thick mining section, no blasting down of silt layers), only auriferous gravels and weathered bedrock are extracted. This assures that the bedrock is well 'cleaned' by extracting all of the local depressions. All the mined material is removed in a frozen state, thus assuring that there is no gold left in the natural riffles of the bedrock. Finally, there is no mixing of silt from the overburden with gravels and bedrock to be processed. The latter itself is a significant factor because dewatering of the thawed gravels was a problem. Presence of a significant amount of silt in processed gravels made sluicing more difficult and caused early packing of the sluice box.

The gravels produced during a particular underground mining season will be stored in a stockpile (fig. 10) to be thawed by solar heat and processed the following summer. If needed, the thawing process will be enhanced by collecting water from the thaw in shallow ditches. Water, after warming up by being exposed to sunshine, will be sprayed on the pile. Subsequently, the ore will be processed in a sluice box with 30 ft of riffles in a 3-ft-wide main box and two 1-ft-wide side boxes. The effluent will pass through a presettling pond and then will be gravity-fed into a main settling pond. The settling pond (fig. 10) has at least a 600-ft length, a 12-ft (maximum) depth, and a 35-ft-long dam. The capacity of the pond is estimated at 6,400 yd^3, whereas the silt/clay volume to be handled per mining season at the maximum mining output of 10,700 yd^3 (a block of 500 by 80 by 9 ft at 80 percent extraction) of gravels is given by the following formula:

$$V_S = \frac{1}{2700Y_s} (V_{g,b} * Y_{g,b} * P_{s,g} + V_{s,b} * Y_{s,b} * P_{s,m}),\ yd^3$$

where:

V_s = Volume of silt and clay disposed (yd^3)

$V_{g,b}$ = Bank volume of gravel/bedrock (ft^3)

$Y_{g,b}$ = Bank unit, weight of gravel/bedrock (lb/ft^3)

$P_{s,g}$ = Silt content by weight of gravel/bedrock (lb/ft^3)

Y_s = Unit weight of loose silt/clay

$V_{s,b}$ = Volume of silt mined with gravels (ft^3)

$Y_{s,b}$ = Bank unit weight of frozen muck (lb/ft^3)

$P_{s,m}$ = Silt content by weight in muck

On the basis of earlier assumed data, the volume of silt disposed per mining season will be (assuming an 8-ft section of gravel and bedrock and a 1-ft layer of muck) 6,200 yd^3, which is within the pond capacity. At the expected 1,500-gpm water flow rate through the sluice box, the settling pond is rated at 4,000-gpm/acre overflow which, with the substantial length of the pond, would produce sufficient settling of fine particles for a low-silt-content water to be recirculated to the sluice box.

PRELIMINARY ECONOMICS OF THE NEW UNDERGROUND TECHNOLOGY

The preliminary costs of mining are calculated on the basis of production data for October-December 1986. Equipment breakdowns, delays in shipment of spare parts, and holiday seasons limited work to 23 12-hr shifts. During that time, a two-man crew advanced the development drift by 198 ft in 24 rounds, producing 1,584 yd^3 (bank volume) of gravels.

Cost of Underground Mining

The cost of components are shown in table 4. All but labor costs are expressed (in the final column) in $\$/yd^3$. Labor cost is listed both in $\$/yd^3$ (at \$20/hr per man) and in hr/yd^3 if one wants to perform calculations for a different hourly scale.

Straight depreciation of capital equipment and facilities with zero salvage value was assumed. The time when a particular piece of equipment was used on the project was usually calculated with breakdown-time factor (BDF) and a unit-operation time when the equipment was used. The breakdown time factor is:

$$BDF = \frac{\text{total time worked-mining time}}{\text{total time}} =$$

$$= \frac{23 \text{ shifts } 12 \text{ hr/shift} - 24 \text{ rounds } 6.75 \text{ hr/round}}{2312} = 0.41$$

or 41 percent.

$$\text{Time used on the project (TUP)} = \frac{1}{1-BDF} \text{ unit operation cycle time (UOCT)}$$

Facilities costs (shop and living quarters and portal) include construction. The depreciation time was calculated assuming 150 days per year (length of mining season). TUP=1.7*(UOCT)

PROCESSING COST

Processing cost for the underground mining was calculated from estimated costs of processing of a yearly average output:

- volume of gravel processed: 10,700 yd^3 (bank volume)
  17,120 yd^3 (1.6 swell factor)
- processing time at 60 yd^3/hr: 290 hr (60 yd^3/hr based on experience from previous years)
- use of water: 7.4 yd^3 water per yd^3 of gravel at 1,500 gpm.

D-8 dozer total working time was assumed to be 25 percent of the sluicing time.

Table 4. Anticipated cost for underground mining on Wilbur Creek.

A. PROCESSING COST*

1. Capital Equipment and Facilities Depreciation (owning) Cost**

Item	Cost, $	Depreciation time	Time used on project (TUP) HR (DAY)	Cost ($/yd^3)
Sluice box	15,000	mine life (26 yr)	1 yr	0.054
E-Z Panner	3,750	mine life	1 yr	0.014
8/6" Diesel	25,000	20,000 hr	1,560 hr	0.182
D-8 Cat***	40,000	4,000 hr	390 hr	0.365
Settling-pond construction	4,000	mine life time	1 yr	0.015
Miscellaneous	20,000	mine life	1 yr	0.072
			Owning cost	0.71

2. Equipment Operating Cost

a. Equipment maintenance cost for used underground equipment estimated at 25% of owning cost).........................$0.18/yd^3

b. Fuel cost (at $1/gal)

8/6" diesel pump at 4 gal/hr (290 hr)***
D-8 Cat at 20 gal/hr (75 hr)
Total fuel cost...$0.16/yd^3

c. Settling pond cleanup (at $1.25/yd^3 of silt handled).......$0.58/yd^3

3. Labor Cost

Two-man crew for 320 hr (at $20/hr)...........0.037 hr/yd^3 or $0.75/yd^3

TOTAL PROCESSING COST: $2.38/yd^3

*Assuming processing of anticipated average yearly output of 17,120 yd^3 loose gravels 60 yd^3/hr and 1,500 gpm of water.
**Straight depreciation, zero salvage value.
***Used only in handling tailings.

Table 4. Cost components for underground mining on Wilbur Creek* (con.)

Total mine life 26 yr, 150 days operation per season.

B. MINING COST

1. Capital Equipment and Facilities Depreciation (Owning) Cost**

Item	Cost, $	Depreciation time	Time used on project (TUP)	Cost ($/$yd^3$)
Shop & living quarters	10,000	10 yr	23 days	0.097
Scooptram	36,000	15,000 hr	135.2 hr	0.205
Jumbo drill	28,000	15,000 hr	57.5 hr	0.068
Generator	13,880	9,000 hr	23 days (14-hr/day)	0.313
Fan & ducts	2,500	15,000 hr	23 (12-hr/day)	0.029
Dump truck	15,395	15,000 hr	135.2	0.088
Portal	25,000	mine life	23 days	0.093
Misc. Equipment	25,000	mine life	23 days	0.093
				0.99

2. Equipment Operation Cost

a. Equipment maintenance cost for used underground equipment estimated at 50 percent of owning cost (no maintenance for facilities assumed)..0.398 \$/yd^3

b. Fuel cost (at \$1.0/gal)

Scooptram	4 gal/hr X 135.2 hr
Jumbo drill	4 gal/hr X 57.5
Generator	2.2 gal/hr X 23 X 14 hr
Dump truck	6 gal/hr X 67.6 hr.................\$1.19/yd^3

3. Explosives cost (ANFO loader included in miscellaneous equipment)

For one round of 24 by 9 by 8.25 ft, 19 holes; 10 lb of ANFO, 1 stick of TOVEX 220, 1 cap per hole:

ANFO	190	lb.	@ \$ 0.33/lb
Tovex	17.6	lb.	@ \$ 1.48/lb
Caps	19		@ \$ 1.75/cap............................\$1.85/yd^3

4. Labor Cost:
Two-man crew, 23 shifts, 12 hr/shift, 0.35 hr/yd^3 or (at \$20/hr)...\$6.97/yd^3

TOTAL MINING COST: \$11.31/yd^3

*Based on unpublished report (Skudrzyk and others, 1987).
**Straight depreciation, zero salvage value.

As in mining-cost calculations, straight depreciation with zero equipment salvage value was assumed.

From these assumptions and calculations, the total cost of underground mining and processing of 1 yd^3 of bank gravel is $13.69. Several cost components may be expected to decrease with time, including: experience gained by crew, better organization of work, smaller breakdown factor, better use of explosives. The haulage cost was calculated for this paper based on the startup of the exploration phase of the project. Insurance is not included.

COMPARISON OF UNDERGROUND METHOD WITH STRIPPING

Definition of the Stripping Method

For this evaluation it is assumed that the stripping method for the Wilbur Creek property would consist of the following:

- removal of the overburden by hydraulicking
- stock piling of the gold-bearing gravels
- recovery of gold by sluicing
- piling of gravel tailings
- removal of settleable solids from effluent in settling ponds and meeting turbidity standards, if at all possible, by using PEO or another method.

Because of the limited space available in the narrow cut produced by hydraulicking, it is further assumed that the effluent from the overburden muck and ore processing would be pumped across the Tolovana River, where a settling pond of sufficient capacity and PEO treatment plant would be located. After treatment, the effluent (clear water, less than 5 NTU) would be disposed into the Tolovana River, and water needed for mining operations would be drawn from the water reservoir on Wilbur creek (fig. 1).

Economics of the Conventional Technology

A limited study concerning the stripping technology was recently performed by Skudrzyk (1986a) and data from it are used in this paper. The data were calculated based on a total mine life of over 60 yr and average costs at present prices of equipment and materials.

Item	Cost ($/$yd^3$)
Hydraulicking (cost of labor and clear-water pumping to hydromonitors up to 2.5 mi from present water reservoir location)	7.69
Gravel processing and tailings disposal	1.50
Settling pond construction, maintenance and pumping of effluent across Tolovana River	15.02
Treatment of effluent with PEO	N/A
Reclamation	Not considered

TOTAL COST OF CONVENTIONAL TECHNOLOGY: $24.21/$yd^3$

Low gravel processing and tailings disposal cost (item B, above) was assumed rather than calculated. The datum is based on approximate costs of handling of gravels at the Wilbur Creek Mine based on equipment operating costs (labor cost not included):

$0.5/yd^3 - stockpiling before sluicing
$0.5/yd^3 - sluice-box feeding
$0.5/yd^3 - tailings stockpiling.

Again, exploration, access, royalties (if any), taxes, and insurance costs are not included.

Economic Comparison of the Two Technologies

As could have been expected, mining unit cost/per unit bank volume of gravel for underground mining is higher ($11.31 yd^3 compared to only $7.69 yd^3 for hydraulicking, table 5). Despite the fact that the volume of the overburden removed is large (stripping ratio of 24.7), hydraulicking is one of the least expensive methods of removing muck overburden. However, if the two methods are compared based on a cost of unit of gold produced, the cost for underground mining (mining of 80 percent of gold from the high-grade channel) is slightly lower (by about 10 percent) than for stripping (mining of the entire 200-ft-wide channel containing 100 percent of gold).

When costs of processing and water treatment to meet the waste water treatment requirements are added---settleable solids for hydraulicking only!---the cost analysis favors underground mining. The stripping is 1.77 times more expensive based on per yd^3 of gravel core cost and 3.54 times more based on unit of gold produced. If costs of water treatment to meet the required 5-NTU limit were added, this would favor underground mining even more.

CONCLUSIONS AND RECOMMENDATIONS

The data reported here pertain to the initial period of time of the underground operation.

Although equipment lowers the up-front capital; it increases maintenance cost and breakdown time. If supplies of spare parts are not local, this can cause major delays and significant losses of production.

When purchasing used equipment, there is a tendency to acquire what is available rather than what fits best for a particular project. Specifically on this project, a drill should be selected that is capable of drilling dry (compressed-air flushing with, if needed, a dust collector) and longer holes (at least 10 ft long for over 200 ft face area). Further research on optimization of drilling is needed. Also, standard diesel-operated equipment (highway truck) when brought underground has to meet exhaust criteria; this, too, should be considered when purchasing.

The project has been, so far, very successful; not only did it provide field data on the underground mining of frozen placer with technology and methods

Table 5. Cost comparison, underground vs. stripping.

1.	UNDERGROUND*		
	- Mining cost**		$11.31/yd³
	- Anticipated processing cost		$ 2.38/yd³
		TOTAL:	$13.69/yd³
2.	STRIPPING* (based on Skudrzyk, 1986a)		
	a. Hydraulicking (cost of labor and clear-water pumping to hydrostatic monitors up to 2.5 mi from present water reservoir location)		$ 7.69/yd³
	b. Gravel processing and tailings disposal		$ 1.5/yd³
	c. Settling-pond construction, maintenance, and pumping of effluent across Tolovana River		$15.02/yd³
	d. Cost of meeting turbidity standards		N/A
	e. Reclamation		not considered
		TOTAL:	$24.21/yd³

*Access, exploration and insurance costs not included (Skudrzyk and others, 1987).
**Haulage cost for initial operation only (does not account for increase with distance).

developed by modern hard-rock and coal mining, but it also seems to be economically feasible, even when costs of meeting water-quality requirements are added.

Because of the tremendous underground placer-mining potential in Alaska, the state government should continue its efforts leading to a development of economic, safe, and minimum-environmental-impact technologies. Drilling and blasting, ventilation, stability of underground openings, optimum layouts of openings, artificial pillars, surface subsidence, and refrigeration of deep placer in discontinuous and warm permafrost, should be studied.

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THE PACIFIC LEGAL FOUNDATION IN ALASKA: AN UPDATE

by

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Getting the accurate information across to the media, the decision makers, and especially the courts is crucial if you are to survive and continue to mine this year, next year, and every year thereafter. Here are the latest developments in the many legal battles that Alaskan miners are engaged in.

The issue of information, and the accuracy of it, struck me recently when I was reading an issue of an environmentalist magazine, American Land Forum, and came across a poem on Alaska. It was penned by an English professor in New York City who became an expert on the pipeline when she visited Alaska. It is not my custom to read poetry, but I will quote one brief passage about the ecological effects of the pipeline on caribou migration:

> "What of antlered multitudes
> flowing across primeval land--
> abruptly piled against unyielding steel
> bisecting their domain,
> miles of brown panic
> milling, trampling
> their centuries of ebb and flow
> brought to a halt."

This is garbage, but it is all too typical of what the media and decision makers are being buried with whenever important policy issues are discussed. While most of you may have little directly invested in the pipeline or ANWR, this sort of grotesque misinformation is rampant in discussion and arguments over placer mining. And it is precisely because of this disinformation that the Pacific Legal Foundation has set up an office in Alaska.

THE 'MINING IN THE PARKS' LAWSUIT--NORTHERN ALASKA ENVIRONMENTAL CENTER VS. HODEL

Let me give you some examples from back in early 1985, when the Sierra Club was working to get an injunction against all mining within the National Park system. Following are a few passages from the Sierra Club's May 16, 1985 brief:

> "[M]ining operations in the national parks in Alaska are polluting hundreds of miles of streams and causing tens of millions of dollars of damage."

After discussing the 'treasures' of the 'crown jewels,' the Sierra Club next began to quote National Park Service reports:

> "The applicable state water quality standards and waste disposal permit stipulations have been grossly exceeded as a result of placer mining wastewater discharges."

> "The average placer miner in the park violated the standard for turbidity by a factor of 700 and caused 'substantial increases in heavy metal concentrations."

> "The mining operations tested by the state were found to be discharging anywhere from 333 to 36,630 pounds of sediment per hour into the park's streams."

After this tirade on water quality, the Sierra Club continued on the effect of mining on the land:

> "Placer mining rips up the terrain being mined...In Denali 26 miles of stream have been physically altered by mining...and it costs between $350,000 and $500,000 to rehabilitate a mile of stream that has been mined... thus...mining operations have caused between $9 million and $13 million in damage to park lands."

And this goes on and on throughout the brief. Adverse effects on moose, caribou, bear, and wolf are discussed. And then in the conclusion, the Sierra Club really goes off the deep end and waxes poetic:

> "These parks are national treasures, 'our generation's legacy for the future'...Yet the Park Service's unlawful actions are causing the contamination of hundreds of miles of streams in the parks and tens of millions of dollars in damage to park lands."

And of course, the brief concludes, "Taking into account the interests of the affected miners does not alter the balance [in favor of granting an injunction to stop mining]."

In short, what the court was faced with was a series of gross misrepresentations and exaggerations about the effects of mining in the parks.

Now I will quote the sum total of what the United States government had to say to the court about the realistic effects of placer mining on water quality:

"____________."

And this is what the United States had to say about the differences between past historical mining and mining as regulated in 1985:

"____________________."

And now let me quote the sum total of what the United States had to say to the court about the effects of placer mining on animal habitat:

"____________________."

And now let me quote the sum total of what the United States had to say to the court about the effect of an injunction on you the miners, and the tremendous hardship that would result:

"____________________."

Now what is a court supposed to do? It grants an injunction! In fact, with this sort of propaganda involved, it is not at all surprising that the Sierra Club was able to hoodwink the court into believing that the state's National Parks were a national disaster area, some sort of environmental apocalypse or, simply put, a Hoboken, New Jersey, waiting to happen.

The court issued an injunction. It was convinced. The court wrote:

> "There are currently approximately 40 mining operations underway in the national parks in Alaska. A number of these are causing extensive environmental damage to the parks."

And then, after the damage was done, Pacific Legal Foundation was called in. We immediately moved to intervene in the lawsuit and asked the court, before it did anything else, to remove the injunction or at least let the miners finish off the 1985 season. We convinced the court to allow the mining to continue until mid-October but could do no more. We then appealed the case to the Ninth Circuit Court, but, as you know, had little success there. The Sierra Club repeated all the charges it made before, but elaborated claiming that "essentially uncontrolled" mining was "tearing apart" the parks, causing "tremendous environmental degradation." The damage to the miners had been done, the record was established by the district court that mining caused all sorts of environmental harm, and we had no opportunity to get more accurate information before the court. When a case is appealed you are stuck with a record established in the lower courts.

THE SIERRA CLUB VS PENFOLD (BUREAU OF LAND MANAGEMENT)

When the mining-on-BLM-lands lawsuit struck, we were determined not to allow the Sierra Club to be the only ones to establish the record. The case started out in the exact same way. Monstrous allegations of environmental harm were made.

I will quote the titles of some of the chapters in the Sierra Club's brief where they asked for an injunction:

> "Placer Mines Are Wrecking National Wild and Scenic Rivers"

"Placer Mines Are Wrecking National Conservation and Recreation Areas"

"Placer Mines Are Wrecking Other Public Lands"

"The Nearly Universal Failure to Reclaim"

"Placer Mines Are Wrecking Important Subsistence Resources."

And, a few especially memorable quotes from their work of fiction are as follows:

"Mining operations...dump hundreds or possibly thousands of tons of sediment into the [Birch Creek] river system each day."

"The impacts to fish in Birch Creek are catastrophic."

The residents of Birch Creek Village suffer and "there have unexplained cases of sickness that the people attribute to drinking the river water...Trapping harvests along the river are down by 50 percent [the fish is so bad it is fed to the dogs]."

In the Steese National Conservation Area there is "massive physical destruction...[and] besides ripping up streambeds and adjacent areas, mining operations ...etch access routes across the tundra...[leaving] scars visible for years."

"According to the Environmental Protection Agency, [actually a letter from Bub Louiselle] 'the majority of the miners (greater than 90 percent) do not reclaim their operations on an annual basis, nor do they reclaim their operations after mining is completed at a given site."

"Each year placer mines on the public lands excavate approximately <u>one hundred million tons</u> of overburden and tailings. It appears that very little, if any, of the one hundred million tons is put back where it came from."

At Minto, "the silt and muck from the mines destroys trapping...makes the water unfit to drink...and drives away the game."

"BLM is to blame for the placer mining fiasco."

"BLM's actions are causing the contamination of wild and scenic rivers, the destruction of hundreds of

> streambeds throughout Alaska, the elimination of fish and wildlife populations, and the restriction of important subsistence uses. Mines are permitted to operate in gross violations of water quality standards and reclamation requirements. These facts constitute indisputable irreparable injury."

Indisputable irreparable injury? And that leads to the question---what did the United States government do to dispute this and put mining in perspective? What did the government say about the historical mining practices compared with those common today? What studies did the government cite to refute the claims that subsistence uses were being damaged? What did the government say about the potential hardship to the miners if an injunction were granted? What did the United States say about the real progress being made to clean up the rivers, to reclaim mined lands, and to revolutionize the industry?

Absolutely nothing.

I do not blame the government. When it somes to arguments between industry and environmentalists, the government hates to take sides. The government---in the BLM case at any rate---has been very aggressive to throw every technical procedure possible against the Sierra Club, and do everything possible to ensure that the BLM will not be thrown into chaos by an injunction. But the federal government will not take sides, and has not argued about the facts surrounding the mining industry and the tremendous hardship that an injunction would cause to hundreds and hundreds of miners and Alaska's rural economy.

Fortunately, this time Pacific Legal Foundation was prepared. As soon as we were asked to get involved in this case, we immediately began to work to prepare the record. Through the tireless work of many people in Fairbanks and Anchorage we gathered together over one hundred declarations, and five volumes of reports, studies, charts, and other exhibits on the true state of Alaska's environment and the effects of placer mining.

We were able to submit evidence that the effects of placer mining today are nothing compared to what they were in years past.

We were able to submit evidence that subsistence use, fishing, hunting, and trapping are not being substantially adversely affected today by placer mining.

And most important, we were able to show that the hardships to many hundreds of miners would be extraordinary and totally unnecessary in light of the efforts made by miners to run clean operations. In short, instead of letting the Sierra Club paint the only picture of grotesque distortion about placer mining, we presented a fair and accurate picture to the court. We were able to show the court that it was the Sierra Club, not the miners, who were bent on creating the most destruction in Alaska.

So far, we have been successful. The court has refused to grant an injunction to stop mining operation of any size on Alaska's lands. Unsettled issues remain, such as whether mining should be stopped until cumulative impact studies are completed (the same general issue as found in the Park Service case), but we hope the court will continue to act reasonably now that it finally has been educated.

The Sierra Club has not taken this lightly, however. It appealed the decision not to enjoin small operations, those under five acres, to the Ninth Circuit. For those operations over five acres the district court in Anchorage is still contemplating whether to reconsider as asked by the Sierra Club in December. Right now, the Ninth Circuit is deciding whether to enjoin all operations pending the full appeal later this summer.

The Sierra Club has also achieved new heights of rhetoric, which appear to be desperate falsehoods, claiming that the average mine:

> "dumps nearly five tons of sediment into the river system each hour...turning otherwise pristine streambeds into sterile moonscapes...thus, the land that is being torn up by mines that BLM refuses to review...may remain eroding rubble heaps for the rest of our lifetimes...[the Birch Creek is a] flowing river of muck...and the outstandingly remarkable national wild river is being rapidly transformed into a hundred twenty-six mile sewer for mining wastes... [And if mining does not stop] life in Birch Creek village as it has been conducted for centuries will stop."

I am not without concern as to what the Ninth Circuit will do with such an onslaught of misleading propaganda. However, since Monday, March 23, 1987, I feel a bit more optimistic than I did last week. Until then, the Ninth Circuit had a rule that an injunction should automatically be issued in environmental cases whenever any charge of environmental harm was made. On March 23, 1987 the United States Supreme Court reversed this rule when it allowed leasing to occur in the Navarrin basin. Pacific Legal Foundation was involved in that case as well, and we argued that injunctions should not issued unless all factors were considered including the economic hardships to users of the public lands such as miners and oil exploration people. Until March 23, 1987, the Ninth Circuit would not even listen to this sort of information. Now it must.

Many of you may have heard in the news that the Supreme Court ruled in that case on subsistence rights in the outer continental shelf. It did, but I believe a more important issue decided was that automatic injunctions shall not be granted just because a groundless charge of environmental harm is made. For that reason I am quite a bit more optimistic about the possible outcome of the case than I was last week. Needless to say, on March 26, 1987 we filed with the Ninth Circuit a short memorandum to remind them of the Supreme Court's decision and asking them to decide quickly to deny an injunction. The court has also been told how important it is to have a decision before spring breakup.

SUMMARY OF WHERE THE VARIOUS LAWSUITS NOW STAND

Northern Alaska Environmental Center vs. Hodel

An injunction will remain in effect against the miners in Denali, Yukon-Charley, and Wrangell-St. Elias National Parks and Preserves until an environmental impact statement is prepared. The impact statement will be completed no sooner than 1988, and perhaps a few years later after the inevitable legal challenges are resolved. Individual miners in these parks may submit plans of operations to the Park Service for review. If it is decided they have no significant effect on the park, the miner or the Park Service may go to court to have the plan approved. All other miners in other parks must go to the Park Service first for the approval of a plan of operations. If the Park Service is unreasonable about looking at a plan of operations, then gather together everything the Park Service says in writing, see an attorney, and have your attorney get in touch with me.

The Sierra Club recently asked for another injunction to forbid the Park Service from approving all plans of operation until full-scale validity exams are performed. We are vigorously opposing this, and a decision is expected no sooner than a couple of months from now. Incidentally, let me note that the Park Service admitted that it has a policy to halt validity exams whenever it looks like a claim might be valid. To quote the government: The government will "stop a mineral examination at the point beyond which additional investigation would simply provide further proof of validity."

Sierra Club vs. Penfold

Operations Over Five Acres: The court in Anchorage is considering whether to change its decision not to enjoin operations over five acres. The Sierra Club asked the court to do this in December.

Operations Under Five Acres: The court refused to grant an injunction and the matter is on appeal.

Cumulative Impacts: The court has not yet ruled on whether cumulative environmental impact statements must be prepared as was decided in the Park Service. However, because of the prior decision against the Sierra Club it may be difficult for the court to issue such an order.

Granite Rock vs. California Coastal Commission

In this case, the California Coastal Commission ordered a small limestone quarry to shut down until the company obtained permits from the coastal commission. Pacific Legal Foundation filed an amicus curiae brief on behalf of the Alaska Miners arguing that the Coastal Commission failed to apply proper procedures and that federal law prevented local governments from trying to shut down mining operations on federal lands. On Monday, March 23, 1987, the Supreme Court ruled against the mining company in a 5-4 decision. Apparently, the court was swayed by states-rights arguments and ruled that there is a difference between regulation of land telling a person what can be done with land compared to regulation telling how land can be developed.

Trustees for Alaska vs. State of Alaska (Section 6i lawsuit)

In this case the Trustees for Alaska sued the state government, claiming that the present location-lease system is unconstitutional and that all mining on state lands must be enjoined until a new system of royalties is in place. Pacific Legal Foundation is representing Fairbanks North Star Borough, which has intervened on the side of the miners. The case was soundly won in Superior Court, but the Trustees appealed to the State Supreme Court. We are currently waiting a decision from that court.

Village of Tuluksak vs. State of Alaska

This is the case where Trustees for Alaska, on behalf of several local groups, are attempting to shut down a mining operation near Bethel. They said, among other things, that the coastal act was violated. Pacific Legal Foundation is representing the Alaska Miners Association, which has intervened to argue that the coastal zone has been improperly interpreted.

CONCLUSION

Pacific Legal Foundation has been in Alaska full time since mid-November 1986. Our goal is to set the record straight and provide the courts with whatever accurate information it takes to turn the tide for miners, oil drillers, foresters, and anyone else who is trying to make a productive contribution to the state's economy. We had been trying to get an office established for some time but, as you know, everything takes time and money. The generous contributions from miners throughout the state were enough to put us over the top. We also had the generous support of a lower-48 foundation, the M.J. Murdock Foundation, to help set up our office. Your contributions are making the fight for miners' rights the best fought legal battles possible.

My responsibility is to get the right information to the courts about placer mining and other development activities. It is only through accurate information that you have a chance. You, too, have a responsibility. You must do all you can to get accurate information about the placer-mining industry out to the public. And, most importantly, if you like what the Foundation is doing and want to see more of the same, then I urge you to redouble your efforts and contributions. We want the Pacific Legal Foundation to be a completely Alaska-supported office. We depend on your contributions, which are completely tax deductible and used entirely for our Alaska project office. I want to see the Foundation remain in Alaska for many years to fight where and whenever necessary on behalf of Alaskans who are working to make this a great state.

Thank you very much.

POSTSCRIPT

Since the time this presentation was made at the 1987 Placer Miners Conference, the Federal District Court in Anchorage ordered BLM to prepare massive cumulative impact studies on all mining in four drainages: the Birch Creek watershed, the Beaver Creek watershed, the Fortymile River watershed, and all streams flowing into Minto Flats. All mines that must obtain a plan of operations from BLM must shut down after the 1987 season until the studies are completed. BLM estimates the studies will be complete in mid-1988. The rulings are being challenged by the government and Pacific Legal Foundation.

THE DEMONSTRATION OF PLACER-MINE WASTEWATER TREATMENT WITH FLOCCULANTS AT ESPERANZA RESOURCES CO., INC. ON FAITH CREEK

by

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During the summer of 1986, Esperanza Resources Company, Inc. and their subcontractor, Northern Testing Laboratories, Inc. (NTL) completed a project under the Alaska Department of Environmental Conservation (ADEC) Placer Mine Innovative Technology Grant Program. A full project report (Grant 1008) has been submitted to the ADEC. The project principal investigators were Michael R. Pollen, President of NTL and Norman L. Phillips, Jr., a contract geological engineer with Esperanza. The project demonstrated the use of multiple settling ponds with nearly complete recycle of the pond water for sluicing, and flocculant treatment of the final wastewater from Esperanza's 150-yd^3/hr placer mine at the Faith Creek drainage near Fairbanks (fig. 1).

An intensive water-quality monitoring effort was initiated at the start of the project to track the watershed chemistry, document the performance of Esperanza's settling-pond system, and monitor the influence of the mine effluent on the creeks before, during, and after the flocculant wastewater treatment system was tested. Parameters included residue measurements for settleable solids (SS), total suspended solids (TSS), and turbidity (fig. 2), and for total, total recoverable, and dissolved arsenic. In addition, field tests for alkalinity, conductivity, dissolved oxygen, hardness, pH, and temperature were run. Flow data was taken from several stations periodically throughout the mining season by ADEC. The project team ran over 2,000 tests with a field laboratory at the mine site and at the NTL water-quality lab in Fairbanks.

SETTLING-POND STUDY

The mining operation was sited on Homestake Creek during the first half of the season (fig. 3). This multiple settling pond system consistently achieved compliance with the EPA placer-mine water-quality discharge permit standard of <0.2 ml/L settleable solids, but was unable to meet the ADEC water-quality criteria of turbidity (5 NTU above background). Homestake Creek turbidity above the operation averaged less than 1.0 NTU from July 10 through August 3 while the Homestake site was in production; downstream, turbidity levels ranged from 80 to 3,000 NTU in daily monitoring (app. A,

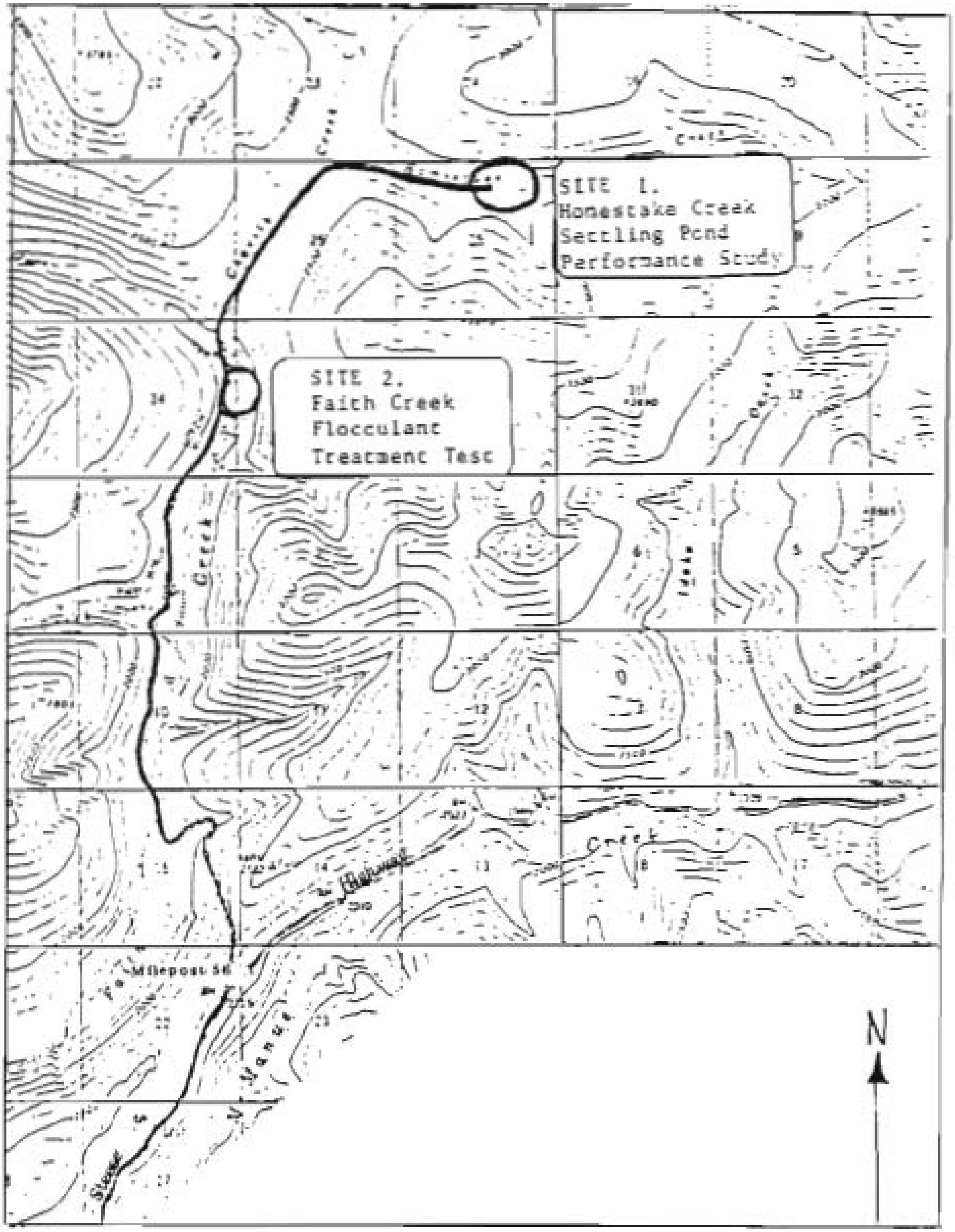


Figure 1. Project locations, U.S. Geological Survey Circle (B-5) Quadrangle, scale 1:63,360.

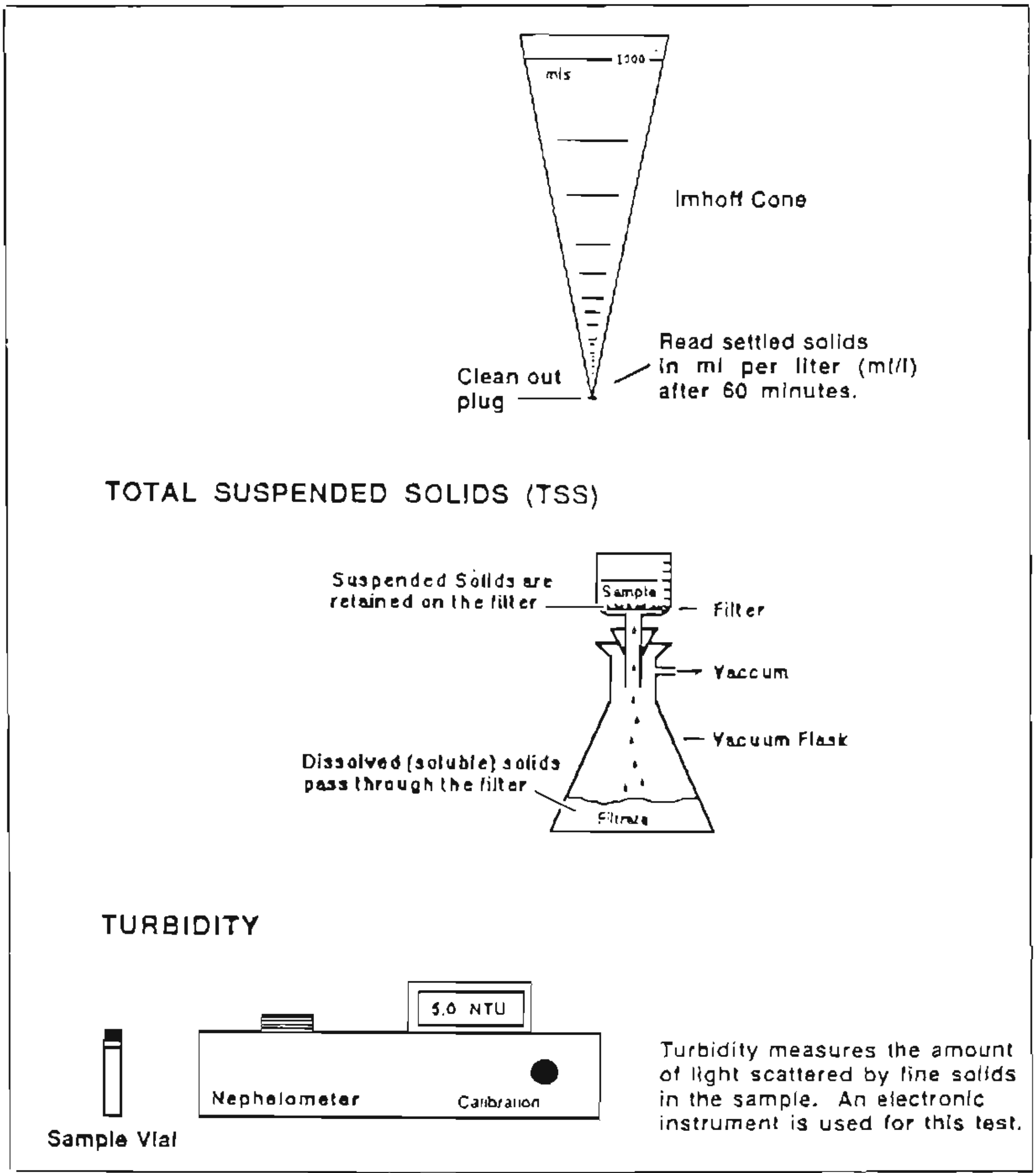


Figure 2. Methods for the measurement of residue in water settleable solids (SS).

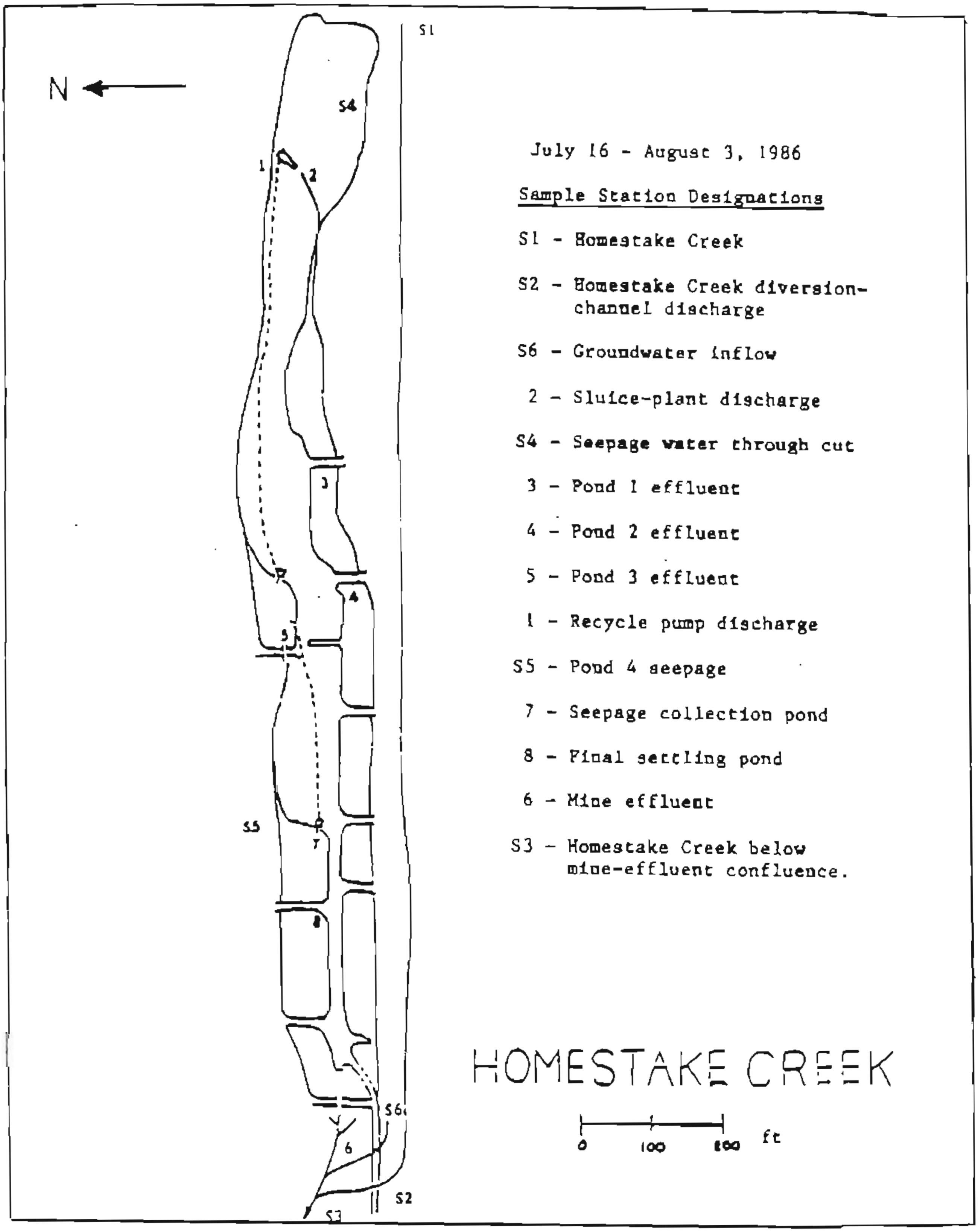


Figure 3. Sample stations, Homestake Creek pond study.

p. 146). Table 1 shows the range of values for residue tests determined on three composite samples taken during July and on four grab samples taken in one 24-hr monitoring period July 18-19. The full set of data from the 24-hr monitoring cycle is presented graphically in figure 4 for SS, in figure 5 for TSS, and in figure 6 for turbidity. The sample station designations for these three figures are shown in figure 3.

FLOCCULANT TREATMENT STUDY

While the monitoring program at Homestake Creek was in progress, a series of laboratory-scale jar tests was run to determine the effectiveness of using flocculants to remove turbidity from the final mine effluent. Sixty-five coagulants and flocculants (table 2) were screened in a series of 111 jar tests on samples of mine effluent in the laboratory in Fairbanks. After the initial screening, the jar-test laboratory was moved to the mine site to test 31 promising chemicals on fresh samples of the mine effluent. In a series of 56 field jar tests, a two-polymer chemistry involving a liquid coagulant followed by a powdered flocculant proved optimum for this project (fig. 7). Four primary coagulants and two secondary flocculants were purchased in sufficient quantity to test on a full-scale treatment system; vendors in Alaska provided information about safe handling and transportation of their products.

A series of jar tests was run on samples taken throughout the Homestake Creek site settling ponds on July 15 to determine the effect of the settling-pond system on the dosage requirements and performance of these polymers. Figure 8 shows the results of these tests, which indicated that by using a multiple-pond system, the polymers, at a given dose, would reduce turbidity and produce less final settling solids after treatment than if applied either directly to the sluice effluent or after only a few settling ponds.

Thus, to minimize the cost and optimize the performance of the flocculant treatment system, the mining operation had to control infiltration water. The multiple-pond system allowed recycling of the 1,500 gpm of water needed for the sluice plant by treating to a quality which did not adversely affect fine-gold recovery, according to the mine operator's screen analyses. Controlled diversion of groundwater and streamwater around the pond system reduced the quantity of wastewater to be treated to less than 200 gpm at the Faith Creek site, where the flocculant treatment system was actually tested. About two-thirds of this effluent was actually contained and diverted through the treatment system. The rest leached through the berm, passed through a series of small settling ponds, and combined with the flocculant treatment system flow before entering Faith Creek (fig. 9).

The chemistry was selected by using mine-effluent samples from the Homestake Creek site. Soon after the chemical order was placed, the mine operation, because of poor pay material at the Homestake site, was relocated downstream to a site on Faith Creek. A series of jar tests on the mine effluent at Faith Creek verified that the selected polymers should perform as well as at Homestake Creek. Before moving from Homestake Creek, a brief pilot test in a 5-gpm system with a small sluice box as a mixing system,

Table 1. Homestake Creek settling-pond sample* data ranges.

Parameter	Sluice effluent	Recycle water	Mine effluent
Settleable solids (ml/L)	9-275	0.4 - 2.5	0 - <0.1
Total suspended solids (mg/L)	17,950 - 213,000	3,280 - 20,800	200 - 3,300
Turbidity (NTU)	12,000 - 42,000	3,500 - 27,000	200 - 5,600
Sample station (fig. 3)	2	1	6

*July 5-6, July 14-15, and July 27-28 composite samples; July 18-19, 24-hr run grab samples (four sets).

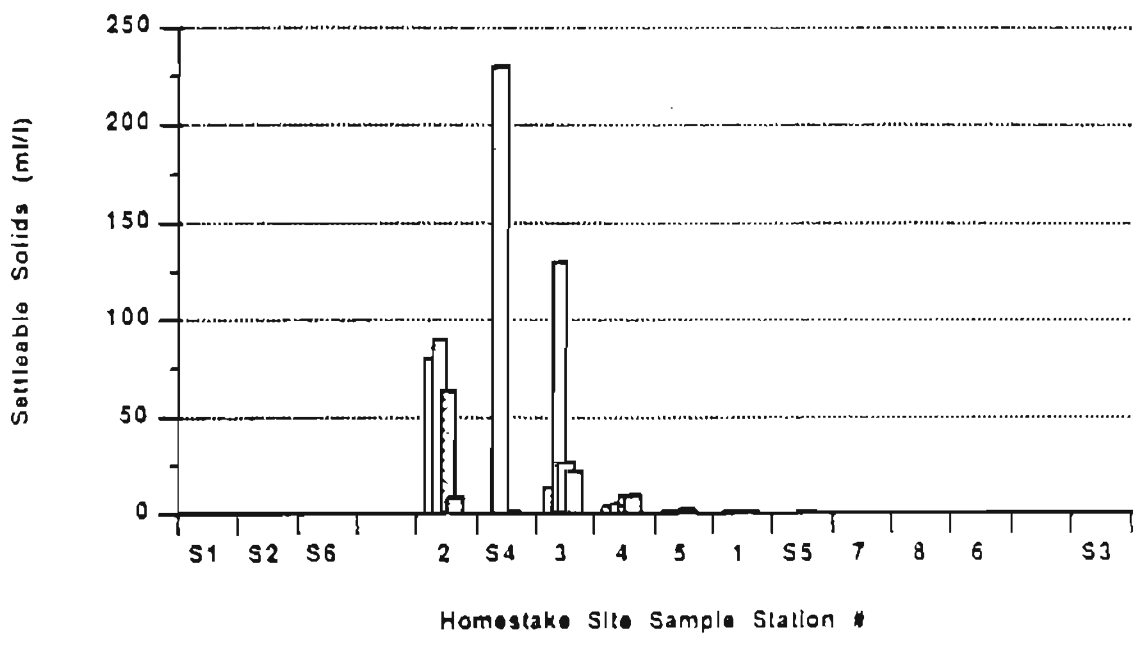


Figure 4. 24-hr SS data, July 18-19, 1986.

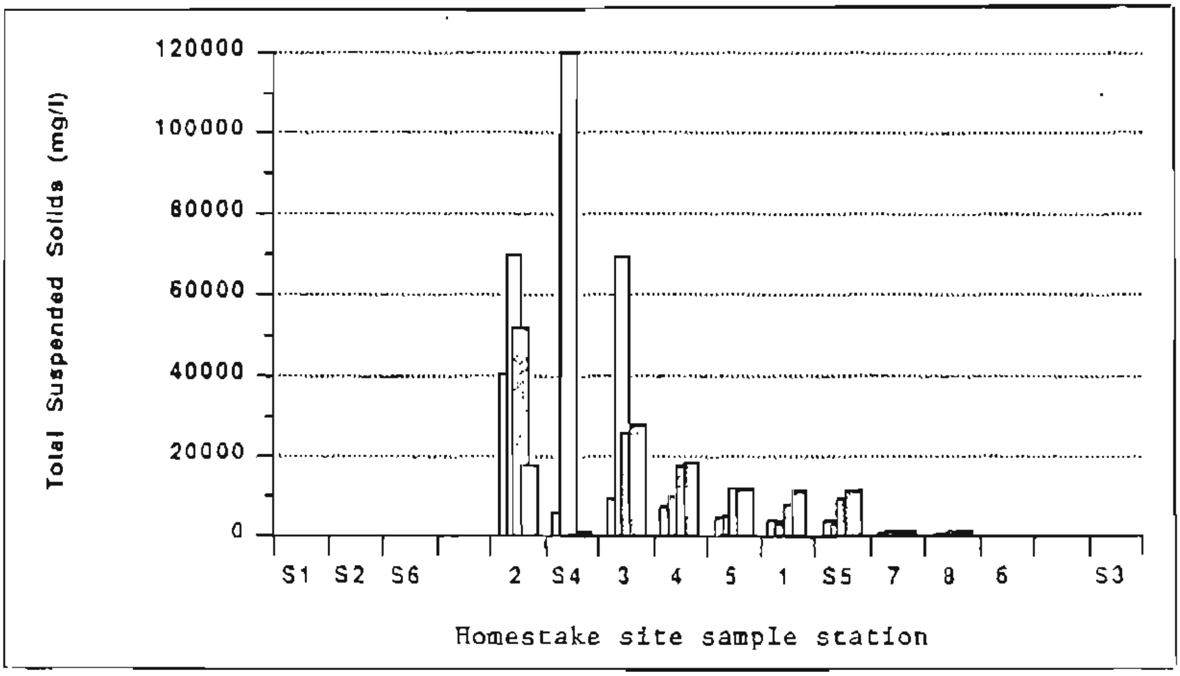


Figure 5. Twenty-four hr TSS data, July 18-19, 1986.

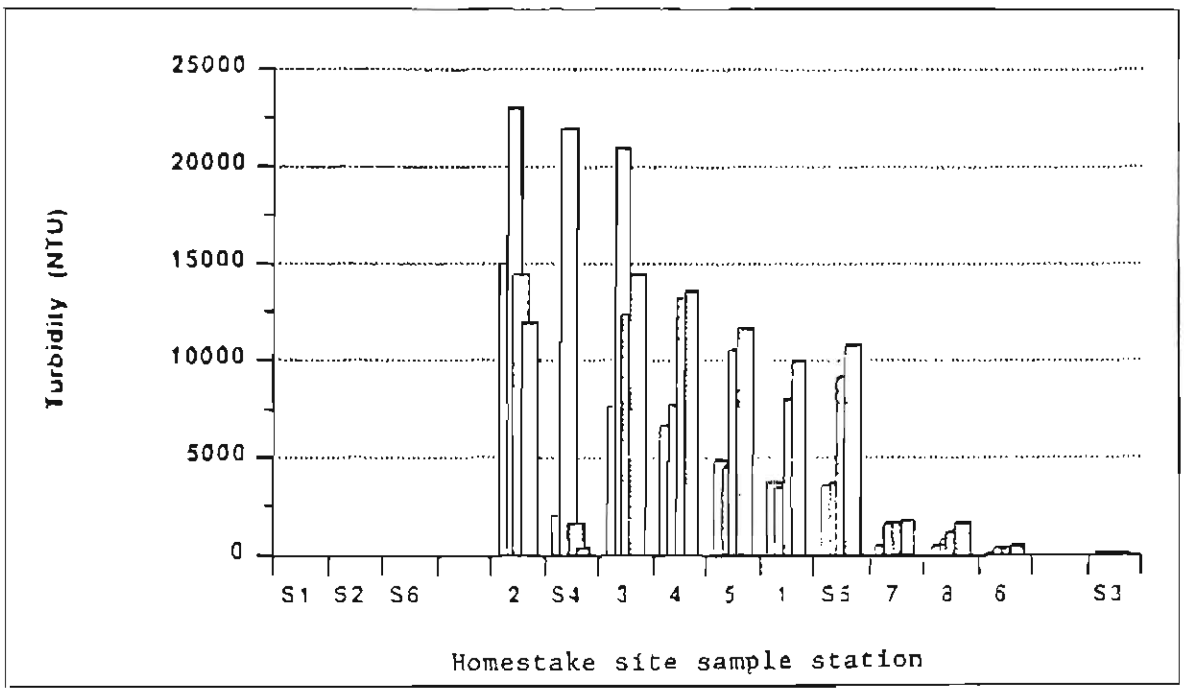


Figure 6. Twenty-four-hr turbidity data, July 18-19, 1986.

Table 2. Coagulants and flocculants selected for screening.

SYNTHETIC POLYMERS: (E = Emulsion, L = Liquid, P = Powder), [Alternate product designation]

Vendor (Trade Names)	Cationic Coagulants	Anionic Flocculants	Nonionic Flocculants
Allied Colloids	401 (L), [701]	E10 (P)	351 (P)
(Percol)	402 (L), [702, LT31] *	E24 (P), [730]	720 (P)
	403 (L), [703]	E207 (P)	
	406 (L), [706, LT35]	155 (P), [725]	
	292 (P), [722]	156 (P), [726]	
	352 (P), [721]	1011 (P), [727] *	
	455 (P)	611 (P)	
	728 (P)		
	757 (P)		
	763 (P)		
	751 (L)		
	776 (P)		
	788N (E)		
American Cyanamid	314 (L)	1202 (E)	16 (P)
(Superfloc)	315 (L)	204 (P)	84 (P)
(Magnifloc)	355 (L), [5516] *	206 (P)	127 (P)
	1303 (E)	208 (P), [834A]	
		210 (P), [837A]	
		212 (P)	
		214 (P)	
NALCO	7134 (L)	7766 (E)	110A(P)
(Nalcolyte, Nalclear)	7107 (L)	8173 (P)	670 (P), [8170]
	8100 (L) *	7774 (E)	
	8103 (L)	7775 (E)	
	8105 (L)	7776 (E)	
	8109 (L), [Ultrion]		
Van Waters & Rogers	4245 (L) *	M-30H (P)	M-30 (P)
(Van Floc)	4275 (L)	M-33H (P) *	3250 (E)
	3310 (E)	3251 (E)	
	M-3358 (E)	3252 (E)	
		3270 (E)	
		3289 (E)	

INORGANIC COAGULANTS:

Chemical Name	Common Name
Aluminum Sulfate	(Alum)
Calcium Hydroxide	(Lime, Hydrated Lime)
Calcium Sulfate	(Gypsum)

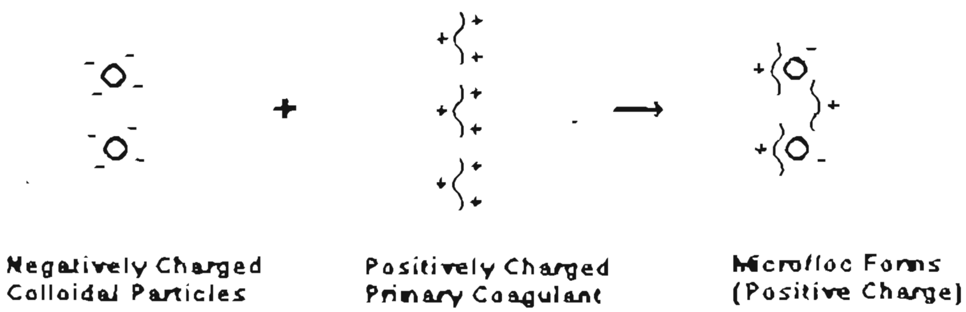
OTHER FLOCCULANTS.

Chemical Name	Common Name
Polyethylene Oxide	(PEO)

* Chemicals selected for full scale testing.

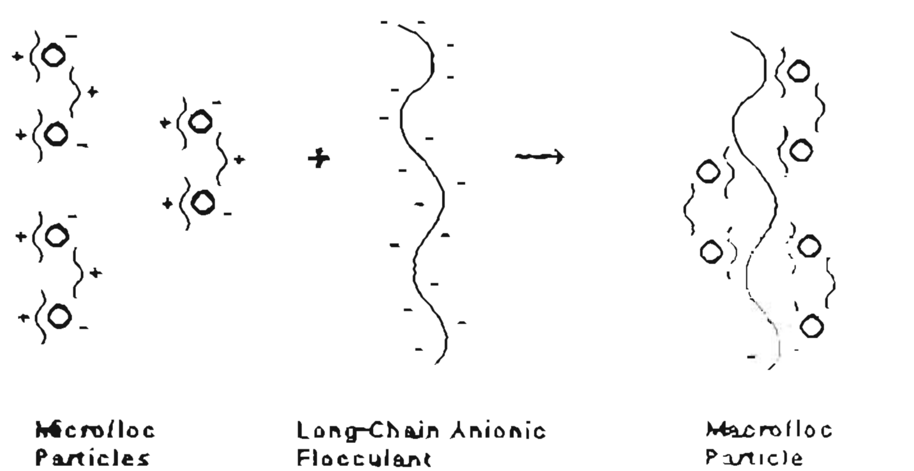
COAGULATION:

Negatively charged colloidal turbidity is held apart by repulsion of like charges until a positively charged coagulant is added, resulting in a larger particle (microfloc) with an average positive final charge.



FLOCCULATION:

A long chain secondary polymer with a high strength negative charge draws the microflocs together into a larger "macrofloc" which has a higher density and will settle faster than the microfloc or colloidal particles.



Polymer charges:

Anionic = negative charge
Cationic = positive charge
Nonionic = balanced negative and positive charges

The polymer charge and the intensity of the charge can be varied depending on the desired application. The length of the chain can also be varied.

Figure 7. Theory of coagulation and flocculation.

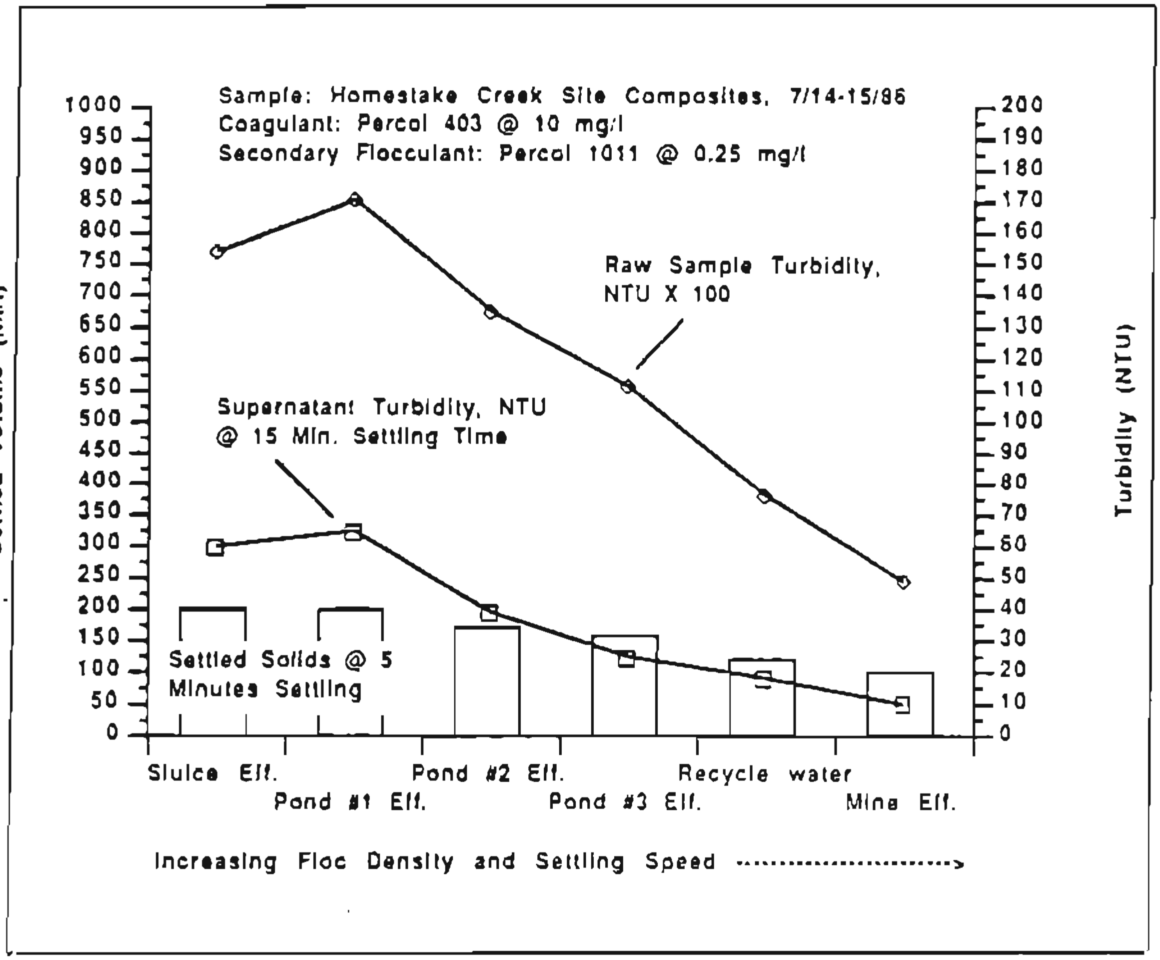


Figure 8. Comparative jar tests of samples through the mine operation.

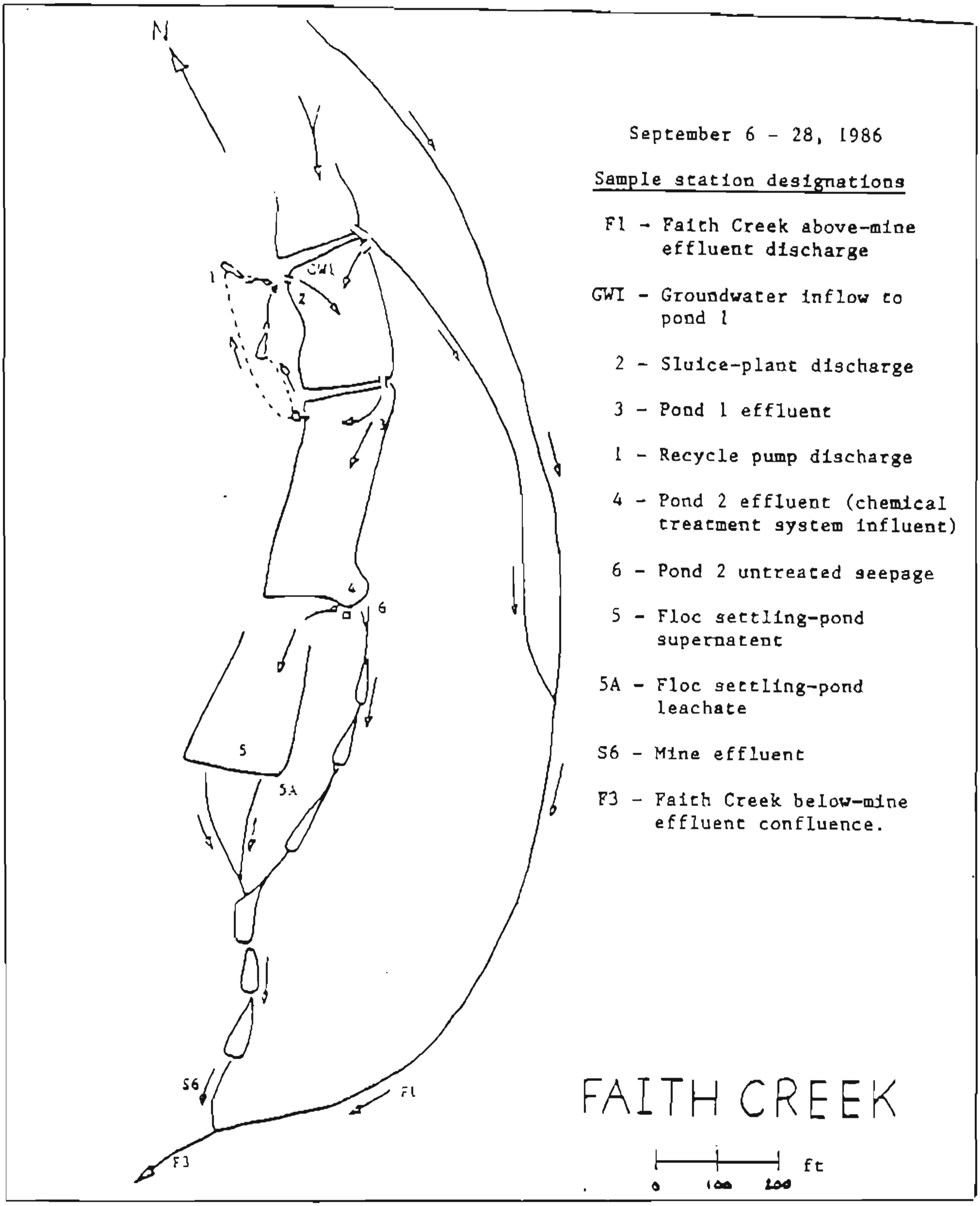


Figure 9. Sample stations, Faith Creek flocculant study.

glass separatory funnels as chemical feeders, several 5-gal plastic pails, and lengths of plastic pipe helped to determine the effectiveness of the polymers on a larger scale treatment of the actual flow. This test achieved an 82 percent reduction of turbidity and strongly suggested the need for very efficient mixing to fully reactivate the polymers.

Following this test, the treatment-system equipment was procured and assembled at the site (fig. 10). A Stranco 'Polyblend' automatic coagulant feed system was used to feed blended coagulant to a Komax in-line static mixer to provide thorough reaction of the coagulant with the mine effluent. The secondary flocculant feed system was constructed by using a plastic drum with a mixer and a Liquid Meteronics chemical feed pump. One minute of detention time between the coagulant and flocculant was provided by a 100-ft loop of 6-in.-diam flexible, corrugated plastic pipe. Another 100-ft-long section of the pipe was used to direct the discharge to a final settling pond where the supernatant could be separated.

Table 3 presents the results of the performance test run from September 7-12, 1986. Following this, the treatment-system operation was continued by the mine operators until September 17, when freezing interrupted the coagulant flow. After the treatment system was shut down, mine-effluent turbidity began to steadily rise until the mine operation was discontinued for the season on September 28, 1986.

During the treatment-system performance test, the final mine effluent impact on Faith Creek was within the ADEC standard of 5 NTU turbidity above background on nine of 11 days. This occurred while sluicing was increasing the turbidity of the two-pond system to over 10,000 NTU prior to shutdown of the treatment system. The maximum 'out of compliance' turbidity on the other two days was 8.1 NTU above background. During the mining season, creek turbidity data collected from both above and below the mining operation showed 10 to 79 days of compliance within the allowable five-NTU-above-background standard as well as the nine days of compliance during the performance test and one other day (September 6), when mine effluent was filling the flocculant-treatment-system feed pond (app. B, p. 147). Turbidity data from the Homestake site and from the Faith Creek site before, during, and after the flocculant treatment system operation are summarized in table 4.

The flocculant-treatment-system discharge also achieved compliance with the EPA standard for total arsenic (0.05 mg/L), treating settling-pond effluent ranging from 0.143 to 0.370 mg/L total arsenic. Virtually all the arsenic was removed in the floc developed by these polymers. This result was expected, since 99 percent of the arsenic was in particulate form and only 1 percent was dissolved.

Total costs for the construction and maintenance of the settling pond and diversion channel system at both the Homestake and Faith Creek sites were about $31,000 for the 100-day mining season. The capital cost for flocculant-treatment-system equipment and assembly was $13,000. Daily operations costs for chemical and fuel were $35-40, plus a labor cost of approximately 3 hr/day, including time for all the routine water-quality monitoring required at the mine site. Costs for the flocculant selection and treatment-system

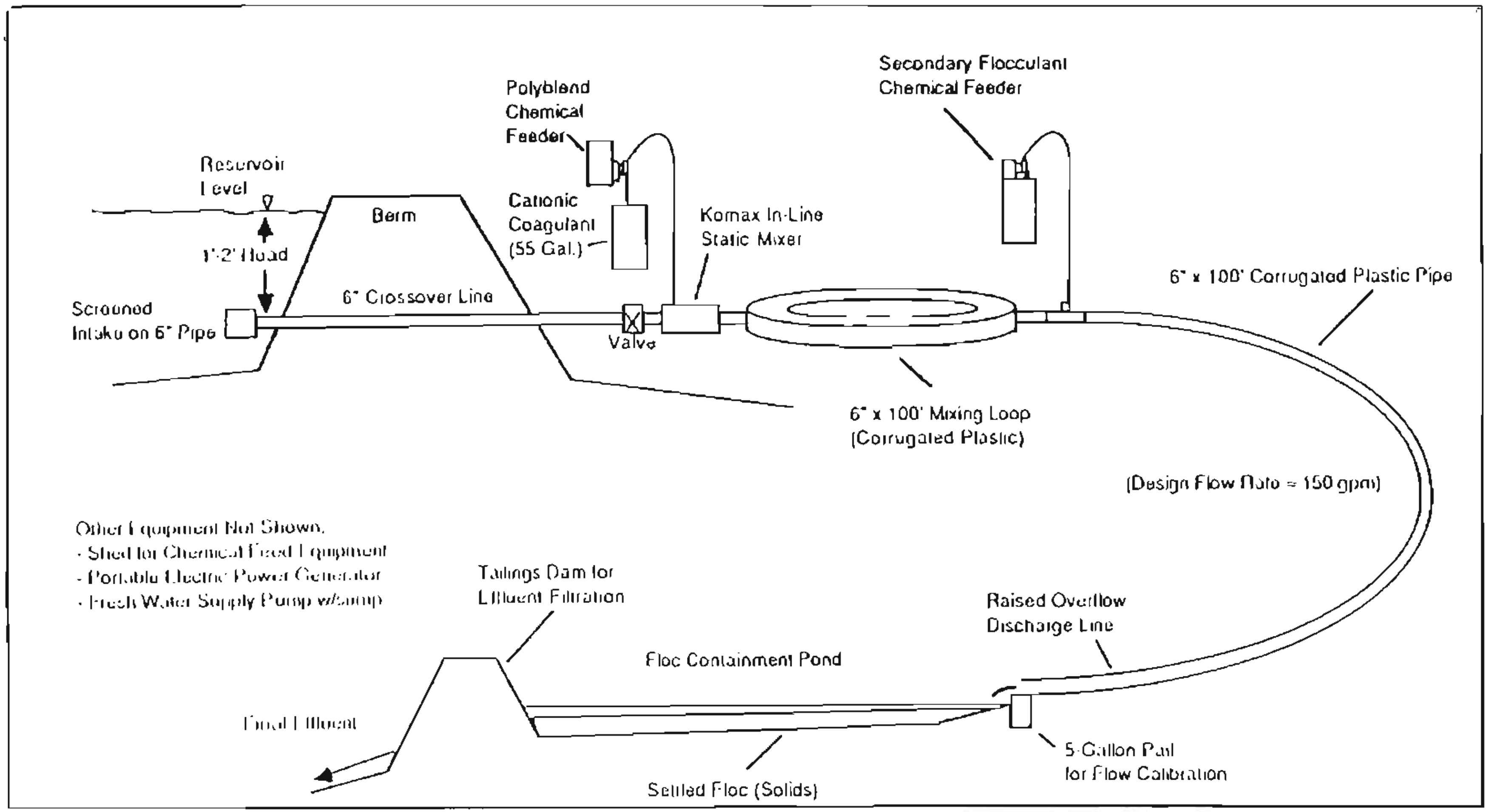


Figure 10. Esperanza Resources Co., Inc., placer-mine water-treatment system, as-built equipment layout (Faith Creek site).

Table 3. Results of full-scale treatment-system performance test.

PARAMETER:	Flow (gpm)	Coagulant Dose (mg/L)	Flocculant Dose (mg/L)	Supernatant Turbidity (NTU) 15 Min.	60 Min.
MINIMUM:	88	7.6	0 22	3.4	3 5
MAXIMUM:	183	12	0.41	27	23
AVERAGE:	111	10	0.28	9.7	7.7
# DATA POINTS:	18	15	17	28	25
NOTES:	(1)	(2)	(3)	(4,5)	

(1) Flow calibrated by three repetitive fillings of a 5-gallon pail at the discharge of the treatment system.

(2) Coagulants were Magnifloc 5516, Nalco 8100, Percol LT31, and Van Floc 4245. All were generically similar cationic polyamines in aqueous (water) solution.

(3) Each coagulant was tested with each secondary flocculant, Percol 727, and Van Floc M-33H. All combinations of coagulants and flocculants attained <10 NTU turbidity, and half achieved <5 NTU. Van Floc M-33H formed a larger floc than Percol 727 at equal dosages, although final turbidity values were similar. Both flocculants are very high molecular-weight polyacrylamide anionic polymers.

(4) Turbidity readings were from settled supernatant of 1,000-ml samples drawn from the floc discharge to the 5-gallon pail. Following an optimization period on the first day, 75% of all supernatant readings after 15 minutes, and 84% after 60 minutes of settling were <10 NTU.

(5) Influent turbidities ranged from 1,300 to 2,800 NTU during the performance test period.

Table 4. Summary of Esperanza Resources Co., Inc., 1986 daily turbidity data.

LOCATION:	Homestake Cr.	Faith Cr. (Before)†	Faith Cr. (During)†	Faith Cr. (After)†
DATES:	7/10 - 8/11	8/9 - 9/6	9/7 - 9/17	9/18 - 9/28
NO. of SAMPLES:	33	29	11	11
TURBIDITY (NTU)				
Creek Background:	0.40	2.2 *	1.8	2.1
Floc Settling Pond Eff:			29	
Mine Effluent:	2200	950	140	850
Creek Below Effluent:	550	120	4 5	26

* Average of 28 data points, not including 950 NTU reading on 8/21/86. Average of all data is 40 NTU.

† Before, during and after operation of the chemical treatment system.

design are site specific, but may be reduced by drawing on the experience gained in this ADEC-funded project. Flocculant-selection services sufficient to select an optimum chemistry for a placer mine are commercially available in Alaska for under $1,000.

The success of this project can be attributed to several factors, each of which is considered vital by the principal investigators. These factors are chemistry (selection of the best chemicals and feed equipment), control (multiple-settling-pond system and complete recycle), and attitude (the mine owners, Richard and Mary McIntosh provided the location, cooperation, and patience to allow this treatment technology to be demonstrated).

APPENDIX A

Esperanza Resources Co., Inc.
Placer-mine water quality field analysis data.
Summary of daily field data collection at the Homestake Creek site for July and August 1986.

			Station S1 (Homestake Creek above cut)			Station S2 (Creek diversion outlet)			Station 6 (Mine effluent)			Station S3 (Combined flow from S2 & 6)			Sluice
Date	Time	Weather	Temp °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Plant Operating
July 10	2142	Rain	8.0	0.0	0.25	8.0	0.0	1.0	10.0	TR	360	10.0	TR	230	
11	1915	Rain	7.0	0.0	0.90	7.0	0.0	1.5	10.0	TR	450	8.0	TR	140	•
12	1915	Clear	8.0	0.0	0.35	9.0	0.0	2.1	11.0	TR	1700	10.0	TR	850	•
13	1920	Rain	6.0	0.0	0.40	6.0	0.0	2.0	9.0	TR	2600	8.0	TR	1400	•
14	1915	Partly Cloudy	8.0	0.0	0.60	9.0	0.0	1.4	10.0	TR	3300	10.0	TR	2100	•
15	1900	Partly Cloudy	9.0	0.0	0.40	11.0	0.0	1.4	12.0	TR	4000	11.0	TR	3000	•
16	2330	Clear	6.0	0.0	0.30	7.0	0.0	1.2	9.0	TR	950	9.0	TR	750	
17	2100	Partly Cloudy	10.0	0.0	0.30	ND	ND	ND	10.0	TR	370	10.0	TR	400	
18	1615	Rain	7.5	ND	0.40	6.0	TR	4.3	9.5	0.0	200	7.0	0.0	80	•
19	*	Rain	6.0	0.0	0.35	7.0	TR	14	9.0	TR	450	7.5	TR	160	•
20	2100	Rain	3.5	TR	1.0	3.5	0.2	12	7.5	TR	3800	4.5	TR	750	•
21	2100	Partly Cloudy	6.0	0.0	0.40	6.0	0.1	4.0	8.5	TR	1900	6.5	TR	230	•
22	1900	Clear	6.0	0.0	0.30	7.0	TR	3.0	9.5	TR	3800	8.0	TR	1000	•
23	1900	Rain	5.0	0.0	0.30	5.0	0.0	4.0	8.5	TR	3300	5.0	TR	950	•
24	1200	Rain	4.0	0.0	0.30	4.0	0.0	4.0	8.0	TR	2300	5.0	TR	700	
25	2030	Rain	5.0	0.0	0.30	5.0	0.0	5.0	8.0	TR	1100	5.5	TR	240	
26	2050	Rain	4.0	0.0	0.40	4.0	0.2	600	7.5	TR	400	5.0	0.2	500	•
27	1950	Partly Cloudy	6.0	0.0	0.50	6.0	0.2	200	8.0	TR	650	6.0	0.2	280	•
28	2015	Cloudy	6.0	0.0	0.40	6.0	0.2	110	9.0	TR	2300	6.5	0.2	400	•
29	1930	Cloudy	8.0	0.0	0.30	8.0	0.1	60	9.0	TR	4500	8.0	0.1	550	•
30	1930	Rain	4.0	0.0	0.50	4.0	0.2	35	8.0	TR	5200	5.0	0.2	650	•
31	2130	Rain	4.0	0.0	0.50	4.0	0.2	12	8.0	TR	7400	5.0	0.2	700	•
August 1	2000	Rain	4.0	0.0	0.50	4.0	0.15	40	8.5	TR	7600	5.5	0.15	500	•
2	1930	Clear	6.5	0.0	0.30	6.5	TR	11	9.8	TR	6400	7.5	TR	450	•
3	2050	Clear	6.5	0.0	0.30	6.5	TR	8.0	10.0	TR	4000	7.5	TR	400	•
4	2000	Partly Cloudy	6.0	0.0	0.40	6.0	TR	25	9.0	TR	1500	7.0	TR	150	
5	1945	Rain	ND	ND	ND	ND	ND	ND	9.0	TR	850	5.0	TR	80	
6	ND	Partly Cloudy	ND	ND	ND	ND	ND	ND	9.5	TR	600	8.5	TR	50	
7	2030	Partly Cloudy	5.0	0.0	0.30	6.0	TR	4.0	9.5	TR	400	5.5	TR	40	
8	1945	Cloudy	4.5	0.0	0.30	4.5	TR	4.0	9.0	TR	230	5.0	TR	30	
9	2040	Partly Cloudy	ND	ND	ND	5.0	TR	4.0	9.0	0.0	190	6.0	TR	20	
10	1930	Partly Cloudy	ND	ND	ND	ND	ND	ND	9.0	0.0	130	6.0	ND	14	
11	1030	Partly Cloudy	ND	ND	ND	ND	ND	ND	9.0	ND	100	6.5	ND	14	
TURBIDITY DATA															(• = Yes)
Average					0.41			42.4			2213			540	
Minimum					0.25			1			100			14	
Maximum					1			600			7600			3000	
No. of Data Points					33			33			33			33	

NOTES: ND = No Data
SS = 60 Minute Settleable Solids by Imhoff Cone
TR = Trace (<0.1 ml/l)
* Average of 3 grab samples taken on 7/19/86
Sluicing at the Homestake Creek site was completed on 8/3/86, and restarted at the Faith Creek site on 8/9/86.

APPENDIX B

Esperanza Resources Co., Inc.

Placer-mine water quality field analysis data.

Summary of daily field data collection at the Faith Creek site for August and September 1986.

			Station F1 (Faith Creek above discharge)			Station 5A (Treatment system effluent)			Station S6 (Mine effluent)			Station F3 (Combined flow from S6 & F1)			Sluice
Date	Time	Weather	Temp. °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Temp. °C	SS ml/l	Turbidity NTU	Plant Operating
August 9	1930	Partly Cloudy	ND	ND	ND				14.0	0.0	35	ND	ND	ND	•
10	1900	Partly Cloudy	ND	ND	ND				10.0	TR	1800	ND	ND	ND	•
11	1900	Partly Cloudy	ND	ND	ND				10.0	TR	1700	ND	ND	ND	•
12	1900	Partly Cloudy	ND	ND	ND				10.0	TR	2400	ND	ND	ND	•
13	ND		ND	ND	ND				ND	ND	ND	ND	ND	ND	•
14	1800	Clear	10.5	TR	2.0				13.0	TR	1300	10.5	TR	140	•
15	1315	Partly Cloudy	8.0	TR	2.3				10.0	TR	1300	9.0	TR	320	•
16	2040	Rain	7.0	0.0	1.9				9.0	TR	1300	7.5	TR	170	•
17	2230	Rain	6.0	0.0	1.8				8.0	TR	700	7.0	0.0	80	•
18	2045	Rain	6.0	0.0	2.3				8.0	TR	130	6.0	0.0	10	•
19	2000	Partly Cloudy	6.0	0.0	2.1				8.0	TR	800	8.5	0.0	130	•
20	1830	Rain	6.0	TR	2.2				8.0	TR	1100	6.0	TR	140	•
21	1950	Rain	3.0	1.3	950				8.0	TR	1100	3.0	1.5	1000	
22	2030	Clear	3.0	TR	8.5				6.0	TR	220	3.5	TR	14	
23	1830	Clear	4.0	TR	3.0				6.0	TR	190	4.0	TR	11	•
24	1815	Rain	4.0	TR	2.7				5.0	TR	900	4.0	TR	55	•
25	1810	Rain	5.5	0.0	2.4				7.0	TR	1400	5.5	TR	150	•
26	1745	Clear	6.0	0.0	2.2				7.5	TR	1200	6.0	TR	110	•
27	2000	Rain	6.0	0.0	1.9				8.0	TR	1200	6.0	TR	100	•
28	1815	Rain	6.0	0.0	1.6				8.0	TR	1300	6.0	TR	30	•
29	1815	Clear	6.0	0.0	1.9				8.0	TR	650	6.0	TR	45	•
30	1800	Clear	6.5	0.0	1.7				8.0	TR	500	6.5	TR	35	•
31	1740	Partly Cloudy	7.5	0.0	2.0				8.0	0.0	1000	8.5	0.0	70	•
Sept. 1	1750	Partly Cloudy	6.5	0.0	1.4				7.5	TR	1200	7.0	TR	70	•
2	1800	Rain	6.0	0.0	1.2				7.0	TR	1300	6.5	TR	70	•
3	1800	Rain	6.0	0.0	1.2				7.0	TR	1300	6.5	0.0	70	•
4	1815	Clear	6.5	0.0	1.2				6.5	0.0	750	6.5	0.0	45	
5	1815	Clear	7.0	0.0	1.2				7.5	0.0	110	7.0	0.0	12	
6	2100	Clear	5.0	0.0	1.3				7.0	0.0	80	8.5	0.0	4.7	
7	1915	Partly Cloudy	5.0	0.0	2.4			17	7.0	0.0	40	5.5	0.0	4.2	•
8	1930	Rain	5.0	0.0	1.9			11	7.0	0.0	260	5.0	0.0	10	•
9	2100	Rain & Snow	3.5	0.0	1.7			15	5.5	TR	110	4.0	0.0	3.9	
10	2100	Partly Cloudy	3.5	0.0	1.4			1.5 †	5.5	0.0	63	4.0	0.0	2.8	
11	2100	Partly Cloudy	5.0	0.0	1.7			5.3 ‡	7.0	0.0	55	5.0	0.0	2.2	
12	2130	Partly Cloudy	5.0	0.0	1.1			9.8 ††	7.0	0.0	45	5.0	0.0	1.9	•
13	2000	ND	5.0	0.0	1.7	7.0	0.0	2.4	7.0	0.0	80	5.0	0.0	2.8	•
14	2020	ND	5.0	0.0	2.1	7.0	0.0	21	7.0	0.0	180	5.0	0.0	4.2	•
15	2030	ND	5.0	0.0	2.1	7.0	0.0	45	7.0	0.0	180	5.0	0.0	5.0	•
16	2010	ND	5.0	0.0	2.2	7.0	0.0	18	7.0	0.0	200	5.0	0.0	5.6	•
17	1700	ND	6.0	0.0	1.4	8.0	0.0	170 ‡††	8.0	0.0	300	8.0	0.0	7.3	•
18	1740	Partly Cloudy	4.0	0.0	1.4				6.0	0.0	280	4.0	0.0	11	
19	1930	Rain	4.0	0.0	1.5				6.0	0.0	450	4.0	0.0	11	•
20	1020	Snow	4.0	0.0	1.3				6.0	0.0	600	4.0	0.0	13	•
21	1950	Partly Cloudy	2.0	0.0	1.2				5.0	0.0	800	2.0	0.0	14	•
22	1830	Partly Cloudy	2.0	0.0	1.1				5.0	0.0	1000	2.0	0.0	24	•
23	1900	Snow	ND	0.0	10.5				ND	0.0	850	ND	0.0	25	•

Date	Time	Weather	Station S1 (Homestake Creek above cut) Temp. °C	SS ml/l	Turbidity NTU	Station S2 (Creek diversion outlet) Temp. °C	SS ml/l	Turbidity NTU	Station 6 (Mine effluent) Temp. °C	SS ml/l	Turbidity NTU	Station S3 (Combined flow from S2 & 6) Temp. °C	SS ml/l	Turbidity NTU	Sluice Plant Operating
24	ND	Clear	2.0	0.0	1.3				5.0	0.0	1200	2.0	0.0	50	+
25	ND	Clear	2.0	0.0	1.3				6.0	0.0	1200	2.5	0.0	50	+
26	ND	Clear	2.0	0.0	1.4				4.0	0.0	1100	2.0	0.0	35	
27	ND	Partly Cloudy	2.0	0.0	1.2				4.0	0.0	1100	2.0	0.0	20	
28	ND	Cloudy	2.0	0.0	1.2				4.0	0.0	1000	2.0	0.0	28	
TURBIDITY DATA:															(+ = Yes)
Average:					23 **			29			761			70	
Minimum:					1.1			1.5			35			1.9	
Maximum:					950			170			2400			1000	
No. of Data Points:					51			11			61			51	

APPENDIX B, Cont.
Summary of Daily Field Data Collection at the Faith Creek Site for August and September 1986 (Page 2).

NOTES: ND = No Data
SS = 60 Minute Settleable Solids by Imhoff Cone
TR = Trace (<0.1 ml/l)
* Sample collected as a two grab composite at approximately 1630 & 2130
** Faith Creek background average turbidity data without August 21 data was 2.1 NTU.
† Sample taken from treatment pond supernatant inside berm. All other treatment system effluent samples were floc pond leachate
†† Composite leachate sample data
††† Treatment system only operated for 1/2 day. Sample taken several hours after system was shut down.

INTRUSIONS IN THE FAIRBANKS AREA AND THEIR RELATION TO MINERALIZATION

by

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Early reports on the Fairbanks district and the Yukon-Tanana region suggested that mineralization could be related to intrusive activity. Early in this century Prindle and Katz (1909) stated igneous rocks are common in all the gold-placer areas of the Yukon-Tanana region, and the available evidence points to them as being, indirectly at least, the cause of the gold occurrences from which the placers have been derived. Their brief (20 p.) work, which describes the distribution of igneous rocks, their delineation from the sedimentary rocks, their mode of origin, and the role they have played in the geologic history of the region, is of prime importance to those interested in mining.

A few years later, Smith (1913) reported that the areas near the intrusive granitic rocks are the places where lodes are most likely to occur. Outcrops of these igneous rocks are widely distributed throughout the Fairbanks region, and in many places where these rocks are not exposed they probably occur a relatively short distance below the surface (Smith, 1913, p. 211).

Hill (1933) supported the relationship between mineralization and intrusions by stating the gold mineralization of the region was intimately connected with acidic intrusive rocks. Most of the gold quartz veins are found in the schists near either dikes or larger stocks of intrusive rock (Hill, 1933, p. 46).

Mertie (1937) supported this relationship. He emphasized the importance of the smaller intrusive rocks and recommended that miners prospect "around bodies of granite rocks, more particularly near the smaller bodies" (Mertie, 1937, p. 265).

Twenty years later, Beyers (1957) concentrated on tungsten deposits in the Fairbanks district. He concluded that there were two kinds of igneous intrusive targets favorable for prospecting for scheelite-bearing lodes---areas near small porphyritic granite bodies and areas at the contact of discordant quartz-rich pegmatite dikes with porphyritic granite, country rock, and limestone. Beyers recommended prospecting in the Pedro Dome, Gilmore Dome, and Fox Creek areas, as well as in the country rock surrounding any small unmapped areas of porphyritic granite. Beyer's work encouraged further study of the Fairbanks intrusions.

More recent studies of the Pedro and Gilmore intrusions now provide information on the age and relations of the intrusions, their chemical composition, and detailed maps of their exposure (fig. 1).

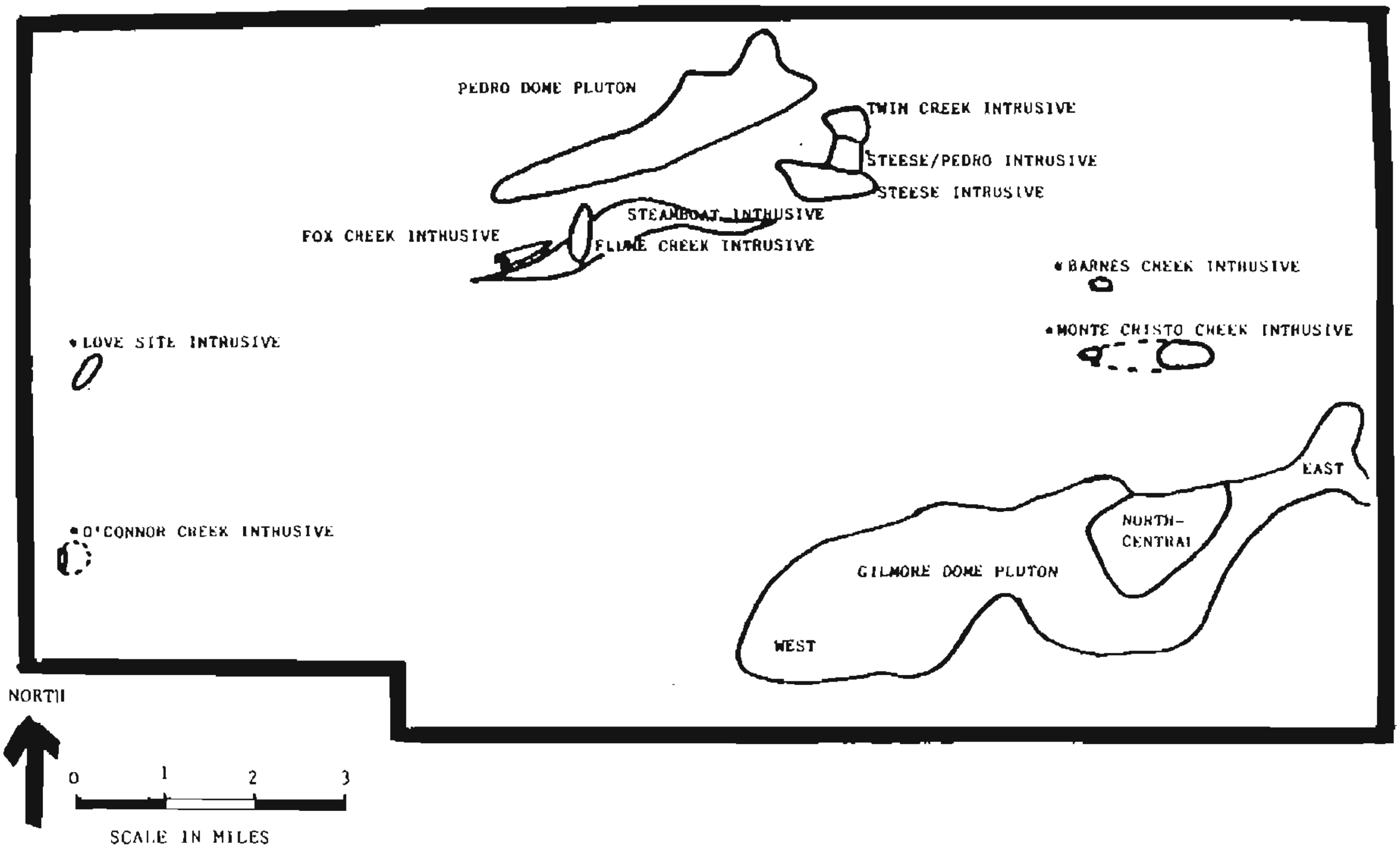

Figure 1. Selected intrusives of the Fairbanks mining district (after Britton [1970], Sherman [1983], and Blum [1983]. *Denotes intrusive name proposed in this report.

In the last decade or two, the relationship between mineralization and intrusive activity became more definite. In 1962 Brown concluded that the mineral deposits in the Fairbanks area were formed by deposition from hydrothermal solutions from quartz monzonite magmas. Britton (1970) suggested that the association of gold mineralization with shear zones and alkalic intrusives was related to the emplacement of late differentiates that were rich in gold-carrying hydrothermal fluids. Chapman and Foster (1969) discussed the relationship of lode deposits to granitic rock bodies. They analyzed random samples of various igneous bodies to obtain a rough evaluation of their metal content. Chapman and Foster (1969) established threshold values for eight elements generally associated with deposits in the district and suggested that the geochemical signature of intrusions might be a useful indicator in targeting lode areas.

Sherman (1983) concluded that mineralization of quartz diorite and quartz monzonite represented two separate mineralization events in the Fairbanks area.

The same year, Blum defined a fractionation series that was responsible for the compositional variations within intrusive rocks in the Fairbanks area. He proposed a crystallization history for the Fairbanks intrusives:

> "...Pedro Dome was the first phase to crystallize, followed by the porphyritic granodiorite phase that forms two small plutons [Monte Cristo Creek and Steese/Pedro intrusive?] and the central portion of the Gilmore Dome pluton. The next magma to crystallize was the porphyritic quartz monzonite phase that forms one small pluton at Pedro Dome [Twin Creek intrusive] and the bulk of the Gilmore Dome pluton. The final phase to crystallize was the aplite rocks that form dikes (Blum, 1983, p. 15)."

In 1982, a joint mapping effort by the Alaska Division of Geological and Geophysical Surveys and the University of Alaska-Fairbanks produced several bedrock geologic maps and reports on the Fairbanks mining district (Forbes and Weber, 1982; Metz, 1982; Robinson, 1982; and Bundtzen, 1982). Details of the metamorphic rock associations resulted in the naming of the Fairbanks, Chatanika, Goldstream, and the Cleary sequences. The Cleary sequence is interbedded within the Fairbanks schist and is composed of rock types including metatuffs, greenstones, marbles, mica schist, graphitic schist, and quartzite. Most mineralization in the Fairbanks mining district is confined to this specific volcanic-sedimentary package (Robinson, 1982; Bundtzen, 1982).

Several factors were probably responsible for the mineralization of the Fairbanks area including late magma differentiates rich in mineralized solutions, fracture or shear zones with structural controls, and favorable metamorphic rocks in which mobilization of fluids could concentrate relatively small amounts of trace elements. Boyle (1965) discussed similar structural factors in his study of the Keno Hill-Galena Hill area (Yukon Territory) and the importance of graphitic schists, phyllites, quartzites, and greenstones as "an enormous reservoir of...major constituents of the deposits of the area."

The Cleary-sequence mapping opened up new possibilities in locating favorable environments for lode deposits in the Fairbanks area. Where intrusions are not exposed, other key rocks may encourage prospecting.

The O'Connor Creek intrusion illustrates an area that displays many factors favorable for mineralization. These rocks display evidence of contact effects within chlorite and mica schists. The identification of intrusive phases that might favor mineralization are being investigated by the author as well as geophysical and geochemical techniques which may be used to identify bedrock, contact metamorphic effects, and potential veins.

In summary, the intrusive history of the Fairbanks area is highly significant and understanding it cannot help but target new areas of potential mineralization.

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THE NATURE OF ISLAND-ARC SYSTEMS AND GEOTECTONIC REGIMES IN ALASKA

by

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This paper will compare aspects of Island Arc Systems studied by the author in the 1970s in the Kurile-Kamchatka region of the U.S.S.R. with the island arcs and geotectonic systems of the Aleutian Islands, Alaska.

The study of the island arcs and their geotectonic systems is of critical importance in understanding the theory of plate tectonics. Island-arc generation is generally considered to be one of the most significant geologic activities involved in plate tectonic motion, namely, compression and subduction of oceanic lithosphere.

Island arcs can be defined by their characteristic structural features, which includes a linear geanticlinal belt located between two depressions. The depression in front of the belt (the leading edge) is generally a narrow volcanic trench, whereas the depression (trailing edge) is usually a comparatively wide deep-water basin or marginal sea. The geanticline usually consists of two independent belts---an outer arc and a back arc---divided by a linear topographic depression partly filled by a thick sedimentary sequence. The outer (leading) arc contains tectonically derived (nonvolcanic) sedimentation formed by tectonic uplift. The back arc contains abundant accumulations of volcanic products; its role in tectonic uplift and sedimentation is negligible.

These island arcs generally display linear belts of strong negative gravity anomalies, active volcanism (usually of calc-alkaline affinity), and active seismicity (a belt of earthquake foci 50-75 km wide occurs along a Benioff zone dipping at 50-60°). Depth of earthquake foci within the Benioff zone ranges from 0 km (at the trench) to 600 km at the rear part of the geotectonic system (marginal seas or equivalent feature). In the past 20 yrs, the Benioff zone has been considered to be a reflection of the subducting lithospheric slab.

A Quaternary volcanic belt is usually located on the side of the geanticline facing the marginal sea. Quaternary volcanoes may be distributed in en-echelon manner, coinciding with the position of open fissures along strike-slip faults. Stratula (1969) noted that in the Kuriles, the type and age of Quaternary volcanoes changed regularly along these echelons. These echelons may also be divided by series of uplifted blocks that may reflect the position of drag folds along strike-slip faults.

Geotectonic systems of island arcs are usually divided into two groups on the basis of their structural features (single vs double arcs) and the height of erosional-tectonic relief (Izu-Bonin vs Kurile-type systems) (Erlich and Groshkov, 1979). An island arc may change into another type of island arc along strike. For example, a single-island arc occurs in the central Kuriles and is gradually replaced north and southward by double arcs; comparable is the single-arc system of the western Aleutians, which is gradually replaced eastward by a double-arc system.

Within the Circum-Pacific belt, typical island arcs are commonly interrupted by elevated blocks that, on one side, preserve the structural and volcanic features characteristic of island arcs while on the other side contain features that are not specific to island-arc systems. These systems have been divided into two types of geotectonic systems: 1) early orogenic (Kamchatka-type) geotectonic systems (found in the south and eastern Kamchatka blocks, Alaska Peninsula, northeastern Japan, the Phillipines, Sumatra, and the North Island of New Zealand) and 2) mature orogenic geotectonic systems (northern Kamchatka, Taiwan, Lusonblock in the Phillipines, and the South Island of New Zealand) (Erlich and Gorschkov, 1979).

NATURE OF THE BENIOFF ZONE

Benioff zones are obviously the most impressive and probably the most important deep-seated feature of island-arc geodynamics. The most common way to study Benioff zone geometry is to study the projection of earthquake foci perpendicular to the geotectonic system. When investigating the geological and geophysical characteristics of Benioff zones, several important features have to be considered.

The trailing edge of these zones is sharp and is expressed by a zone of steep gradient isoclines. This zone coincides with the high gradient zone of linear negative-gravity anomalies, located between the oceanic trench and an island-arc system. These anomalies have been described for Indonesia (Bemmelen, 1970). Anomalies of the same type were also described along other island arcs (Minato and others, 1965). A deep-seated seismic sounding study performed in the Kuriles indicates that these gravity anomalies reflect a linear zone of sharply increased thickness of the crust (fig. 1). The geological nature of this line is easily observable in areas where normal island arcs enter uplifted blocks such as Japan, New Zealand, and Kamchatka. In these areas, major deep-seated fault zones divide paired systems of belts with different types of metamorphism (for example, the Alpine Fault in the South Island of New Zealand and the Median Line in southwestern Japan). In Kamchatka, this fault zone coincides with the boundary between a series of uplifted blocks that separates the geanticline system of the eastern peninsulas and the eastern volcanic belt. In Alaska, a possible analogy of this type of fault is the deep-seated Denali fault zone.

Benioff zones characteristically display a blocky structure reflected in the different density of earthquake foci patterns. Along the strike of corresponding geotectonic systems, gaps exist between areas of major earthquakes. This occurs in the Kurile-Kamchatka region (Fedotov, 1966) and in the Aleutian region (Sykes and others, 1980). With time these gaps may fill. This tendency provides possibilities for the statistical probability of the occurrence of major earthquakes and may be useful in earthquake prediction.

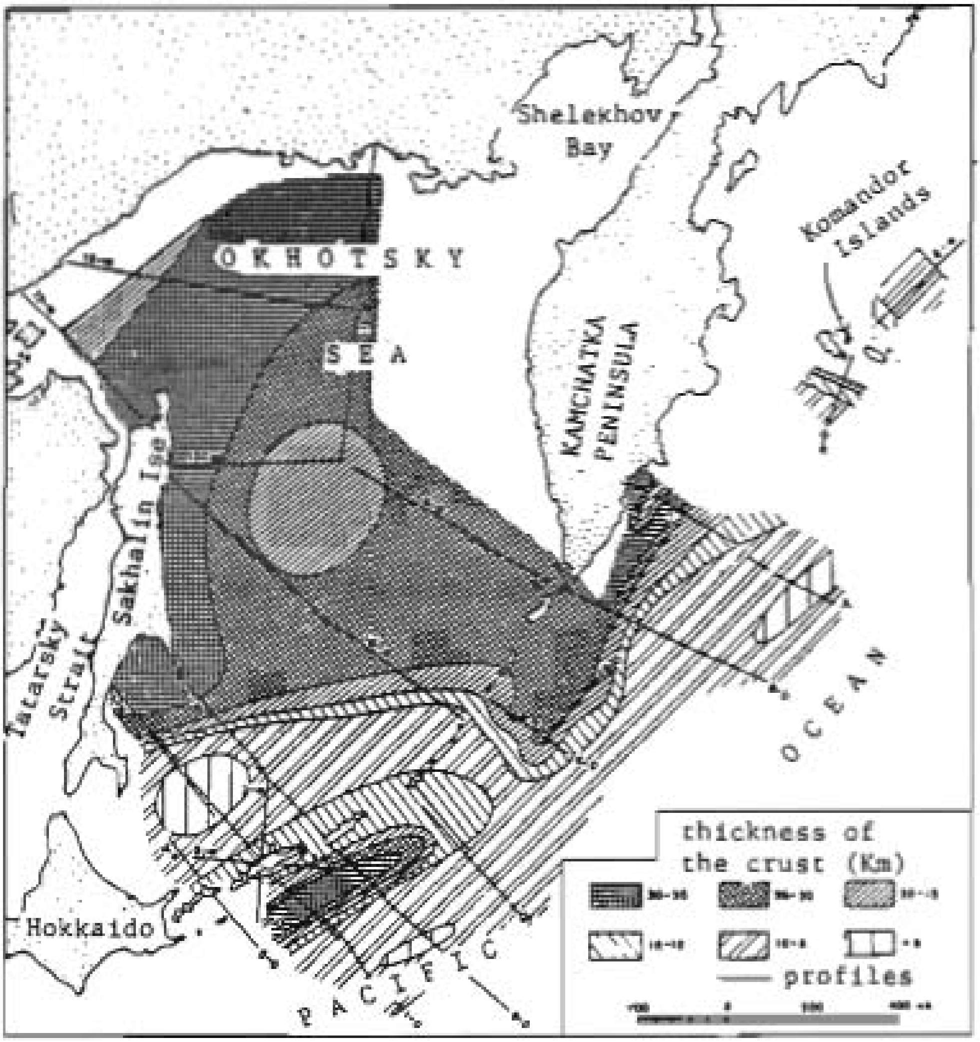

Figure 1. Increasing thickness of the crust in the Kamchatka region, interpreted from earthquake foci distribution patterns (from Kosminskaya and others, 1964).

Areas of major earthquakes in any given moment of time can be considered to be blocks with a tendency to advance (accelerated movement) toward the front of the geotectonic system. Because of the uneven distribution of earthquake energy, there may well be a system of strike-slip faults within and along different blocks, which displaces the major deep-seated fault zone that corresponds to the trailing edge of the Benioff zone.

The major stress vector of earthquake foci is not oriented perpendicular to the strike of the geotectonic system but rather strikes at an acute angle (fig. 2). The general distribution of forces on the Benioff zone boundary is generalized in figure 3. As a result of the orientation of forces along major faults on the Benioff-zone boundaries, horizontal displacement may occur along the strike of the geotectonic system. Because of constant reorientation of the main stress vector of earthquake foci, the direction of horizontal displacement along the strike of the Benioff zone will depend on the

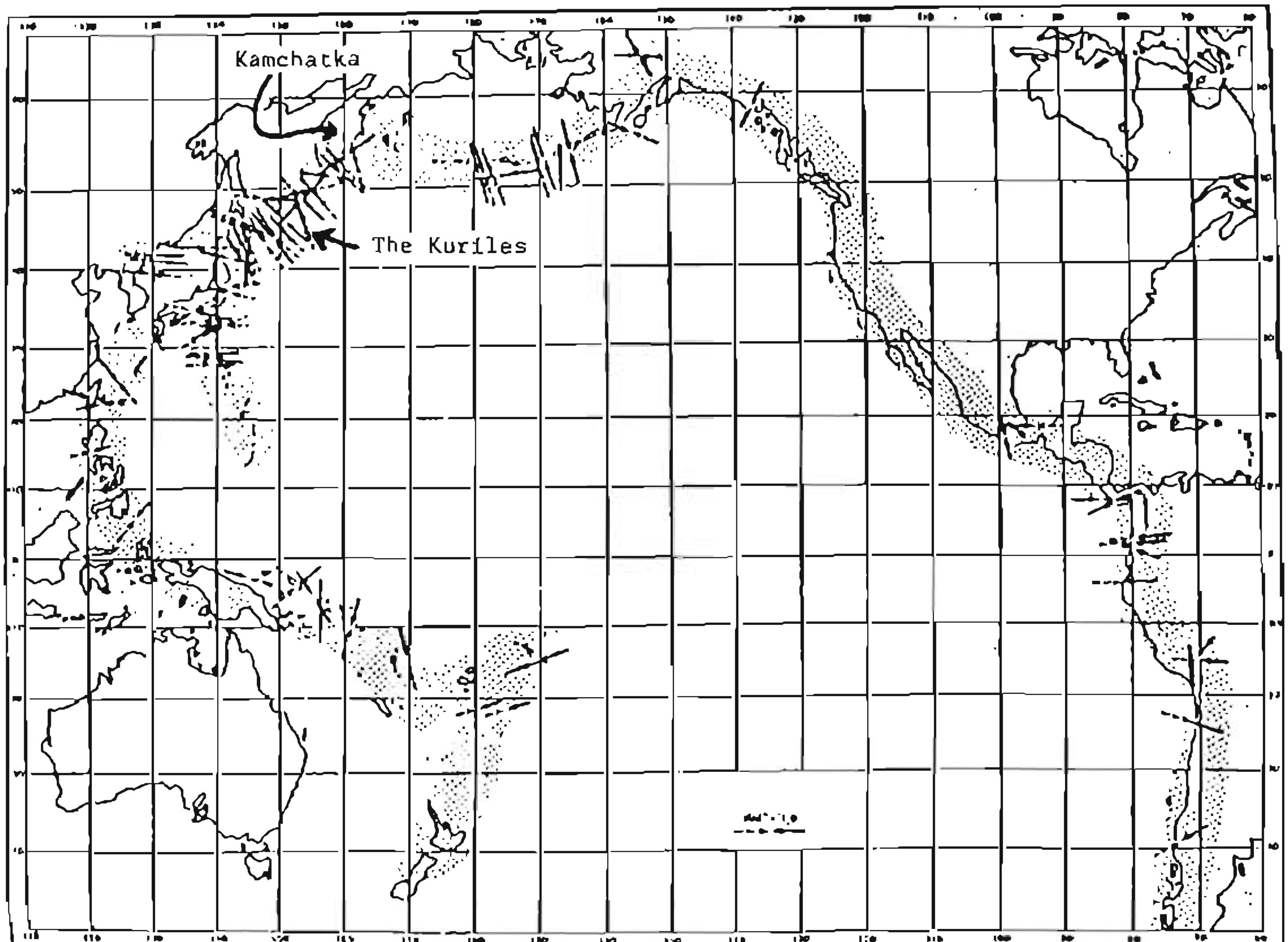

Figure 2. Stress vectors of Circum-Pacific Benioff zones.

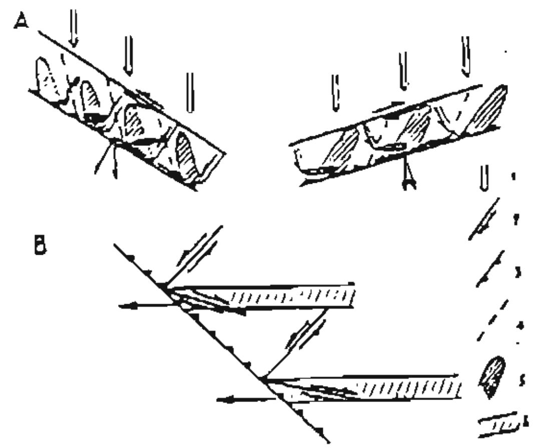

Figure 3. Typical orientation of forces along the Benioff zone.

strike of the geotectonic system as a whole. The Aleutian arc is an example of an arc with more than one direction of horizontal displacement. The eastern branch of the Aleutian arc and its corresponding shelf block generally display right-lateral strike-slip offset, whereas the western branch generally displays left-lateral offset. Other differences in the two branches are also observed. The western branch of this arc (from Buldir Island to Kamchatka) is characterized by the gradual decrease of recent volcanic activity in the southwestward direction, and northeast-striking structural features are superimposed on older, northwest-striking structures in the Komandor Islands. In contrast, in the eastern branch of the arc, recent volcanism increases toward the northeast, where it becomes abundant in Alaska.

The development of horizontal displacement along an island arc may change along strike with time. Some features characteristic of a system may move with time, and additional rigid blocks can become involved in active island-arc development. Examples of such involvement can be seen in rigid blocks of western Canada and the Antarctic, where Andean Neogene volcanic belts occur along the continuation of the island arc.

When discussing Benioff zones, note that the outer boundary does not coincide with the oceanic trench but occurs between the trench and the geanticline of the outer arc. Thus, the oceanic trench cannot be considered a reflection of subduction or Benioff-zone processes. Further, there may be no connection between the formation of oceanic trenches and the Benioff zone. The planes of displacements within certain earthquake foci may not be parallel to the Benioff-zone boundary, but may have a substantial vertical component. These considerations suggest that Benioff zones may be features considered as a type of intensely compressed block between two linear strike-slip faults rather than as a subduction zone. Increased depth of the earthquake foci toward the inner part of geotectonic system reflects an increased depth of compression toward the trailing part of the geotectonic system. Considering the absence of direct information on the nature of earthquakes within Benioff zones, it is preferable to use the term 'Benioff zone' to denote plates with increasing depth of earthquakes rather than genetic terms such as 'subduction zone.'

ISLAND GEODYNAMICS

The absence of a connection between Benioff zones and oceanic troughs raises important questions about geodynamic conditions within island-arc geotectonic systems. The structural concepts presented here indicate that the generally accepted interpretation that island arcs as a whole are a zone of compression is not justified. In contrast, island arcs do commonly display a series of linear grabens, which indicates extension. The grabens are represented by paired oceanic trenches and volcanic belts that are typical of extensional tectonics. The overall geodynamic condition in oceanic troughs can therefore be characterized as extensional. This interpretation resolves the contradictions between 'subduction' theories and extensional geodynamic environments along the continuation of deep-water (oceanic) trenches. An example of such is the graben of the Bay of California, which is a continuation of the Central American trench. Similar examples can be observed in the continuation of the New Britain oceanic trench into the narrow linear depression filled with a very thick (about 10,000 m) Tertiary sedimentary sequence, stretching along the northern coast of New Guinea, and in the continuation along the axis of the Izu-Bonin oceanic trench into the Kwanto basin in central Honshu, which is also filled with an extremely thick (8,000-10,000 m) Neogenic sedimentary sequence.

The extensional nature of island-arc volcanic belts becomes even more obvious within uplifted blocks that typically interrupt island-arc belts. These features include the linear grabens and graben-synclinal structures such as the Taupo graben in New Zealand's North Island, the Semangko graben along the Sumatro, and the graben-synclinal systems in Kamchatka. The size of these structures and intensity and character of their movements can be compared with widely recognized zones of extension within midocean ridges and intracontinent grabens.

Zones of compression within island arcs are concentrated within narrow areas that coincide with Benioff-zones' intersection with the surface, which occurs between the oceanic trenches and the tectonic geanticline of the outer arc.

CHARACTERISTIC FEATURES OF KAMCHATKA-TYPE GEOTECTONIC SYSTEMS

Geotectonic systems developed within elevated blocks that occur within belts of normal island arcs and have typical island-arc geodynamic characteristics (namely, oceanic trench, belts of negative-gravity isostatic anomalies, a Benioff seismic zone, and active recent volcanism) are considered to be early orogens or Kamchatka-type geotectonic systems (Erlich and Gorshkov, 1979). These systems include the eastern and central blocks of Kamchatka, the Alaska Peninsula, northeastern Japan, southwestern Japan, Sumatra, and the North Island of New Zealand. These Kamchatka-type tectonic systems have an average pre-Pleiocene erosional tectonic relief of 1,000 m elevation and extend up to 2,000 m elevation. In comparison, normal island-arc geotectonic systems have a pre-Pliocene tectonic relief close to sea level and do not exceed 500 m elevation. Kamchatka-type systems are developed on tectonically heterogenic basement and are characterized by continental crust with a thick and well-developed granitic layer. Within the outer geanticlinal belt, the formations within these systems may differ in age. Examples of Upper Cretaceous to Paleogenic formations occur in Kamchatka, whereas Paleozoic formations occur in northeastern Japan. Jurassic to Paleogene formations occur in the southwestern Japan (the Shiamanto complex), Precambrian complexes in the Philippines, and Jurassic graywackes in the North Island of New Zealand. A number of regions evince Neogene development preceded by Upper Cretaceous to Paleogene geosynclinal development, spilite-keratophyric volcanic complexes, and hyperbasitic intrusions.

The Neogene structure of these geotectonic systems is characterized by structures typical of normal island arcs, including a linear geanticlinal uplift surrounded by two linear troughs filled by very thick (8,000-10,000 m) sedimentary, mainly terrigenic sequences. Volcanic rocks in these depressions are rare or absent. The environment of sedimentation in these troughs gradually changes upward from abyssal sediments in the lower part of sequences to shallow-water or even subcontinental sediments in their uppermost parts. Thus, the Neogenic troughs located in the frontal part of the geotectonic systems are probably equivalents of filled oceanic trenches, and troughs located in the trailing part are equivalents of marginal sea-type structures completely filled by terrigenic sediments. Relicts of such linear depressions are often located in the frontal geanticlinal belts of Kamchatka-type geotectonic systems, including the eastern trough of Kamchatka, the northern trough of New Guinea, the trough exposed within the chain of small islands stretched along Sumatra's southeast shore, and the Neogene trough on the frontal southeastern shore of North Island, New Zealand. This interpretation is supported by the transition of oceanic trenches along the strike into linear depressions filled by sedimentary terrigenic Neogene deposits. Thus, the New Britain oceanic trench is continued along strike by the northern depression of New Guinea, and the Izu-Bonin deep-water trench is continued along strike by the Kwanto basin in the central Honshu.

Characteristic of the recent structure of Kamchatka-type geotectonic systems is the presence of a series of linear anticlinal belts that contain dislocated pre-Pliocene basement elevated to an average height of 1,000 to 1,500 m interrupted by linear zones of grabens and graben-synclinal structures and Pre-Pliocene basement within latter structures is subsided and filled by volcanic, volcaniclastic, and unconsolidated Quaternary sediments. These grabens and graben-synclines control the structural position of most of the Pliocene-Quaternary volcanic belts within Kamchatka-type geotectonic systems.

Quaternary graben-synclines have not been reported in Alaska, but distribution of the roof height of dislocated pre-Pliocene complexes indicates that such linear structures are here. Within the Katmai region, a linear belt of volcanoes extends northeast from the Kejulik Mountains to Kamushik Bay. This linear structure is 25 to 30 km wide and contains dislocated pre-Pliocene complexes that are normally subsided below erosional levels and are exposed only within small elevated blocks, such as Falling Mountain near the

Katmai volcano. This structure is characterized by an intense negative-gravity anomaly (Kienle, 1969). A similar structural position of Quaternary volcanic belts can be seen in the Ugashik region. Considering the distribution of both Pliocene-Quaternary volcanics and active volcanoes in Alaska, a second volcanic belt in the Naknek Lake-Iliamna Lake region may parallel the major volcanic belt.

Oceanic trenches near Kamchatka-type geotectonic systems change their morphological characteristics. If their 'telescopic' shape decreases their depth, their floor is flattened and widened. These changes are equally characteristic for parts of the Kurile-Kamchatka trench along Kamchatka, part of the Kermadek trench (Hikurangi depression) that extends along the North Island of New Zealand, and the continuation of the Ryukyu trench along the Kyushu Island and southwestern Japan. Strong indications exist that all these parts of oceanic trenches, including the part of the Aleutian trench stretching along the Alaska Peninsula, are very young (Minato and others, 1965, Erlich and Gorshkov, 1979; and Drews and others, 1961). These specific features, along with the previously mentioned presence of a leading linear trough filled by Tertiary sedimentary sequences, suggest that the leading edge of these geotectonic systems underwent accelerated displacement oceanward during the Pliocene to Quaternary periods.

The trailing structural element of the island-arc geotectonic system (notably the deep-water basin of the marginal seas) also disappears near Kamchatka-type geotectonic systems. The south Okhotsk deep-water basin, located in the rear part of the Kurile island arc along the strike, is replaced by the western Kamchatka linear rear trough and is filled completely by an 8,000 to 10,000 m terrigenic sequence of Tertiary rocks.

Accelerated displacement of the leading edge of Kamchatka-type geotectonic systems is clearly expressed in recent structures. In the southern Kamchatka, horizontal displacement along strike-slip faults is common and reaches up to 15 to 25 km. Extensive active centers of Quaternary volcanism occur along the continuation of these faults (Erlich and Gorshkov, 1979). Within the eastern Kamchatka similar strike-slip faults are expressed in narrow bands of earthquake foci with depths ranging from 50 to 100 km. These bands crosscut large caldera complexes and display large, circular negative-gravity anomalies that are considered to be a reflection of shallow chambers of silicic magmas (Luchitsky, 1974). In Alaska this type of strike-slip fault may exist within the Katmai region (Keller and Reiser, 1959).

A major difference between recent Alaskan structures and the double arc of the eastern Aleutians is that the distance between the oceanic trench and the geanticline of the outer arc grows abruptly in the eastern Aleutians. Also, a linear topographic depression occurs between the geanticline of the outer arc and the belt of Neogene volcanoes expressed by the region from Shelikov Strait to Cook Inlet. In the northern part of this depression are the volcanic groups of the Wrangell and Talkeetna Mountains. The structural position and tectonic nature of this depression suggest similarities with the central Kamchatka depression in the Kamchatka Peninsula, where the large Kliuchevskaya and Sheveluch volcanic groups are located.

The most intensive compressive dislocations are concentrated in the outer geanticlinal belt from Kodiak Island to the Kenai Peninsula. In this area the widening of the geotectonic system can be explained by accelerated displacement of corresponding blocks along east-west strike-slip faults. Traces of these faults can be found in the east-west relief scarps that crosscut the Alaska Peninsula.

The development of the island arc belt as a whole is characterized by the existence of short (to tens of thousand years) pulses of intensification of geological processes. These pulses are reflected in simultaneous intensification of volcanic and tectonic processes. The intensity of these pulses varies for the different parts of the belt and is slower in the normal island-arc systems and more intensive in Kamchatka-type geotectonic systems. Such simultaneous intensification of silicic volcanism and tectonic uplift is directly reflected in accumulations of coarse sedimentary sequences in the zones of subsidence, as in the central Kamchatka depression (Erlich and Gorschkov, 1979). These pulses of intensification are characterized by intensive silicic volcanism, particularly in the Circum-Pacific region (Erlich and Gorshkov, 1979).

CHANGES IN THE RATE AND CHARACTER OF VOLCANISM IN DIFFERENT TYPES OF ISLAND-ARC GEOTECTONIC SYSTEMS

Changes of structural position of volcanic belts in Kamchatka-type geotectonic systems are accompanied by sharp changes in the rate of magma chemistry and production.

1. The rate of magma production is reflected by the volume of volcanic material in features. The frequency and size of stratocone and shield cones for the Kamchatka-type systems and normal island arcs are shown in figure 4 (for the Kuriles).

2. The size and intensity of magma-chamber formation are reflected in the diameter of calderas formed during the same interval. Normal island arcs are characterized by average caldera diameters of 2 to 4 km, although the diameter of their largest calderas reach 8 to 12 km. In the Kamchatka-type geotectonic systems the average diameter of calderas equals 8 to 12 km; the largest such structures reach 40 to 60 km in diameter.

3. Rate of magma evolution. For normal island arcs (for example, the Kuriles) during the Quaternary time, one volcanic cycle is characteristic: formation of a basaltic/andesitic plateau and continuing through andesitic stratovolcanic formation to the formation of silicic domes, pumices, and ignimbrites. In the Quaternary development of Kamchatka-type geotectonic systems, two similar volcanic cycles developed. The first cycle occurred in Upper Pliocene to Lower Pleistocene time and ended from the end of the Middle Pleistocene to the beginning of Upper Pleistocene time. The second cycle occurred in the end of the Upper Pleistocene to the beginning of the Holocene time.

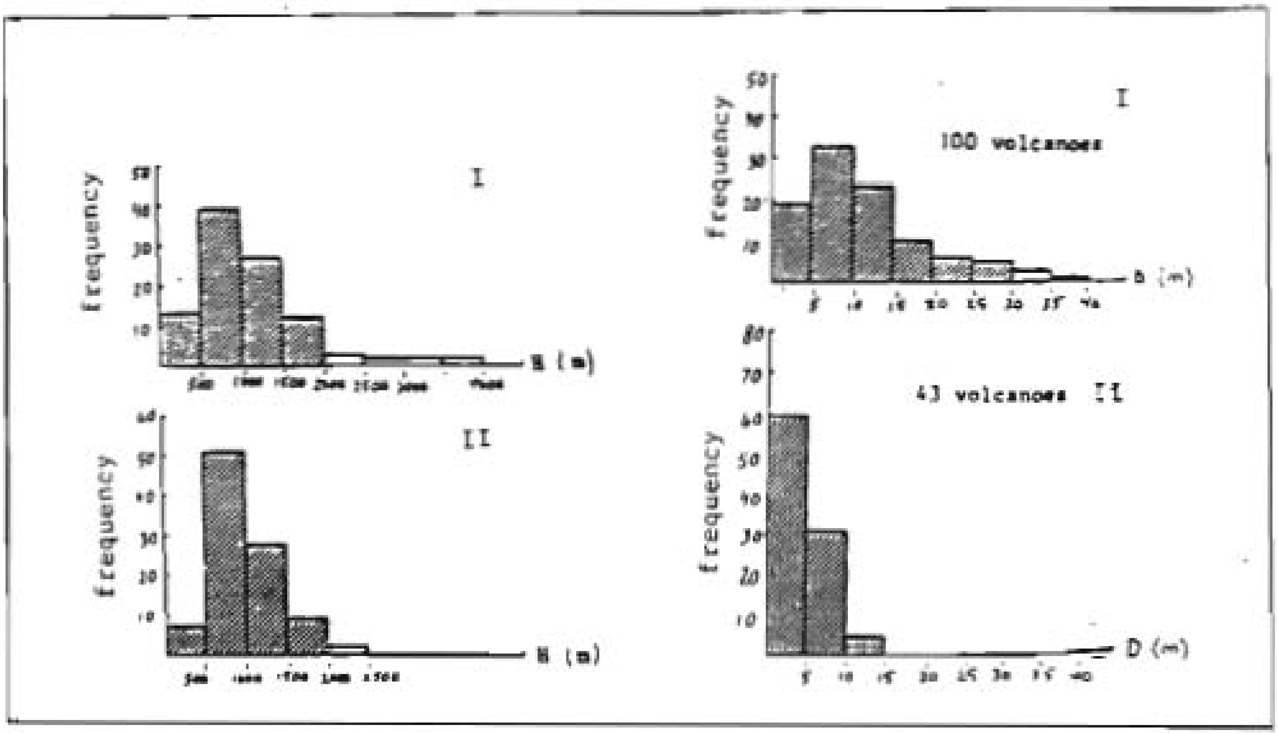

Figure 4. Frequency of diameters (D) and relative heights (H) in meters of Quaternary volcanic structures. I - stratovolcanoes and shield volcanoes of Kamchatka-type structures, II - stratovolcanoes and shield volcanoes of normal (Kurile) island-arc structures.

The rate of magma production decreased abruptly when the amplitude of recent uplift of geotectonic system reached a critical height. When the height of the erosional tectonic watershed relief reached about 2,000 m above sea level, the rate of magma production decreased abruptly; volcanism decreased dramatically and occurred only as small separate basaltic volcanoes (such as the Prindle volcano in the Tanacross area or basaltic lava fields in Seward Peninsula grabens).

Quantitative changes are accompanied by changes in the character of the volcanic processes. The latter changes in turn are reflected by changes in the chemical types of basalts, the quantity of basalts, the maximum silica content, and the behavior of volatiles in magma; these are in turn reflected in the ability to vesiculate and, consequently, in the types of calderas developed within different types of geotectonic systems. For example, Hawaiian-type calderas and complex basaltic volcanoes (Heiken, 1976) are rare to absent in normal island arcs but widespread within Kamchatka-type geotectonic systems. Detterman and others (1981) suggest the existence of this type of complex of basaltic volcanic structure around the Veniaminoff volcano in the Chignik region (fig. 5). Another type of caldera in the Kamchatka-type geotectonic system that is connected with poorly vesiculated silicic magma is represented by the Khangar and Alney-Chashokondzha calderas in Kamchatka (Erlich, 1986). This type caldera may also occur in Alaska---the Kaguyak and Aniakchak craters (fig. 6). These calderas characteristically display an absence of great amount of silicic pyroclastic material connected with

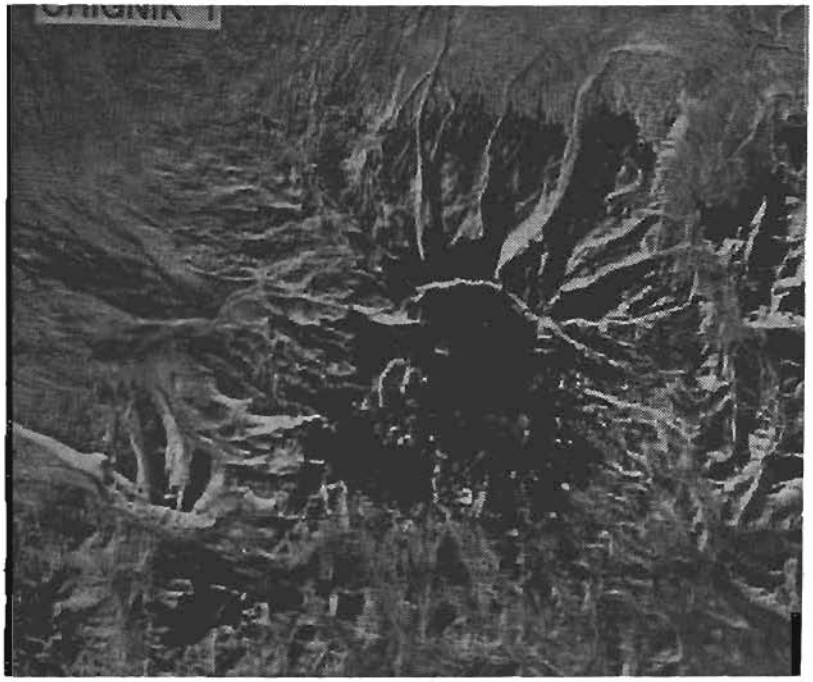

Figure 5. The Veniaminoff volcano, Chignik region, Alaska.

Figure 6. The Kaguyak and Aniakchak craters, Alaska.

caldera-forming eruptions (Detterman and others, 1982). Calderas of this type may have been formed by slow subsidence of the caldera floor during crystallization of magma chambers beneath the calderas.

Chemical composition of basalts and the frequency of these various types of chemistries also differ between different geotectonic systems. Intraoceanic island arcs are characterized by essentially tholeiitic basaltic volcanism with minor dacitic pumices. In normal island arcs, andesites prevail, with minor amounts of high-alumina basalts. Volcanic rocks with a silica content greater than 65 percent usually do not occur in normal island arcs. Kamchatka-type geotectonic systems characteristically contain a bimodal distribution of volcanic rocks, including basaltic volcanism (represented by shieldlike stratovolcanoes), basaltic stratovolcanoes, or great areas covered by basalts connected with numerous small icelandic-type shield volcanoes and cinder cones, as well as dacitic volcanism and ignimbrites. Complexes of extrusive domes composed of dacites and rhyolites may also occur.

IMPLICATIONS FOR PLACER POTENTIAL

The model presented suggests that the areas of the most intensive Quaternary uplift (the Aleutian Range, Kodiak Island, and the Kenai Peninsula) encountered conditions favorable for the generation of hydrothermal activity from possible magma chambers or an increase in the regional heat flow. Areas that could be favorable for placer deposits would also require intense accumulations of loose material during the Quaternary. One area that meets these criteria is the linear depression between the Aleutian Range and the region from Kodiak Island to the Kenai Peninsula. The possible existence of specific time intervals especially favorable for magmatic or hydrothermal processes in this region permit it to be a possible target for either marine or alluvial buried placer deposits.

In this region, the potential for placer deposits include the areas near the boundaries of actively uplifted zones (Shelikov Strait, Cook Inlet, Bristol Bay shore) and areas around volcanic groups (foothills of the Wrangell and Talkeetna Mountains).

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GEL-LOG POLYMER TREATMENT AND FLOCCULATION IN SETTLING PONDS

by

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A gel-log polymer treatment for settling-pond flocculation known as 'log in the stream' has been studied for about 10 yr, particularly in coal-mining operations in Pennsylvania.

THE GEL LOGS

The gel-polymer logs are contained in a plastic mesh netting and are about 9 in. in diameter and 18 in. wide. These logs normally weigh about 40 lb. After being placed in water, the logs will absorb 1 gal water in 2 days. The logs continue to swell for 3 to 5 days, until the dissolution process begins. Eventually the product is totally dissolved. The logs are used in a variety of temperatures; however, the viscosity of the product fluctuates with water temperature.

To illustrate the usefulness of this product, one settling pond of a coal mine in Pennsylvania yielded 5,000 tons of ash build up every two months. After gel-log treatment in the hydrobins of the plant, the same settling pond yielded 500 tons of sediment build up after 6 months. The gel logs extended the life of the pond, helped meet effluent standards, and reduced the material handling costs (ash removal).

To determine the applicability of gel logs for a particular situation, samples are collected and tested in a jar with a 3/8-in. cube of gel polymer. Settling rates can be obtained from these simple tests. Instructions for these tests are given in Appendix A. Factors that must also be considered in these analyses include temperature, type of solids, and abrasiveness of the surface contacts.

Advantages of the gel-log system of flocculation treatment are:

1) The gel logs are easy to handle and require no equipment investment, electrical requirements, or water connections.

2) Dosage rates can be easily adjusted by the adding or removing of a gel log.

3) The system can function unattended for several days and automatically respond to variations in flow rates (the more water that flows over the log, the more material goes into the system).

4) It is easy to determine when replenishment is needed (visual inspection).

5) The gel-log polymers can be effective at low dosages.

A recent study prepared by the U.S. Bureau of Mines in Pittsburg concluded that, because of the slow release of these types of polymers, they were allowed to completely recoil. This slow release allowed these polymers to be effective in dosages of 200 ppb, whereas other products required much larger effective dosages (up to 3-4 ppm).

One way of using the gel-log polymers is to put the logs in a flume or sluice that feeds into a settling pond. Some operators put the logs in a cross-sectioned 55-gal drum. The channel that feeds into the settling pond should be lined with plastic to ensure that the material does get into the pond.

APPENDIX A

Detailed Instructions for Polymer Selection

Neutron's family of Photafloc gels comprises a relatively complete catalogue of products, including neutral polymers, two series of anionic copolymers and two series of cationic copolymers. Within each series, the degree of ionic functionality covers the entire range from close to neutral to very high charge density. Our standard test kit contains eight diverse samples, at least one of which is likely to work in almost any situation. The evaluation procedure is simple, and no expensive equipment is required.

Required equipment comprises a watch with a second hand, a two gallon container filled with a representative sample, a stirring rod, a ladle, a thermometer, and a set of clear, 16 ounce screw cap jars with a wide mouth.

Fill one sample jar with a representative sample of water or slurry to be treated, allowing about 20% headspace. Set aside to use as a blank for comparison with the test samples.

Fill one or more additional test jars with equally representative samples, filling all jars to about the same level.

Add one pre-cut gel cube to each test jar and tightly cap. Shake each test jar in turn with moderately vigorous motion for the same short period of time. Shake for approximately 10 seconds to start. Carefully watch for floc formation and settling. Note floc size, relative settling rate, and stability. For best comparisons strive for uniform treatment of all samples, running each test to completion, one at a time.

Water temperature and quality may effect polymer dissolution rates, and some fragile flocs may not withstand prolonged or violent shaking. Thus, some experimentation may be required to optimize this procedure to individual needs.

The first rule of testing is to allow ample time. You may be lucky, but a thorough screening and evaluation may the better part of a day. In some cases, it is possible that considerations of relative cost, relative fish toxicity, or other factors may result in the choice a polymer other than the one that appears to provide the best performance. Accordingly, it is important to recognize that more than one polymer will probably work, and that it is unwise to settle on the first polymer that performs.

For the more sophisticated operator, best results are sometimes obtained by sequential treatment with cationic and anionic products (always in a sequence and never pre-mixed). To test for this effect, pour the treated contents of one jar into a clean test jar and add the other polymer. Do not put anionic and cationic cubes into the same jar.

Your field service representative or Neutron Products' technical personnel can assist in the evaluation of the data and help select optimal product and treatment levels.

NEUTRON PRODUCTS inc

HEAVY-EQUIPMENT MAINTENANCE MANAGEMENT

by

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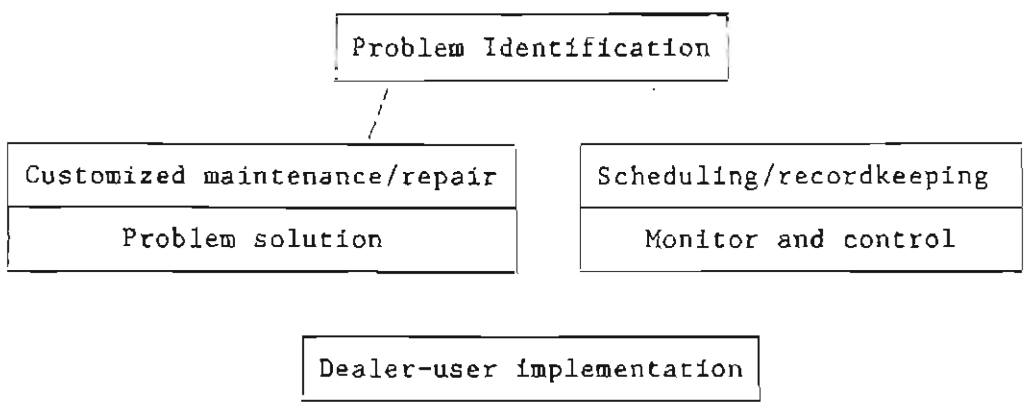
The most important thing to remember about heavy-equipment maintenance is that you have to care about your equipment. The biggest cost involved with heavy equipment is not the purchase price, but the cost of maintaining your heavy equipment.

There are several things you and your dealer can do to set up a maintenance schedule to ensure that you will get the most value for your money. However, it is up to you to have an active interest in your equipment's management. The following diagram shows the steps involved in setting up a customized heavy-equipment maintenance plan.

EQUIPMENT ANALYSIS



Maintenance management allows you to do what you want with your equipment. Careful recordkeeping is the followup on your maintenance monitoring. Careful monitoring of the condition of your equipment can allow your problems to be planned problems, not unplanned ones which can cost you even more time and money. To illustrate an example of economic planned repair and maintenance, consider the equipment owner who knows that his undercarriage might need to be replaced soon. He then lets the supplier know what he might need and the parts can be there when he needs them. This insures that the owner will have a minimum of down time and he will not have to pay for rush shipping costs.

Repair indicators that you need to be aware of include: operational performance, gauges, visual clues, and sound indicators. The Caterpillar Company can help you with the following programs:

SOS - Scheduled oil sampling. The dealer will look at the wear and condition of oil and tell you what is happening or what could happen to your equipment.

Filter Check - Check all filters.

CTS - Customized Track Service. Allows you to plan for your needs and preorder.

Technical Analysis - You can get a report on condition of your equipment and what might be prone to failure.

Caterpillar also offers the following booklets to help you understand your equipment: "Cold-weather operations," "Know your cooling system," "It's not worth the risk," and "Oil in your engine."

In short, proper equipment maintenance will ensure that you get the best value for your money. And if you don't maintain it, you not only won't be pleased with it, but your operational costs will be higher.

NEW DIRECTIONS IN THE DEVELOPMENT OF THE ALASKA MINERAL INDUSTRY FOR THE SMALL MINER

by

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Placer gold has been the most successful mineral commodity mined by small-scale operations in Alaska and it's a little presumptuous to suggest to many small-scale placer gold miners that there are better opportunities to explore. Nevertheless, this paper will discuss some alternatives to conventional placer-gold mining.

DRIFT MINING

Drift mining employs underground mining methods to extract frozen placer deposits, usually beneath thick layers of overburden. Hundreds of drifts were operated years ago but only a few exist today. Examples of currently active underground mining operations are the Wilbur Creek operation in the Livengood district, the Wild River Ventures operation near Wild Lake in the Brooks Range, the Sherrer operation in the Innoko district, and several prospect drifts in the Fairbanks area. In the past, these areas operated in conventional surface placer mines. However, problems with the removal of thick overburden in a manner that can satisfy the current water-quality standards have plagued mine operations. The Wilbur Creek mine recently tried to resolve this problem by using drift-mining techniques in the buried placer paystreaks. The project was funded through the Alaska Placer Demonstration Project. During 1986-87, the mine produced 200 yd/day at a cost of $19.87/yd^3 and the mine was efficiently operated. Whether this venture is economical cannot be determined at this time. However, a number of deeply buried placers deposits exist throughout the Interior and western regions of the state and the techniques learned at Wilbur Creek may benefit individuals active in these areas.

Rare-earth Elements

There may be a future in the extraction of so-called high-technology minerals such as elements niobium and tantalum. Super-conductor, space, and military technologies continue to find new uses for such commodities. The first documentation of an Alaskan niobium resource has recently been published (Warner and others, 1986) and the U.S. Bureau of Mines is currently working on a promising niobium - rare earth element deposit at Bokan Mountain near Ketchikan. Although the spot market for tantalum ($20/lb) and niobium ($6/lb) is depressed today, they were as high as $108/lb and $24/lb, respectively, as recently as 1982. Because current and projected demand of these commodities is relatively small in terms of total volume, small-scale producers have and can compete with larger companies, provided market opportunities are secured.

COPPER

Copper mining has been successful in south-central and in southeastern Alaska. Before 1930, there were 30 copper mines in the Prince William Sound region and another 25 mines near Ketchikan from 1897 to 1938. Most of these operations were of a small scale---employing 50 or fewer individuals. After the Kennecott Mines in the Chitina valley closed in 1938, most production ceased. However, selected shipments of chalcocite ores from the district were intermittently shipped by aircraft by a small operator from Chitina until 1970. The price of copper has been severely depressed, but appears to be moving up at the time of this writing.

COAL

Although current coal-mining activities are confined to a single large-scale operation at Usibelli, coal has been mined by the small miner. Paul Omlin successfully operated the Priemer Mine in the Mantanuska field with three individuals and supplied home heating fuels in the Wasilla and Willow areas.

PLATINUM

Alaska has been the largest producer of platinum metals in the U.S., with most platinum production derived from the Goodnews Bay placer district. Production through 1986 has amounted to 668,000 oz unrefined platinum-group elements. The metallogeny of this important strategic metal is still poorly understood. Recent work by the U.S. Bureau of Mines (Barker and others, 1985; Southworth and Foley, 1986) has produced some interesting new resources of platinum not only in the Goodnews Bay area but also in the Alaska Range and the Chugach Mountains. In the Alaska Range, the U.S. Bureau of Mines has been examining small plugs of dunite and diorites originally known to be associated with nickel deposits. Anomalous platinum has consistently been encountered in massive-sulfide samples, and the deposits appear to be similar to the Wellgreen property in the Yukon Territory, a nickel prospect now being developed as a platinum property. Because most platinum has been found almost by accident in Alaska, the chance of additional discoveries remain high. Terranes with mafic ultramafic and alkaline igneous complexes throughout the state deserve to be evaluated for both placer and lode platinum.

ANTIMONY

In 1986, Alaska was the only state in the union that produced and shipped antimony concentrates (about 24 tons). Antimony has fluctuated wildly over the years. Today it can range from $.50 to $9/lb on the metals market, depending on demand. There are times when antimony supplies are so scarce that a producer could obtain a premium spot market price, which fluctuates because the U.S. imports much of its antimony from sources that are politically unstable.

Antimony ore from at least 18 deposits were shipped to market in 42 of the last 100 yr (including the last four). Most production has been confined to the Seward Peninsula and Interior, but antimony has also been extracted from the Southeastern Panhandle to the Brooks Range. The largest of the Interior operations was the Scrafford Mine off Old Murphy Dome Road in the Fairbanks area. The largest antimony mine in Alaska was the Stampede Mine in the Kantishna district, which produced ore from the 1930s to the 1970s.

Antimony mining is suitable for small-scale mining operations because deposits commonly occur in high-grade concentrations (30 to 60 percent antimony). Therefore, the material can easily be concentrated and stockpiled until the price is high.

Companies in Korea and Japan that have expressed an interest in purchasing Alaskan stibnite (antimony disulfide). In 1984, when the price of antimony was particularly high, several companies in Belgium and West Germany were also interested in Alaska as a potential supplier of antimony. HCA International of Sarasota, Florida, has recently attempted to obtain stibnite concentrates from Alaska.

MERCURY

Most of the mercury produced in Alaska has been derived from southwestern Alaska. Mercury prices fluctuate greatly and have ranged from \$180 to \$1,000/per flask (76 lb). Mercury has been successfully mined from over a dozen deposits in the Kuskokwim region which produced about 40,000 flasks from 1923-74. About two-thirds of this production came from the Red Devil Mine. Virtually all these mines have been small-scale operations.

The price of mercury is now low; many of its major industrial applications (herbicides, fungicides) have been abandoned because it is a toxic substance. However, it continues to be used in scientific control instrumentation and medical and electrical applications. In addition, crystalline cinnibar is sold in Japan for medicinal purposes at premium prices. Oriental purchasers continue to contact the Alaska Division of Geological and Geophysical Surveys for potential suppliers of this commodity---without too much luck, I'm afraid.

TIN

Alaska's most successful tin mines have been in the Lost River region of the Seward Penninsula and the Tofty area near Manley. At Tofty, the tin-ore mineral cassiterite is mined as a byproduct of gold mining; tin is the primary commercial product in the Lost River region. In 1986 Alaskan tin-mining operations approached historical production highs. Other areas where tin has been produced include the Ruby-Poorman district and the Old Crow area of east-central Alaska.

All Alaskan tin production has been derived from relatively small-scale mines. Because cassiterite contains nearly 80 percent tin, successful concentration can allow for shipment to market from remote areas--often by aircraft. Until recently, the price of tin has been more stable than many of the other strategic minerals, mainly because its price was controlled by the World Tin Council. Tin reached a peak price of \$10/lb in the late 1970s but the council was not able to control the price when excess amounts of tin entered the market. Three years ago the price of tin collapsed to $2.50/lb. It is now slightly over $4/lb.

TUNGSTEN

Tungsten has been produced in the Flat, Nome, Hyder, and Fairbanks districts. The principal tungsten-bearing mineral is scheelite, or calcium tungstate, a mineral that fluorescences bright blue in ultraviolet light.

Most production has been from lode sources, but placer scheelite has been shipped from the Iditarod district since the 1950s. The price of tungsten has fluctuated greatly since World War I. Although prices have reached $10/lb, today's depressed market (about $2.5/lb---if you can find a buyer) has discouraged most Alaskan tungsten miners, at least in the short term.

OTHER STRATEGIC MINERALS

Other strategic minerals that have been produced in small quantities in Alaska include graphite on the Seward Peninsula, chromite near Seldovia, and uranium near Ketchikan. Most operations produced as a result of government subsidies during times of critical need but have not been exploited after price supports were removed.

INDUSTRIAL MINERALS

A variety of industrial minerals, including a variety of building stone, sand and gravel, gypsum, barite, clay, asbestos, diatomaceous earth, and garnet, have been exploited in Alaska over the years (Bundzten and others, 1982). The market and transportation access have traditionally been the factors that have determined economic viability. Most of these commodities are low-unit-value ores and have been extracted and marketed locally at scales ranging from quite small to large.

Sand and gravel and building stone have been profitably mined in Alaska by small companies, especially since post-World War II construction and the North Slope oil-field development. However, the demand for sand and gravel in Alaska has been decreasing, and the 1986 levels of production dropped nearly 30 percent from the previous year.

The possibilities of exporting industrial-mineral products to Pacific Rim nations should not be ignored. As an example, 24 of 39 commodities imported by the Republic of Korea during the years 1981-85 are nonmetallic industrial minerals (table 1). Much of the total $1.82 billion worth imported by that country were nonmetallic commodities; this total excludes the value of energy products. Japan and Taiwan import even larger volumes of these commodities.

The key to the development of Alaska's industrial minerals for export will include 1) acquisition of a better data base in the state (very little is known about the distribution of these commodities), 2) thorough researching of transportation and other access provisions, 3) determination of specific physical properties needed for specific markets, 4) providing market

Table 1. Mineral imports in volume and value for the Republic of Korea, 1981-85 (from Alaska Office of Mineral Development, 1986).

Mineral	Imports (metric ton)				Value (in thousands of dollars)			
	1982	1983	1984	1985	1982	1983	1984	1985
Gold (kg)	1,221	1,785	1,954	9,603	14,337	18,148	22,444	16,224
Silver (kg)	1,679	5,659	6,697	14,713	369	1,665	1,279	1,127
Copper	394,250	371,278	347,665	355,210	161,847	172,013	150,708	143,963
Lead	-	-	-	10,871	-	-	-	3,846
Zinc	79,176	114,399	103,172	162,946	20,433	23.147	28.752	37,489
Iron	11,464,397	10,121,155	10,287,389	12,418,239	277,802	253,876	282,684	318,884
Manganese	232,678	215,631	249,992	271,529	15,294	17,990	14,565	16,078
Molybdenum	-	-	179	115	-	-	673	640
Aluminum	5,334	6,095	9,305	9,177	868	1,065	1,587	1,613
Tin	17	574	1,749	2,687	146	5,628	15,525	20,933
Chromium	4,846	3,853	5,027	3,875	964	798	1,039	809
Titanium	33,455	37,699	44,071	39,569	3,738	3,644	3,761	3.953
Zirconium	2,370	2,514	7,097	5,928	551	558	1,486	1,163
Antimony	391	996	1,250	883	339	644	1,511	1,215
Crystalline graphite	414	440	1,250	1,078	624	696	1,067	1,091
Amorphous graphite	39	3	4	6	41	6	6	23
Talc	12,486	4,243	20,981	23,192	1,411	1,247	2,884	2,817
Feldspar	39	13	37	804	3	1	3	101
Kaoline	129,624	36,258	36,032	41,979	7,533	9,283	8,950	10,302
Marble	73	369	587	972	31	133	310	440
Dolomite	9	653	155	224	5	124	27	36
Limestone	4,601	5,000	10,512	317	58	63	113	39
Silica stone	43	307	613	219	27	118	252	58
Silica sand	102,386	142,632	122,966	140,431	3,670	4,891	3,616	4,205
Diatomite	4	15	-	11	6	9	-	4
Asbestos	44,038	11,305	59,693	57,143	17,914	22,141	21,460	18,150
Fluorite	18,352	19,200	29,947	46,093	1,505	1,568	2,421	3,916
Phosphate rock	1,505,405	1,591,886	1,651,629	1,770,968	103,393	84,052	77,165	79,749
Mica	180	434	569	962	258	386	435	542
Sulfur	440,660	448,076	567,836	592,867	54,691	44,903	56,940	78,623
Barite	350	469	949	1,113	45	64	183	216
Andalusite	628	719	998	1,980	140	135	175	328

Table I. (con.)

Mineral	Imports (metric ton) 1982	1983	1984	1985	Value (in thousands of dollars) 1982	1983	1984	1985
Magnesite	7,566	24,123	-	12,206	2,554	8.632	13,481	4,504
Gypsum	17,312	218,722	222,205	151,814	592	6,474	5,673	3,359
Bituminous coal	9,039,487	10,310,407	12,343,765	17,312,766	600,910	605,020	678,181	904,894
Anthrachite coal	2,448,589	805,208	866,457	2,658,948	164,817	49,276	47,918	137,694
Bentonite	-	2,860	3,816	5,662	-	613	789	997
Fullers earth	-	1,201	1,585	1,149	-	200	329	194
Uranium	715	364.5	785.3	-	50,327	2,850	47,574	-

Total imports value (in thousands of dollars)
1982: $1,507,243 (excluding petroleum)
1983: $1,362,061 (" ")
1984: $1,495,966 (" ")
1985: $1,820,219 (" " and uranium)

opportunities for these commodities, and 5) a change in perspective by those in industry and government concerning these commodities. The last point cannot be overstated, as industrial-mineral potential continues to be overlooked.

CONCLUSION

Geologists and mining engineers with the Alaska Division of Geological and Geophysical Surveys, the U.S. Bureau of Mines, and the U.S. Geological Survey in Fairbanks, Juneau, and Anchorage can supply geological information concerning all of the commodities summarized in this paper. Perhaps one of the most frustrating aspects of mineral development for the small miner concerns the aspect of marketing. Charlie Green and Jim Deagen (Alaska Department of Commerce and Economic Development) can provide information concerning the marketing and specifications needed by Pacific Rim importers.

The purpose of this paper is not to discourage placer-gold mining, but to discuss alternative types of mineral extraction in Alaska that could be profitable. Interested individuals are invited to contact the author.

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NORTH STAR GOLD INVESTIGATIONS

by

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INTRODUCTION

The purpose of this paper is to discuss ideas and future work on gold by the Alaska Division of Geological and Geophysical Surveys (DGGS). We refer to this part of our gold work as the North Star Gold Investigations. Future gold studies in the Interior were conceived as an outgrowth of our earlier mineral studies in central Alaska that were also requested by the mineral community, and which included cooperative efforts between the DGGS and other agencies. Like the earlier studies, the North Star Gold proposals were formulated in response to requests from the mining industry and we expect that the studies will be cooperative efforts involving the University of Alaska, the U.S. Bureau of Mines and, perhaps, additional individuals and agencies.

STRATIGRAPHIC SEQUENCE OF INTERIOR ALASKA

One of the more interesting concepts that evolved from our earlier work is that there is a repeatable stratigraphic sequence or lithologic pattern that has been documented for much of Interior Alaska, a sequence that is found in the Fairbanks district, the upper Chena River, the Kantishna district, much of the Alaska Range, near Boundary, and in the central Yukon Territory. The stratigraphic succession and lithologies in this package of rocks are illustrated in figure 1.

The bottom of the stratigraphic package consists of the Fairbanks schist which, in its upper part, contains an interstratified unit of inferred volcanic derivation known as the Cleary sequence. Above the Fairbanks schist is the Chena River sequence, which occurs in the Fairbanks district, the upper Chena River area, and possibly the Alaska Range. Overlying this unit is a thick sequence of calcphyllites, black quartzites, and black phyllites known as the Keevy Peak Formation. The Totatlanika Schist occurs above the Keevy Peak Formation, either in depositional or thrust-fault contact. In the Alaska Range, these bedrock units are capped by the coal-bearing group and the Nenana Gravels, which often occur as elevated benches and plateaus. Modern stream gravels are confined to channels and canyons incised within the bedrock units and Nenana Gravels.

Of major significance is the fact that certain parts of this stratigraphic package appear to have a characteristic metallogeny or is enriched in certain mineral commodities. The Cleary sequence is enriched in gold, arsenic, antimony, and tungsten. Parts of the Keevy Peak Formation appear to be enriched in zinc. The Totatlanika Schist appears to be enriched in precious

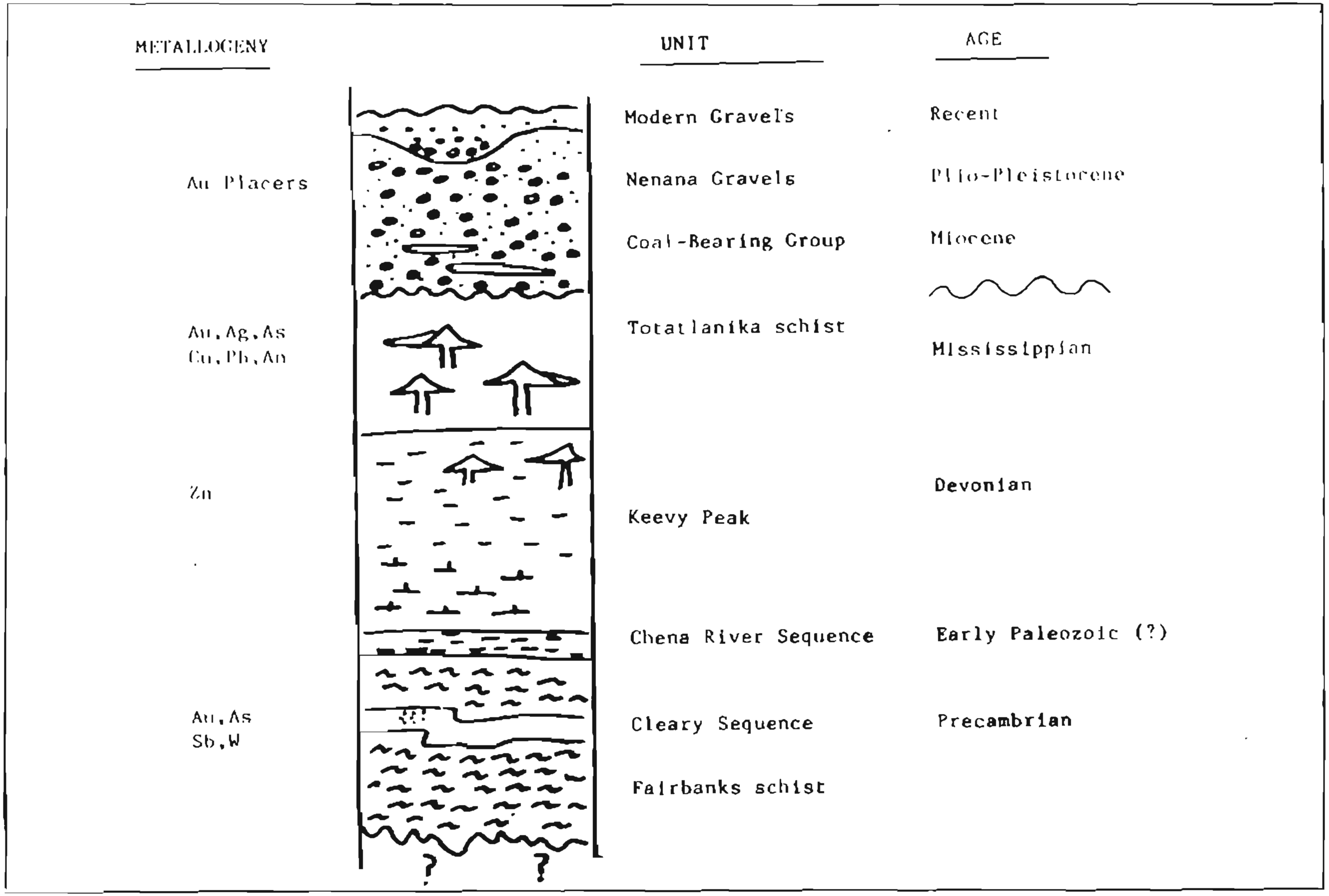

Figure 1. Generalized stratigraphic sequences, Interior Alaska (not to scale).

and base metals, and the gravels are the focus of modern-day placer mining and possibly contain paleoplacers.

Two of the objectives in our proposed investigations are:

1) To establish the genetic relationships of metals to the host-rock packages.
2) To delineate high-grade target zones within these broad regions for further studies and exploration by the private sector.

Figure 2 is a sketch map showing distribution of productive rock sequences in Interior Alaska as we presently understand them. In our interpretation, the Cleary sequence occurs in a semicontinuous northeast-trending belt from Kantishna to the Circle district, where the probable extension of the Cleary sequence is known as the Bonanza Creek sequence. The Cleary sequence may also be present in the core of an anticline along the north front of the Alaska Range.

Within the Totatlanika schist in the Alaska Range are a number of sites that have been staked and explored for precious-metal-bearing massive-sulfide deposits. The Delta district alone within this unit hosts over 100 mineral occurrences, and the three of the largest occurrences contain tens of millions of tons of ore that grade up to 0.1 oz/ton gold, 3 oz/ton silver, and 6 to 10 percent combined base metals. The inferred resource is probably over 2 million oz gold. These types of occurrences, which are in various stages of exploration, occur westward along the Totatlanika Schist belt and include the Sheep Creek deposits, Red Mountain, Anderson Mountain, and at the west end, the Liberty Bell deposit, which is enriched in gold and arsenic, with mineral reserves in the old workings of 100,000 tons of ore grading 1.2 oz/ton gold and probably much larger reserves of lower grade ore. The Totatlanika Schist also occurs throughout Interior Alaska, where it is much less well exposed than in the Alaska Range, but also hosts numerous placer-gold occurrences. Careful exploration may well reveal lode massive-sulfide deposits in Interior Alaska similar to those that occur in the Totatlanika Schist of the Alaska Range.

Most of the lode and placer gold mined from the Fairbanks mining district occurs within or near the Cleary sequence. The Cleary sequence is also enriched in tungsten and arsenic; because of the latter, water wells within the Cleary sequence often contain anomalous arsenic.

A recent paper presented by P.A. Metz, Mineral Industry Research Laboratory, University of Alaska, reported that volcanically derived rocks of the Cleary sequence rocks contain gold enrichments of 30 to several thousand ppb (normal rocks contain 5 to 10 ppb gold). Arsenic and antimony concentrations in the Cleary sequence were several hundred times greater than those found in average rocks. We speculate that metals were introduced to the Cleary sequence through exhalative submarine volcanic processes. This type of process is responsible for many of the world's significant mineral deposits.

DGGS is now involved in a mineral assessment of 550 sq mi in the Steese-White Mountains area, located 75 mi northeast of Fairbanks. We have recog-

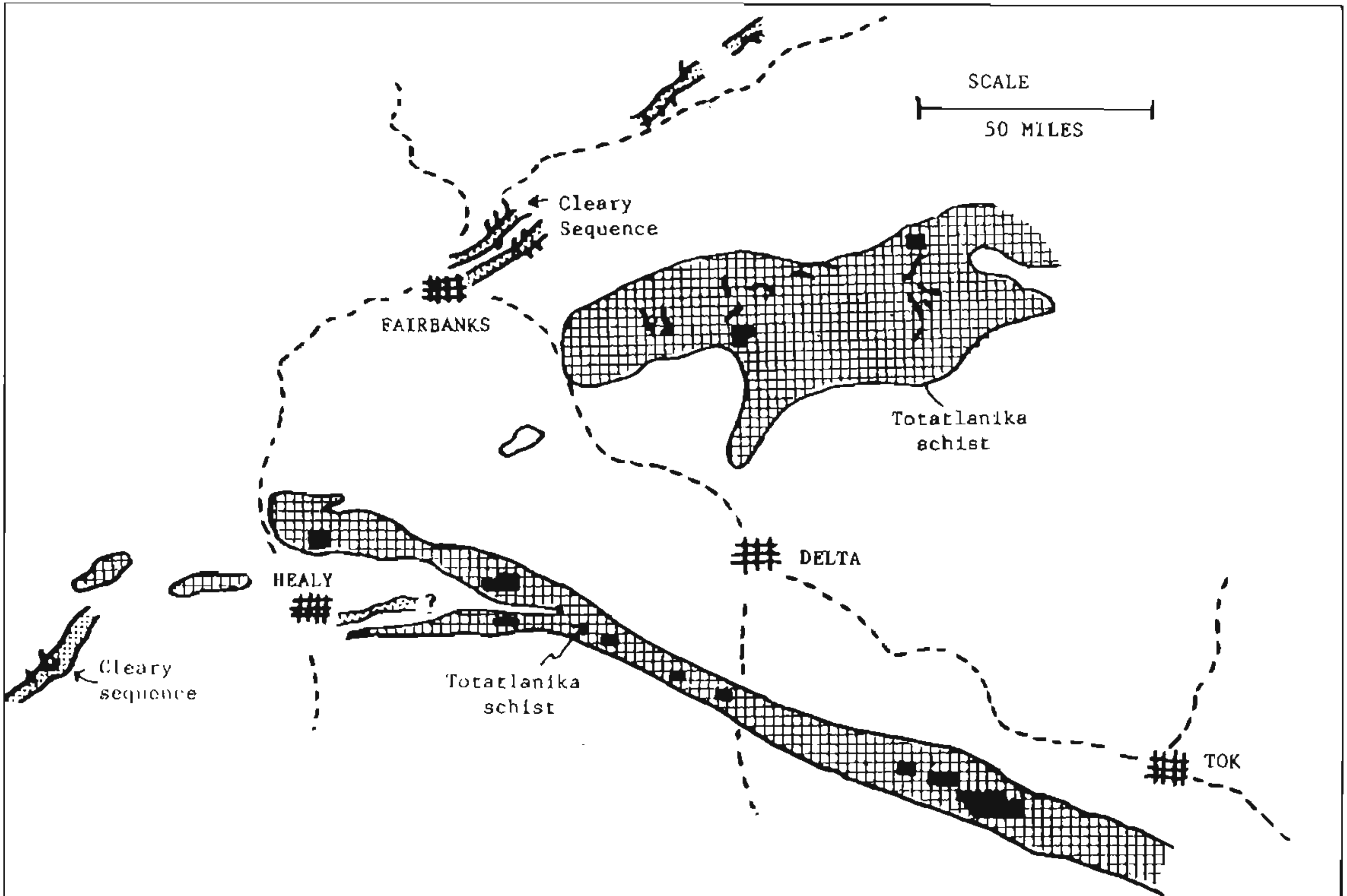

Figure 2. Productive sequences in Interior Alaska.

nized rocks there that are probably correlative with the Cleary sequence of the Fairbanks area. The productive placer mines in this area occur within or near this belt of probable Cleary-sequence rocks. Microprobe analyses of tourmalines collected within these rocks by Rainer Newberry, University of Alaska, suggest these rocks are of volcanic origin. Further investigation of these rocks is one of the components of the proposed North Star Gold investigations.

The Precambrian rocks of Interior Alaska are remarkably similar to the lithology that hosts the Hemlo deposit of Canada, which has a reserve of some 19 million oz gold. The gold is not readily apparent because of its very fine nature, and the deposit was discovered along a roadcut that had been exposed for 20 yr prior to discovery. There appears to be remarkable similarities between the Hemlo rocks and the Precambrian rocks of Interior Alaska.

The Nenana Gravels, which overlie the coal-bearing group in the Healy area, are known to be gold bearing in certain locations and represent high-level paleoplacer deposits. There are some similarities between the Nenana Gravels and the White Channel deposits, which occur in the Yukon and are responsible for the Dawson gold placers. For example, the White Channel is derived in part from the Klondike Schist, which is considered to be the equivalent of the Totatlanika Schist, one of the sources of the Nenana Gravels. The similarities of the Nenana Gravels and the White Channel deposits in terms of metalliferous source rocks invites a closer look at their paleoplacer gold potential.

CONCLUSION

We hope to perform additional studies on gold in Interior Alaska as indicated on figure 3. The successful outcome of this work and the followup and development activity by the mineral industry could have a major impact on Interior Alaska. This season we are planning small-scale site-specific field investigations around some of the known mineralized areas. The North Star Gold Investigation will begin as a small effort and may expand in a few years. We are encouraged by the potential of the rocks in Interior Alaska and look forward to studying them.

BEDROCK INVESTIGATIONS

- Reconstruct volcanic environment
- Identify paleovent areas
- Identify metamorphic core comples complexes and surrounding structural zones
- Establish genetic relationship of gold to bedrock stratigraphy

PLACER INVESTIGATIONS NENANA GRAVEL

- Identify bedrock sources
- Determine transport directions and agents
- Reconstruct depositional environment and link to metalliferous bedrock source terrane
- Evaluate paleoplacer gold potential of Nenana Gravel

Figure 3. Objectives of the North Star gold project.

THE PUBLIC-RECORD DATA BASE

by

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The public-record data base is defined as natural-resource information collected and disseminated by public agencies. This information is disseminated by State of Alaska agencies, by federal agencies, by the University of Alaska and its affiliates, and by various libraries.

Numerous state agencies collect or distribute natural-resource information of interest to the mining community. Within the Department of Natural Resources (DNR), information is available from the Mining Information Offices, the Division of Geological and Geophysical Surveys, the Division of Mining, the Division of Land and Water Management, and the Recorders Offices. The Department of Fish and Game, Department of Environmental Conservation, and Division of Minerals and Forest Products in the Department of Commerce and Economic Development also dispense natural-resource information.

The DNR Mining Information Offices serve as a primary source of information about mining claims in Alaska. These offices are jointly managed by the Division of Mining and the Division of Geological and Geophysical Surveys. The primary office for mining information is located in the Alaska National Bank of the North Building in Fairbanks. Mining information is also available at the Division of Mining Office in the Frontier Building in Anchorage. We hope to reopen the Juneau Mining Information Office on a part-time basis in mid-1987.

The Fairbanks Mining Information Office provides a variety of services to the mining community. It is jointly managed by the Division of Geological and Geophysical Surveys and the Division of Mining. Records of all state and federal mining claims filed since 1953 are available---some as hard copy, others on microfilm. Overlays of 1:250,000-scale topographic maps show both federal and state mining claims. Most overlays are current to 1980 or 1982. Overlays of the Fairbanks, Big Delta, Livengood, and Circle Quadrangles were updated in 1986. The U.S. Bureau of Mines recently initiated a program to update the mining-claim overlays in southeastern Alaska. After these are updated, they will update mining-claim overlays for other regions of the state.

The Fairbanks Mining Information Office also maintains a Kardex file of known mineral occurrences or prospects and patented and unpatented state and federal mining claims. Pertinent data from documents received from the Recorders Offices are noted in the Kardex by mining-claim location. Until June 1986, information recorded in the Kardex file was automated on a monthly basis to produce the MINFILE, which consisted of a series of microfiche updates on mining-claim activity. Until June 1986, the microfiche was distributed to the Anchorage, Juneau, and Ketchikan offices so that they could maintain up-to-date information on mining claims in Alaska. Using the Kardex file, the staff can search mining claims by owner, location, claim name, creek name, and geographic location. Land status plats for the entire state are also available at the Fairbanks Mining Information Office. New aperture cards are received after they are updated in Anchorage. Paper copies of land status plats are generated at a cost of $2 per plat.

An LAS terminal for retrieving information on state mining claims is also present in the Fairbanks office. Because the 1986 annual labor is now being posted on the LAS system, the LAS information base is not quite as up-to-date as that of the Kardex system. The Kardex file is up-to-date within one month of receipt of the mining-claim documents from the Recorders Office.

Selected literature references pertinent to mineral or mining-claim locations are also posted in the Kardex file. These references provide background information on the geology and mineral resources of particular mineral locations. Although the reference list is by no means complete, it does provide a starting point for research on the geology and mineral-resource potential of the area of interest.

Publications by the Division of Geological and Geophysical Surveys (DGGS) are also available at the Fairbanks Mining Information Office. These documents include maps and reports on the geology and mineral resources of Alaska.

The Mining Information Office in Anchorage is located in the Frontier Building. Land status plats of the entire state are on file so that land status can be researched. Via an LAS terminal, information on state mining claims can be retrieved. Once preliminary research is completed, users are referred to the Recorders Office for the most up-to-date information. DGGS publications are also available at the Mining Information Office in Anchorage.

DGGS maintains offices in Fairbanks, Anchorage, Eagle River, and Juneau. The office of the State Geologist is located in Fairbanks. The primary functions of DGGS include determining the location and potential of energy, mineral, and water resources and providing information on health and safety related to ground water and geologic hazards. The results of geologic fieldwork conducted by the division are published as maps and reports available to users at minimal cost. DGGS also maintains the Alaska Geologic Materials Center, which was established in 1985 in cooperation with the U.S. Geological Survey. The Materials Center, which is located in Eagle River, serves as a central repository for geologic materials collected in Alaska. These materials include nonproprietary drill-core, records, and rock samples that are catalogued and archived. They are then available for examination by the public.

The Fairbanks office of DGGS is an official repository for U.S. Geological Survey Open-file reports on Alaska. These reports constitute a substantial part of the geologic data base on Alaska and are available for review or copying.

In cooperation with the Division of Minerals and Forest Products and the Division of Mining, DGGS produces the annual edition of 'Alaska's Mineral Industry.' In this document, staff geologists outline mineral activity in Alaska by year. The report includes discussions of mineral exploration, development, and production statewide.

Another state agency that provides information of use to the mineral community is the Division of Mining, which maintains offices in Fairbanks and Anchorage. Primary functions of the Division of Mining include overseeing coal leases, offshore mining, permitting, and mining-claim adjudication. Division personnel are available in both Fairbanks and Anchorage to answer your inquiries.

The Division of Land and Water Management (DLWM) maintains regional offices in Fairbanks, Anchorage, and Juneau. Its primary function is to maintain records on Alaska's land and mineral estates. Each regional office has land status plats for the entire state and an LAS terminal to research land status. DLWM staff are also available to answer your inquiries.

The Recorders Office maintains bases in most primary towns in Alaska. Their function is to record all documents, including all mining-claim documents. Their data are also entered into the LAS system. Although the Recorders Office is a central repository for all mining-claim documents, research capabilities are limited because of access by owner name only.

In addition to the Department of Natural Resources, several other state agencies provide natural-resource information of interest to the mining community. These include the Department of Fish and Game, the Department of Environmental Conservation, and the Division of Minerals and Timber Products in the Department of Commerce and Economic Development.

Federal agencies that provide natural-resource information include the U.S. Geological Survey (USGS), the U.S. Bureau of Mines, and the U.S. Bureau of Land Management. Like the Alaska DGGS, the USGS Branch of Alaskan Geology synthesizes the results of geologic fieldwork into maps and reports that are available to users. A monthly listing of publications is available through USGS offices in Anchorage and Fairbanks. A free annual circular on geologic studies in Alaska is also available through Information Offices in Anchorage and Fairbanks. The U.S. Geological Survey maintains the Alaska Distribution Section in the Federal Building in Fairbanks, where topographic and thematic maps are available. Bulletins, Professional Papers, and topographic and geologic maps are available at the Anchorage Public Information Office. Open-file reports are officially available through the Denver offices of the USGS, but copies can be generated from mylars on file in Anchorage. The Branch of Alaska Geology is headquartered in Anchorage on the campus of Alaska Pacific University. A branch office is located at the College Observatory in Fairbanks. Professional staff are available at both offices to answer your inquiries.

The National Cartographic Information Center-Alaska (NCIC), a part of the U.S. Geological Survey, maintains its offices on the Alaska Pacific University campus in Anchorage. NCIC works in cooperation with its state affiliate, the Geo-Data Center at the University of Alaska (Fairbanks) Geophysical Institute. NCIC staff are available to assist the public in locating and ordering aerial photographs, satellite imagery, digital cartographic data and software, and other map materials. NCIC maintains an aerial-photo collection that covers selected areas of Alaska flown from 1926 to 1986. Scales range from 1:20,000 to 1:120,000 in color infrared or black and white.

Another source of resource information is the USGS Technical Data Unit, which is in the process of moving from Menlo Park to Anchorage. This unit specializes in maintaining unpublished basic field data on Alaska. They archive field notes, thin sections, and other field and laboratory data used to generate geologic maps and reports. The Branch of Alaska Geology has also been active in developing a geologic library in conjunction with Alaska Pacific University.

The U.S. Bureau of Mines maintains offices in Fairbanks, Anchorage, and Juneau. Their efforts are concentrated on compiling statistics on the domestic and foreign production, uses, and consumption of mineral commodities. Both 'country' and 'commodity' experts are employed by the Bureau of Mines. Primary publications include annual editions of the Minerals Yearbook and Mineral Commodity Profiles.

Most formal publications of the U.S. Bureau of Mines are housed at the University of Alaska-Fairbanks. Selected publications on Alaska's mineral resources are available in the O'Neill Building, University of Alaska-Fairbanks. The Juneau office of the U.S. Bureau of Mines houses one of the most complete collections of literature on the geology of Alaska. Professional staff are available at all offices to answer your inquiries.

The U.S. Bureau of Land Management (BLM) maintains records of all mining claims on federal land. In addition, land status records are available to determine areas open to mineral entry. For patented mining claims, BLM maintains minerals surveys and field notes. For unpatented mining claims, case files are maintained. BLM can search mining claims by owner name and location. Staff are available to answer your inquiries and help with land status research.

The University of Alaska and its affiliates also collect and distribute natural-resource information of interest to the mining community. In the School of Mineral Engineering, the Mineral Industry Research Laboratory recently established a geologic-materials storage facility in Fairbanks. Additional sources of information in the School of Mineral Engineering include staff in the Mining and Geological Engineering Program and the Mining Extension Service, which offers evening courses on basic prospecting, geochemical prospecting, and rock identification. Professional staff in the UAF Department of Geology and Geophysics are available to answer inquiries about the geology and mineral resources of Alaska. Air photos are available for perusal at the Geo-Data Center at the Geophysical Institute, which maintains a collection of NASA high-altitude photography for virtually all of Alaska, along with Landsat data from 1972 to the present. Indexes to historical aerial photography are also available. Staff will show the photos or assist in ordering copies.

Other sources of resource information associated with the University include the Arctic Environmental Information and Data Center located in Anchorage, the University Museum, and the Tanana Valley Community College Mining Technology Program.

Most libraries in Alaska provide information of interest to the mining community. Examples include collections at the Rasmuson Library (University of Alaska) in Fairbanks, particularly the Skinner Collection, which includes copies of all literature on Alaska. Masters and doctoral theses on Alaska's geology are found in this collection. The Government Documents Collection includes USGS and U.S. Bureau of Mines publications. The University of Alaska-Anchorage Consortium Library includes a geologic collection, as does the Alaska Resources Library in Anchorage. The USGS, in conjunction with Alaska Pacific University, is establishing a substantial geologic library on the APU campus in Anchorage. Lastly, one of the most complete collections of geologic and minerals literature is contained in the U.S. Bureau of Mines Library in Juneau.

One of the most substantial sources of information on Alaska's geology and minerals are the professionals working in the public sector. They are interested in and familiar with the geology of Alaska.

GEOLOGICAL FACTORS GOVERNING THE FORMATION OF THE GOLD PLACER DEPOSITS OF THE FAIRBANKS MINING DISTRICT, ALASKA

by

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ABSTRACT

This paper presents new evidence on the bedrock sources of the placer gold, bedrock structural control of stream drainage, and surficial depositional controls of placer formation in the Fairbanks mining district. Of the major controls of placer formation, the most important is the alteration of stream drainages by basement structures, which can lead to stream capture, stream reversal, and sediment resorting.

INTRODUCTION

The Fairbanks mining district is located in central Alaska and is bounded on the south by the Tanana River, on the north by the Chatanika River, on the west by Ester Dome, and on the east by the Little Chena River (fig. 1). The district is in the northwestern part of the Yukon-Tanana Uplands Schist Terrane (Foster and others, 1973) and occupies about 400 sq mi.

Gold was discovered in the Fairbanks district in 1902 and since then the area has produced 7,500,000 troy oz of placer gold and 250,000 troy oz lode gold. The district also produced several thousand tons of antimony and several thousand short-ton units of tungsten. Although it has been the singlemost important placer district in Alaska, the major controls of placer formation are only now being recognized. This paper presents new evidence on the bedrock sources of the placer gold, bedrock structural control of stream drainage, and surficial depositional controls of placer formation.

PREVIOUS INVESTIGATIONS AND PLACER STRATIGRAPHY

Prindle and Katz (1913) were the first to provide a general description of the bedrock and surficial geology of the Fairbanks district. Many of the rock units that they defined have been retained by subsequent investigators; however, the most significant contribution was their detailed descriptions of the gold placer deposits. Of particular importance are data on the thickness of both overburden (including reworked loess and organic material locally known as muck) and the auriferous alluvial gravels. Prindle and Katz (1913)

Ed. note: The author's presentation of 'Applications of Stream Sediment-Drainage Pattern Analysis to Gold Placer Exploration' is not currently available in written format; hence this paper is offered as a substitute.

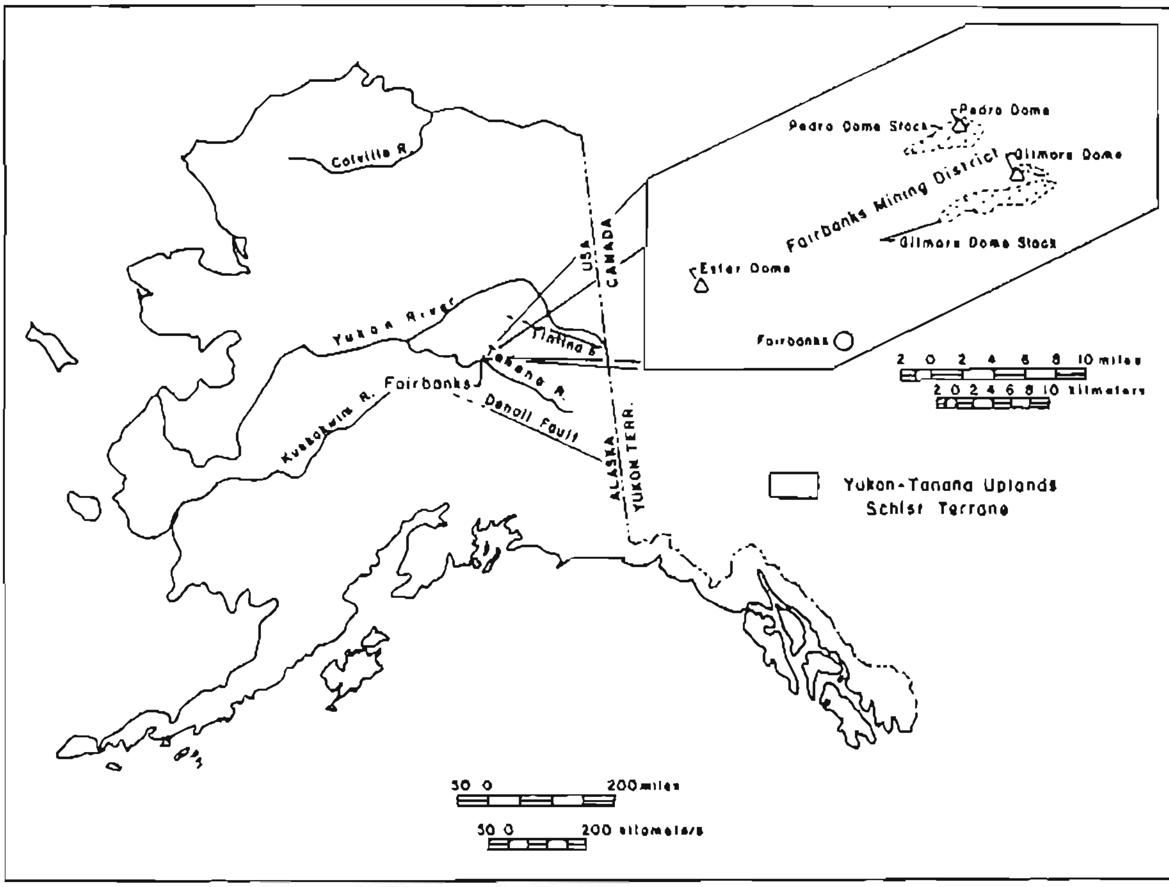


Figure 1. Location of the Fairbanks mining district and the Yukon-Tanana Uplands Schist Terrane.

also recorded data on the thickness, length, and width of the economic concentrations or paystreaks of placer gold, as well as the average depth to bedrock for most of the creeks in the district. Although their predictions of ultimate production were 50 percent lower than the total achieved to date, their estimates must be recognized as good in view of the data available.

In addition to providing fundamental geological descriptions, Prindle and Katz (1913) were the first to describe two important mechanisms of placer formation: headward migration of stream drainage and changes in base level. Headward migration of stream drainage was described only qualitatively, and no analytical study was made of stream gradients, stream profiles, or the critical point between degradation and aggradation. The existence of both older terrace deposits and buried placers was noted as evidence of major changes in sea level.

Further, Prindle and Katz (1913) related the placer deposits to gold-quartz and gold-quartz-sulfide veins, the formation of which in turn was attributed to the intrusion of granitic rocks in the district. Smith (1913a) provided detailed descriptions of several lode deposits, which gave support to the observations made by Prindle and Katz (1913).

Smith (1913b) also reported gold fineness values for 167 placer and six lode occurrences. He noted that placer gold tended to be finer than lode gold and that fineness increased downstream. However, he did not describe in detail how fineness changed nor did he relate fineness values to the mechanisms of placer formation.

Chapin (1914, 1919), Mertie (1918a), and Hill (1933) also described the lode deposits of the district, and each noted the close spatial relationship of the placer deposits to the lode occurrences. The gold-quartz veins were considered to be the sole lode source and the veins were thought to be related to the granitic intrusive rocks of Gilmore Dome and Pedro Dome. Hill (1933) included a brief description of the placer deposits and provided records of placer production from 1903 to 1931.

Mertie (1937, 1940) reiterated the mechanisms of placer formation described by Prindle and Katz (1913) but also noted the importance of Pleistocene and recent climatic variations affecting the geomorphology of the entire Yukon-Tanana region.

Tuck (1968) noted that although the alluvial and beach placer gold is usually disseminated throughout the gravel sections, the most productive zones are all on or near the contact of the gravels with the underlying bedrock. Historically, commercial operations were confined to paystreaks containing gold particles greater than 1 mg, and gold farther than a few feet from bedrock tends to be finer grained than this. Gold values tend to be distributed laterally along bedrock with major concentrations in narrow paystreaks usually less than 100 m wide. Gold particles over 1 mg are usually limited to downstream transport of 3,000 to 5,000 mg, however, gold grains of less than 1 mg may travel much greater distances (Tuck, 1968).

Chapman and Foster (1969) also related the gold-quartz vein deposits of the district to the Cretaceous granitoid intrusions exposed at Pedro and Gilmore Domes and concluded that the quartz veins were the sole source of the placer gold.

Péwé (1975) described 15 stratigraphic units of Quaternary age in central Alaska, of which 13 type sections are in the Fairbanks mining district. Two stratigraphic units, the Cripple Gravel and the Fox Gravel, are the main gold placer formations of the area (fig. 2).

The Cripple Gravel is a brown auriferous gravel of late Pliocene or early Pleistocene age. The unit consists of poorly sorted to well-stratified, coarse, angular sandy gravel with lenses of silt and sand. The gravel clasts consist of quartz-mica schist, quartzite, chlorite schist, biotite-garnet schist, feldspathic schist, calc-schist, graphitic schist, phyllite, slate, quartz monzonite, granodiorite, and granite. Gravel fragments range from 2 to 15 cm in diameter with basal cobbles and boulders from 25 cm to 1 m in diameter. A minor white facies of the gravel occurs at the mouth of Engineer Creek and is composed of well-rounded quartz and quartzite cobbles and boulders in a gray sand matrix. The white-gravel facies is devoid of other metamorphic or igneous rock types. This mature facie was a major source of high-

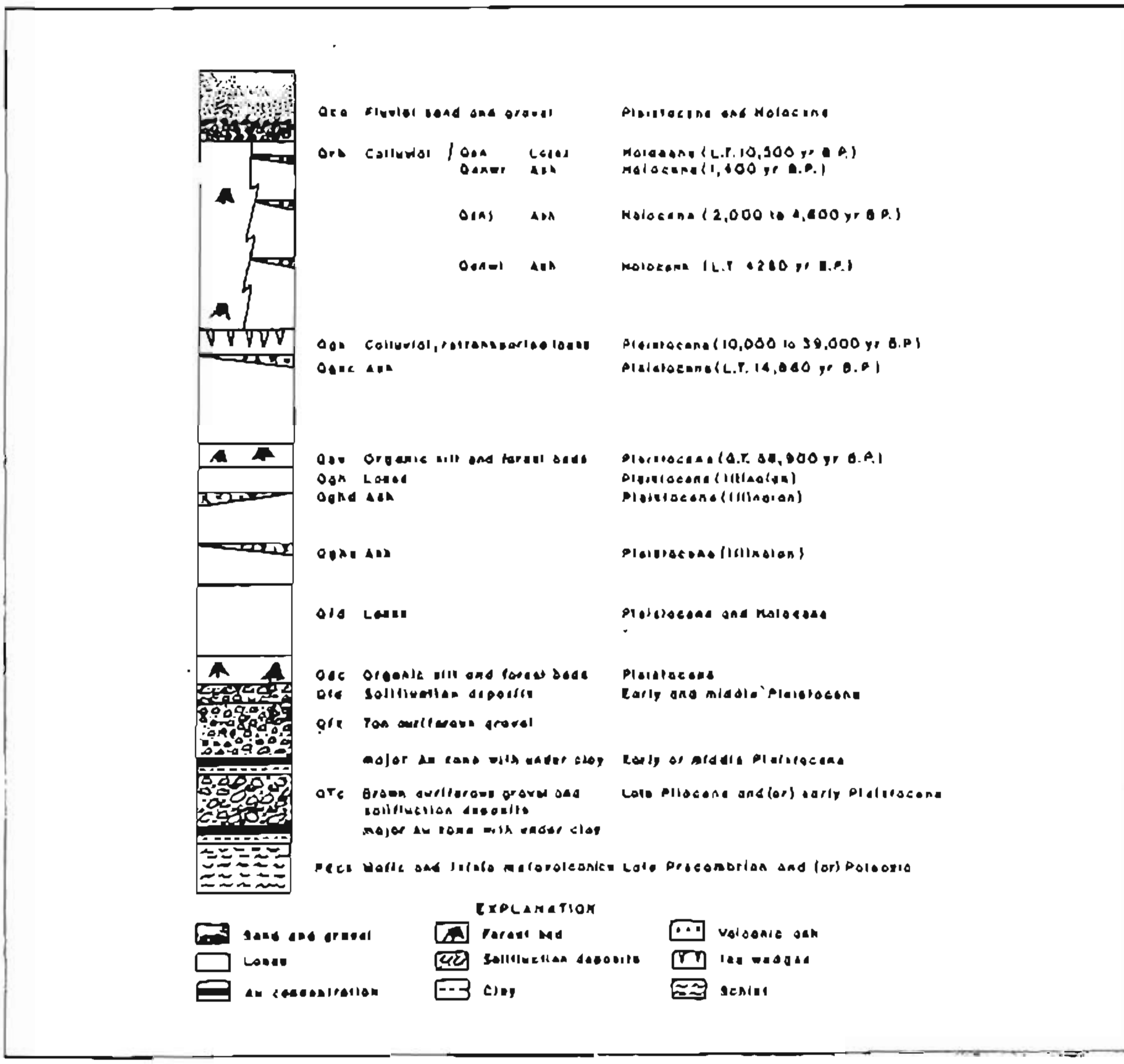


Figure 2. Schematic section of the surficial stratigraphy of the Yukon-Tanana Uplands Schist Terrane (after Péwé, 1975). Units are as follows: Qca = Chena alluvium; Qen = Engineer loess; Qenj = Jarvis ash bed; Qgs = Goldstream formation; Qev = Eva formation; Qghd = Dome ash bed; Qfd = Fairbanks loess; Qta = Tanana formation; Qtc = Cripple Gravel; Qrb = Ready Bullion formation; Qenwr = White River ashbed; Qenwi = Wilbur ash bed; Qgsc = Chatanika ash bed; Qgh = Gold Hill loess; Qghe = Ester ash bed; Qdc = Dawson Cut formation; Qfx = Fox Gravel.

grade mineralization and is similar to the White Channel deposit of the Klondike district, Yukon Territory.

The Cripple Gravel ranges from 1 m thick on terrace benches to 25 m thick in the lower end of stream drainages.

The Cripple Gravel is generally underlain by a 0.5- to 2-m-thick auriferous clay that overlies bedrock. Bell (1974) attributed the clay to either pre-Pleistocene weathering or to percolation of groundwater during Pleistocene interglacial times. The Cripple Gravel is in turn overlain by the Fox Gravel and either the Tanana formation, Dawson Cut formation, Fairbanks loess, Gold Hill loess, or the Goldstream formation (Péwé, 1975).

The Fox Gravel is a tan auriferous gravel of early or middle Pleistocene age. The unit consists of poorly to well-stratified, angular sandy gravel with lenses of sand and silt. The gravel contains a similar population of clasts as found in the Cripple Gravel and the composition varies according to the bedrock source areas. Gravel size distribution is comparable to the Cripple Gravel. The Fox Gravel is restricted to valley bottoms below loess and solifluction deposits and is exposed only in mining excavations.

The Fox Gravel lies directly on bedrock or on the Cripple Gravel and is overlain in turn by the solifluction deposits of the Tanana formation, the Dawson Cut formation, Fairbanks loess, Gold Hill loess, or the Goldstream formation. The Fox Gravel is the predominant source of placer gold in the Yukon-Tanana Upland (Péwé,1975).

The Fox Gravel varies in thickness from 1 m in the upper stream drainages to over 30 m in the lower reaches. Where the unit lies near bedrock it is usually underlain by a 0.5- to 2-m-thick yellow to blue clay layer that is thought to have the same origin as the clay layer below the Cripple Gravel (Bell, 1974).

Péwé (1975) defined four criteria for distinguishing the older Cripple Gravel from the Fox Gravel. First, the Cripple Gravel is always on bedrock benches above modern valleys that contain Fox Gravel. Second, gold in the Cripple Gravel has a higher average fineness than Fox Gravel gold when the samples are taken from the same creek and the same distance from the lode source. Third, Cripple Gravel pebbles are stained by iron oxide to dark brown, whereas Fox Gravel is tan and less stained. Fourth, Fox Gravel may contain a few bones of Pleistocene mammals; Cripple Gravel is devoid of bones.

BEDROCK SOURCES OF PLACER GOLD

Although a detailed description of the lode deposits of the Fairbanks district is beyond the scope of this discussion, a review of the sources of placer gold is not. As noted, the Fairbanks mining district is located in the northwestern part of the Yukon-Tanana Uplands Schist Terrane (fig. 1). Figure 3 is a generalized geological map of the terrane, which is bounded on the south by the Denali Fault and on the north by the Tintina Fault (Foster and others, 1973). The terrane is composed of Precambrian or Paleozoic metamorphosed sedimentary and volcanic rocks that contain lower-greenschist to amphibolite facies, eclogite facies, and granulite facies mineral assemblages. The metamorphic rocks were formerly designated the Birch Creek Schist (Mertie, 1937). The metamorphic rocks are unconformably overlain by Paleozoic, Mesozoic, and Tertiary sedimentary and volcanic rocks. The meta-

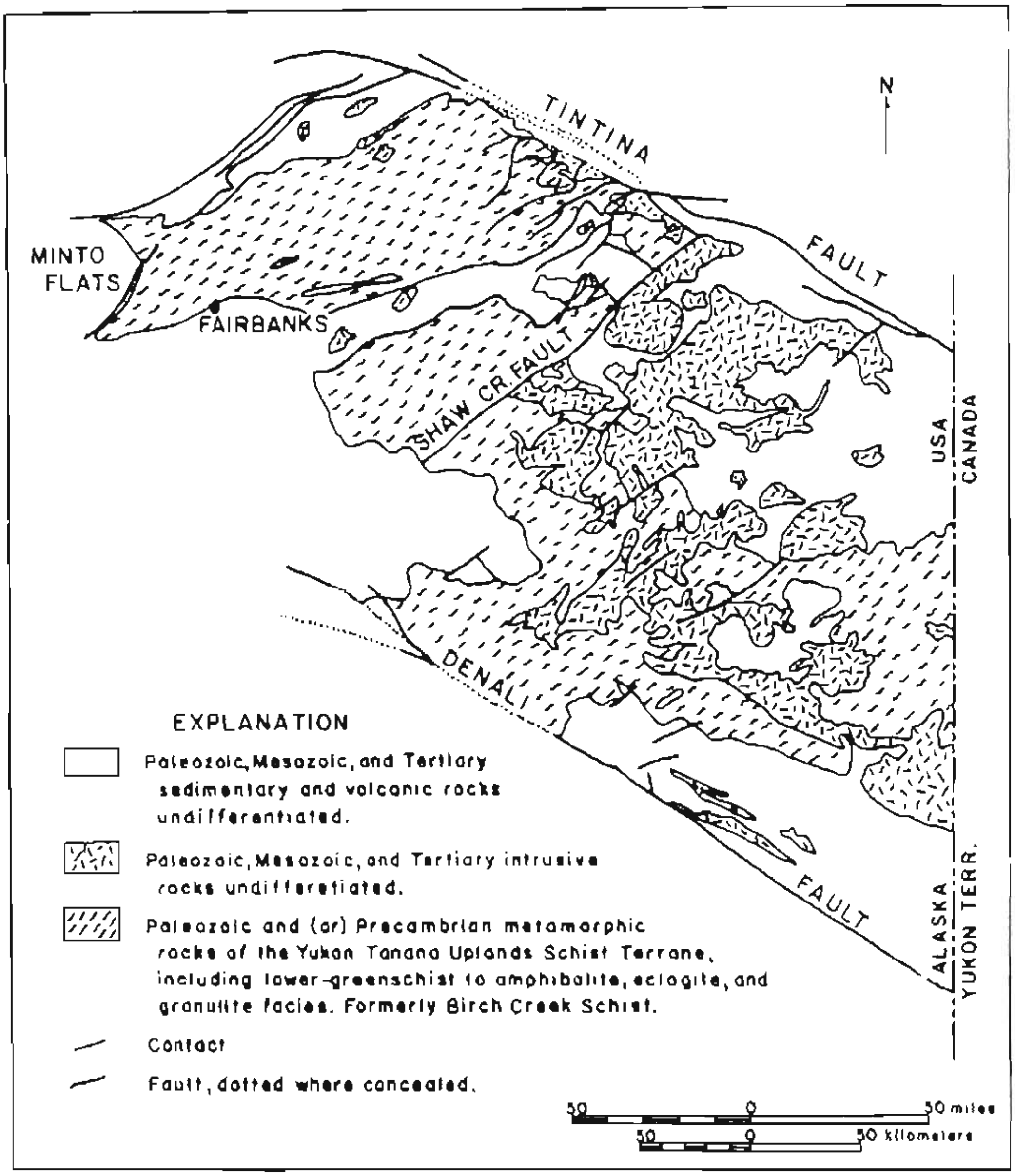


Figure 3. Extent of the Yukon-Tanana Uplands Schist Terrane in Alaska (after Foster and others, 1973).

morphic and sedimentary sequences are intruded by Paleozoic, Mesozoic, and Tertiary rocks ranging from peridotite to granite.

Metz (1977) and Metz and Robinson (1980) suggest that the antimony-tungsten and associated gold mineralization of the Fairbanks district is related to previously unrecognized metavolcanic rocks in the Yukon-Tanana Uplands Schist. Recent geological mapping (Metz, 1982; Bundtzen, 1982; and Robinson, 1982) has delineated the extent of these metavolcanic rocks (fig. 4). Most of the known lode mineral occurrences in the district are within these metavolcanic rocks, which have been named the Cleary sequence (Metz, 1982); however, the lode occurrences are not necessarily spatially associated with exposed intrusive rocks.

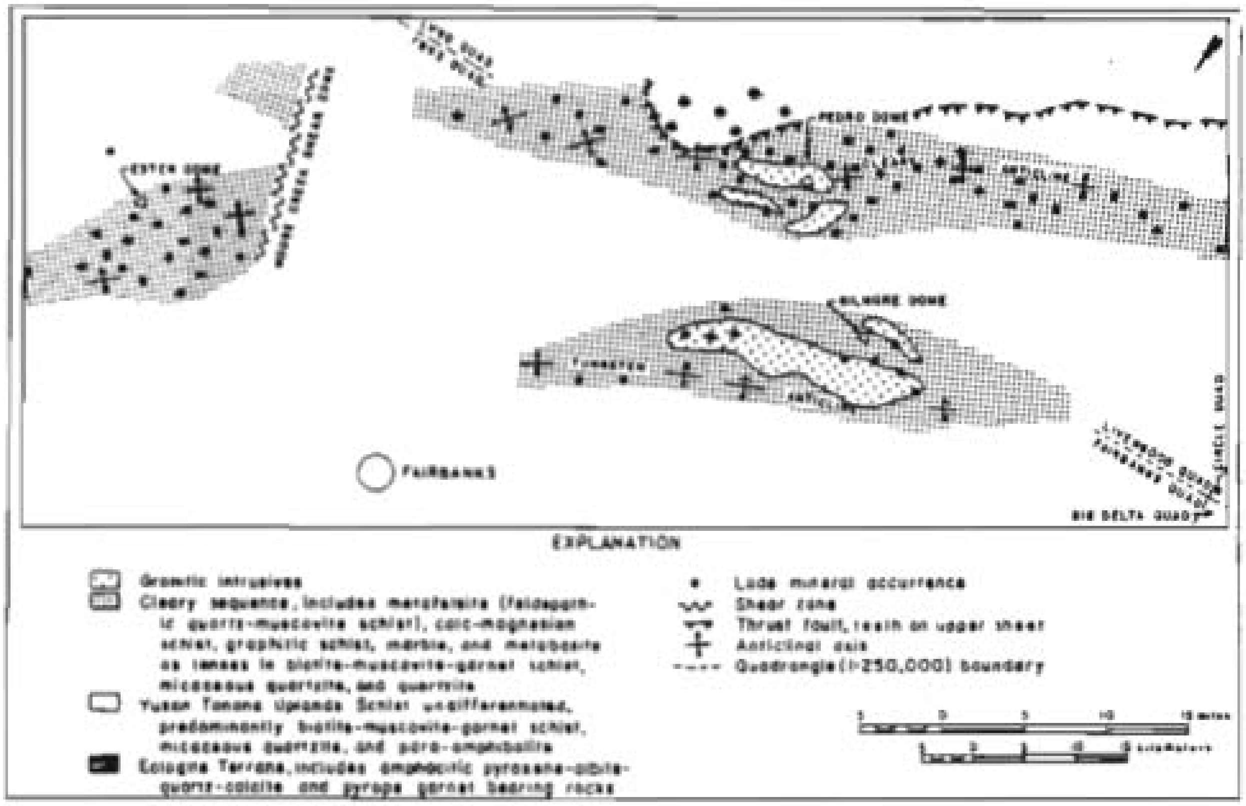


Figure 4. Distribution of lode mineral occurrences and major rock units in the Fairbanks mining district.

Detailed investigations of the lode mineral deposits have resulted in the definition of five major types of lode mineralization within the district:

1) Volcanogenic Stratabound Mineralization - in which intergrowths of zinc, antimony, lead, and copper sulfides ± gold and scheelite occur in conformable laminae and lenses parallel to both foliation and compositional banding in the metavolcanic host rocks

2) Lead Sulfosalt-bearing Quartz Sulfide Veins - with argentiferous galena, sphalerite, chalcopyrite, stibnite, arsenopyrite, and gold, which occur in Cretaceous intrusions

3) Tungsten Skarn Mineralization - scheelite-bearing calc-silicate mineralization found adjacent to the Gilmore Dome and Pedro Dome granitic stocks

4) Gold-bearing Polymetallic Quartz Sulfide Veins - which crosscut the metavolcanic host rock of the schist sequence

5) Stibnite Gash Veins and Fracture Fillings - associated with axial plane shears in the metavolcanic host rocks.

Two types of lodes contain free gold that contributed to placer formation. Sulfide lenses and disseminations in the metavolcanics contain free gold. These lenses have been deformed during metamorphism; thus, native gold within the sulfides must be of premetamorphic origin and is probably syngenetic (figs. 5a and 5b). Gold in the sulfide lenses is always fine-grained (<1 mg) and thus probably has not contributed significantly to the gold mineralization exploited in the commercial placer operations to date. This source of fine-grained gold may provide the finer grained material disseminated vertically and laterally throughout the gravels, extending for considerable distances beyond major paystreaks. This size fraction of gold would also tend to accumulate further downstream with finer grained sediment.

The gold-quartz vein deposits that have been the sole source of lode-gold production in the district contain fine- to coarse-grained gold. The veins are postmetamorphic and are probably the main source of gold subsequently accumulated in these Pliocene-Pleistocene placer deposits (figs. 6a,b).

MECHANISM OF ALLUVIAL PLACER FORMATION

Mosley and Schumm (1977) and Adams and others (1978) conducted extensive laboratory studies to determine possible mechanisms of alluvial-placer formation; however, these studies did not attempt to relate the flume model results to field observations in major alluvial placer-producing areas. The major alluvial gold placers of the Fairbanks district provide an opportunity to test these models, particularly the significance of the basin rejuvenation process proposed by Adams and others (1978) as the predominant mechanism responsible for placer formation.

The locations of the gold placer deposits of the Fairbanks district together with the present stream drainage systems are shown in figure 7. In addition to the study of the stratigraphy of the Pliocene-Pleistocene surficial deposits, the examination of the patterns of stream drainage and the longitudinal profiles of the streams provides the best evidence of the critical mechanisms of alluvial placer formation.

Stream Longitudinal Profiles

The significance of stream longitudinal profiles can be summarized as follows. First, the concavity of the profile reflects the relative increase in discharge in the lower reaches of the drainage system. Extreme concavity of the profile indicated increased discharge in the lower reaches, whereas a linear profile would indicate constant discharge for the entire system. Convex upward profiles are possible in arid regions, where discharge decreases because of evaporation and infiltration.

Second, the stream profile is an expression of changes in the size distribution of the bed load material in a downstream direction (Shulitz, 1941). The slope-discharge relationship (Sternberg, 1875; Woodford, 1951) and the slope-grain size relationship (Shulitz, 1941) are exponential functions, the former having a positive exponent and the latter a negative exponent.

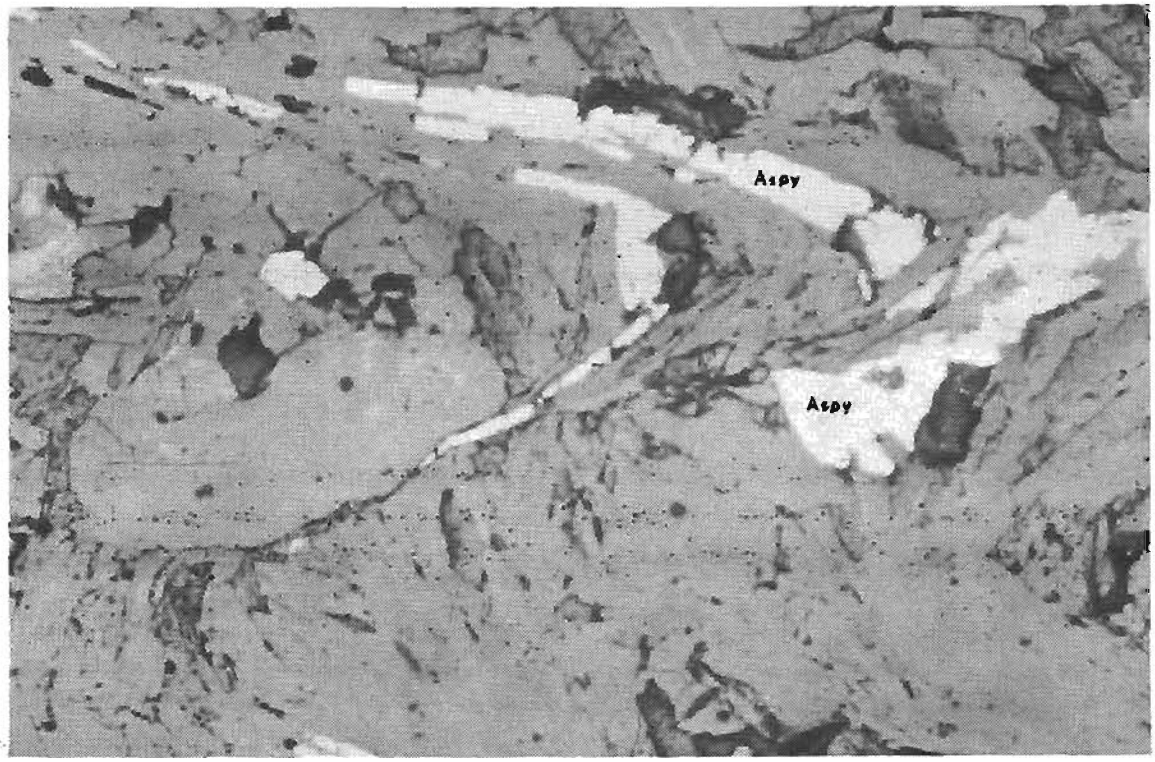


Figure 5a. Isoclinally folded lense of arsenopyrite (light gray) in feldspathic quartz muscovite schist (field of view 1.5 mm).

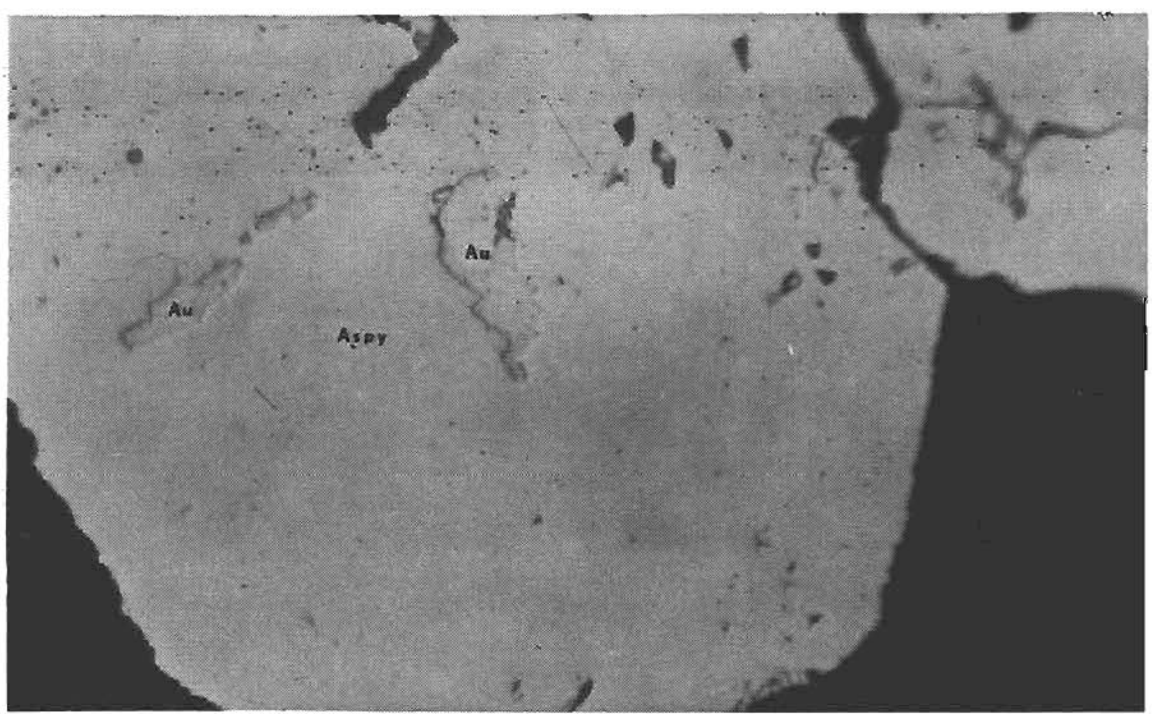


Figure 5b. Gold (white) included in arsenopyrite (light gray) in folded lense of figure 5a (field of view 0.15 mm).

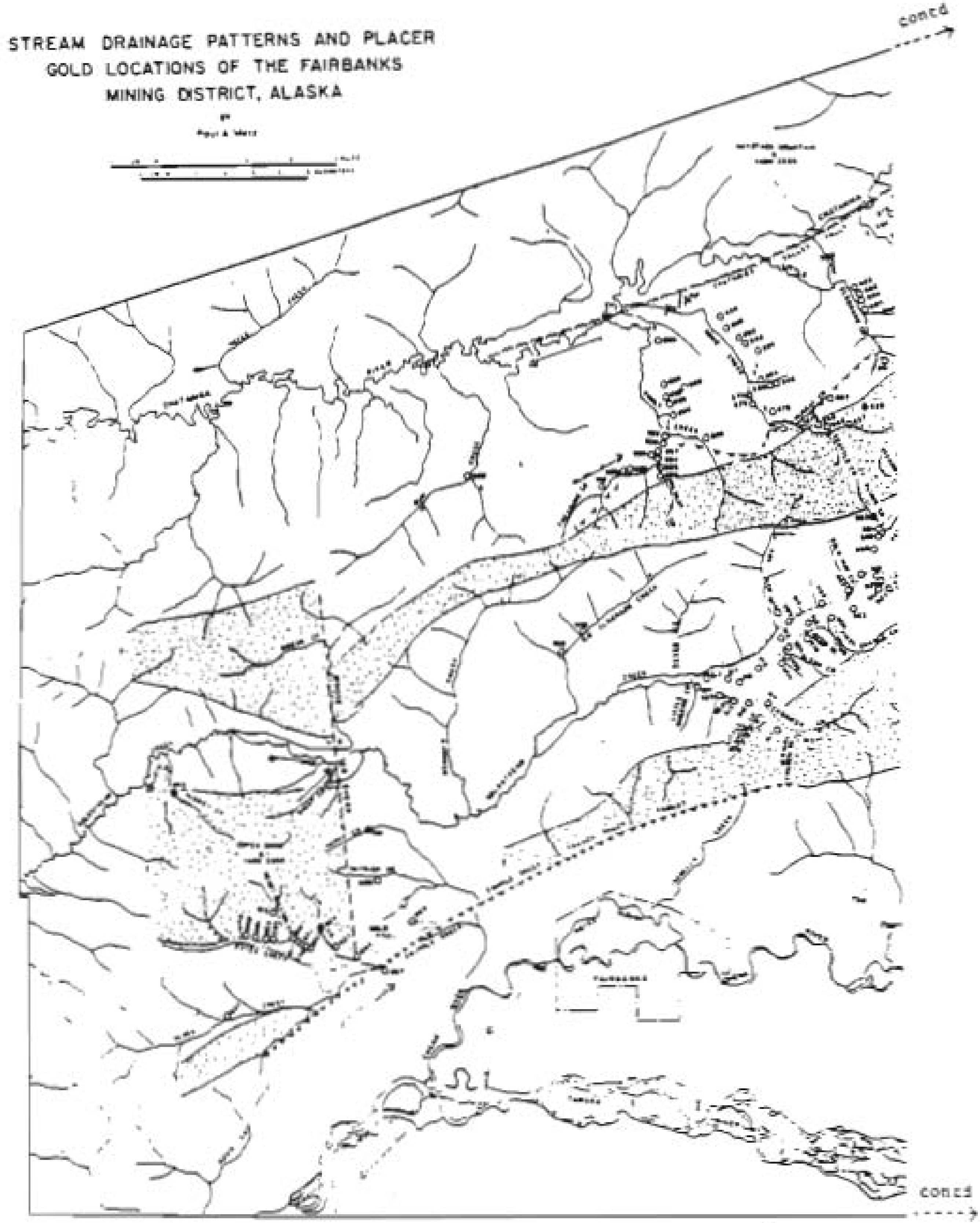
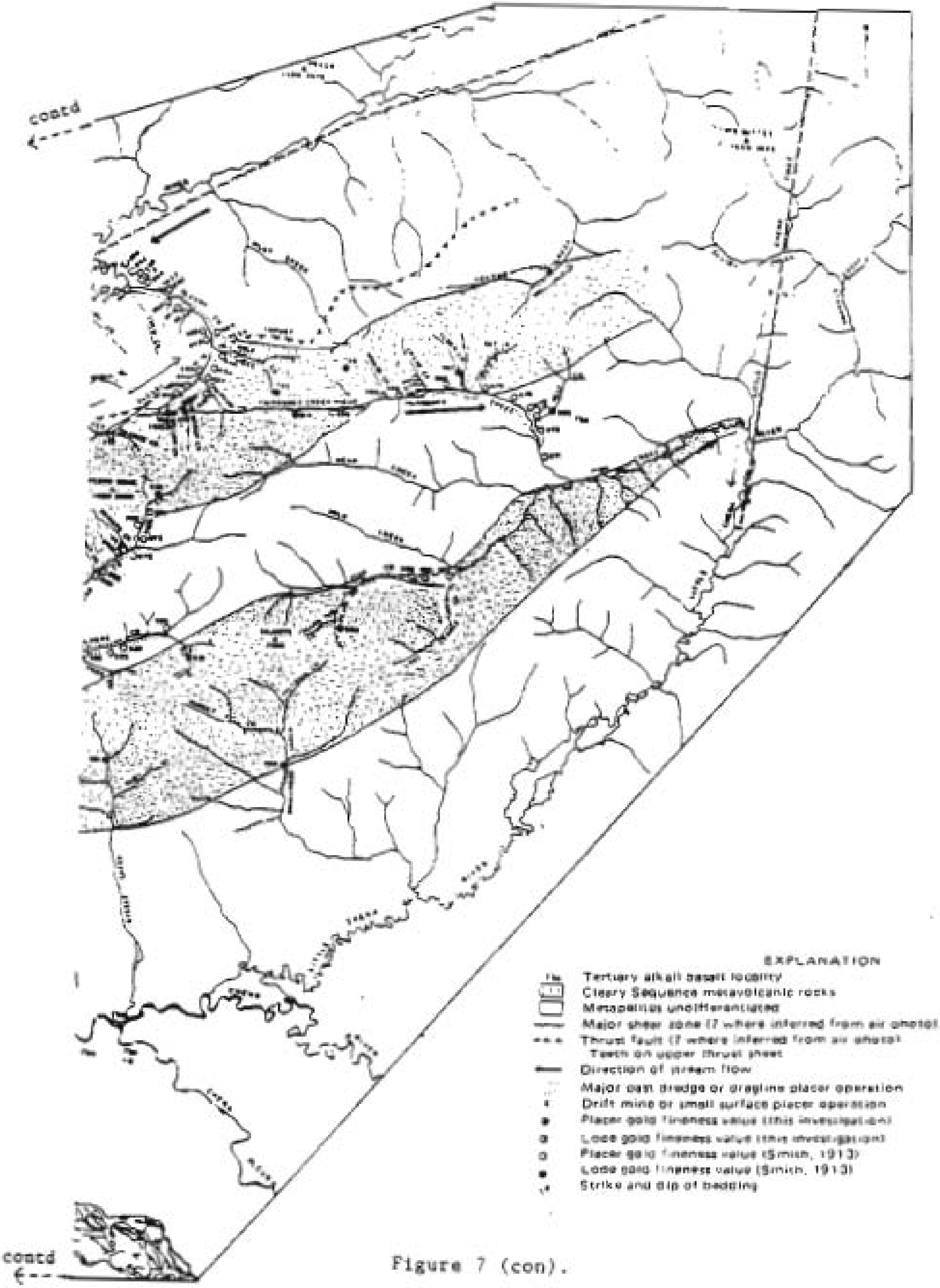


Figure 7. Stream drainage patterns and placer gold locations of the Fairbanks mining district.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EXPLANATION
Tertiary alkali basalt locality
Cleary Sequence metavolcanic rocks
Metapelites undifferentiated
Major shear zone (? where inferred from air photo).
Thrust fault (? where inferred from air photo)
Teeth on upper thrust sheet
Direction of stream flow
Major cast dredge or dragline placer operation
Drift mine or small surface placer operation
Placer gold fineness value (this investigation)
Lode gold fineness value (this investigation)
Placer gold fineness value (Smith, 1913)
Lode gold fineness value (Smith, 1913)
Strike and dip of bedding

Figure 7 (con).

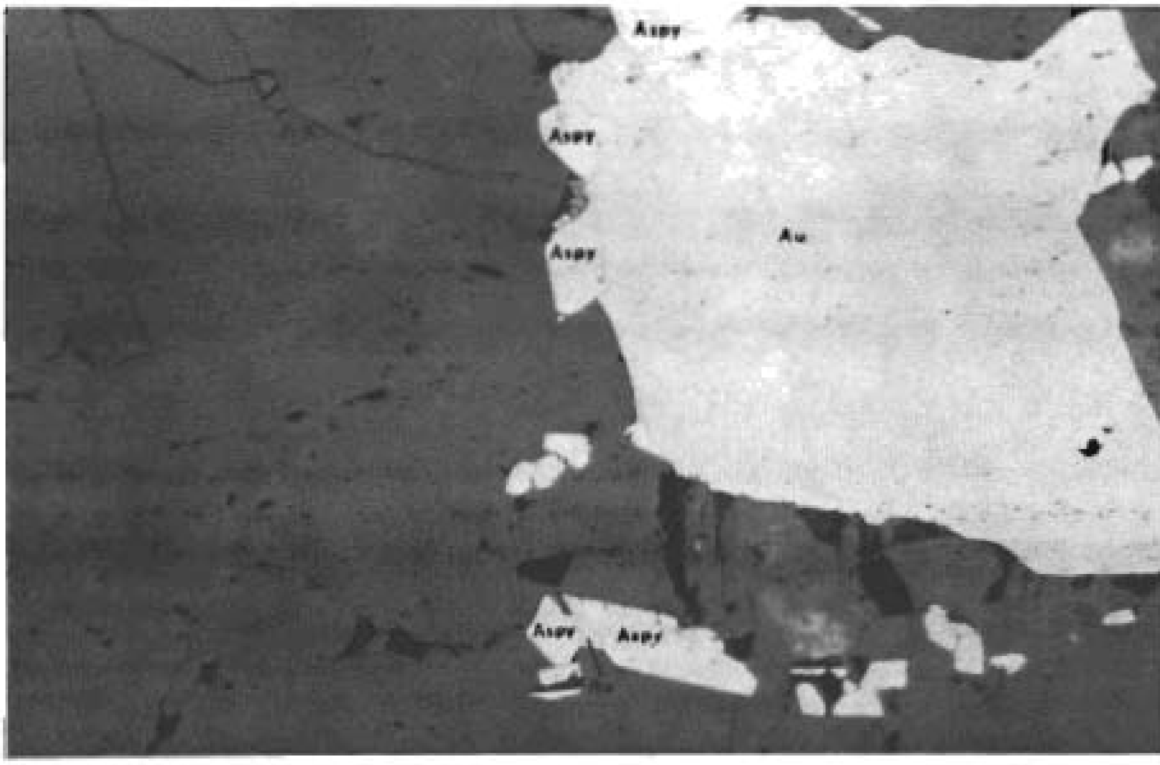

Figure 6a. Gold (white) and arsenopyrite (light gray) in quartz (dark-gray) vein (field of view 1.5 mm).

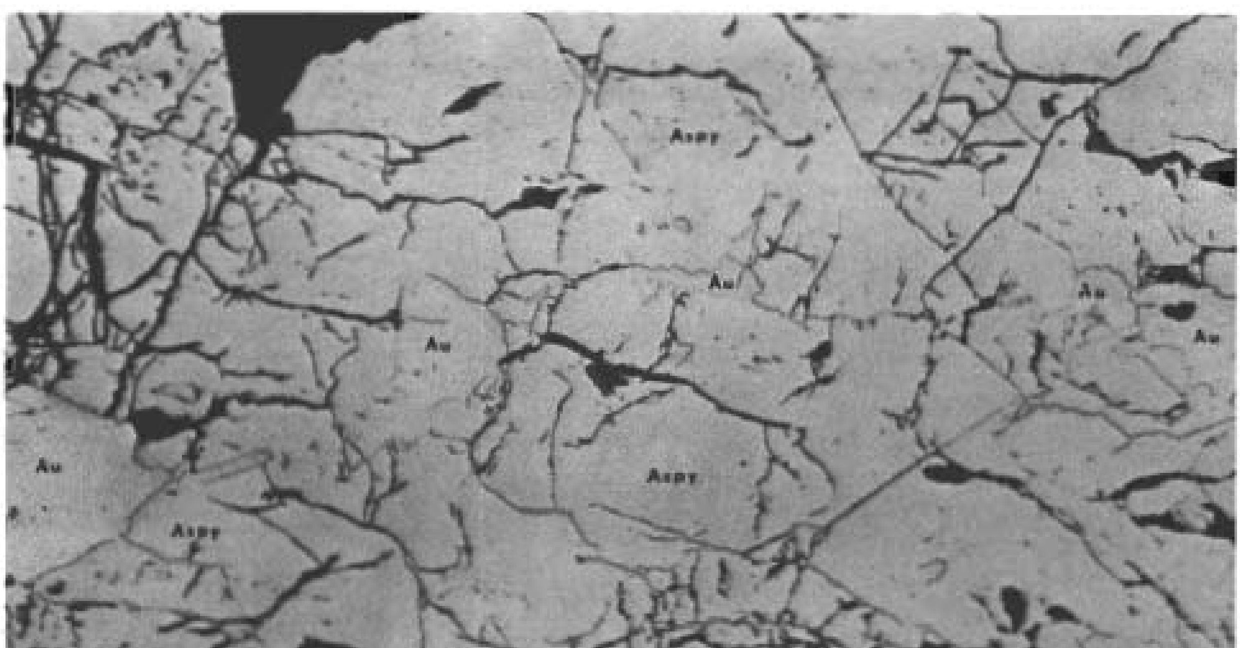

Figure 6b. Gold (white) replacing arsenopyrite (light gray) in quartz (dark-gray) vein (field of view 1.5 mm).

Gravels are found in the steeper upper parts of the profile, whereas finer material accumulates in the lower reaches. There is a critical point on the profile above which material of a given grain size and density is maintained in a continuous state of transport (the zone of degradation) and below which the material begins to accumulate (the zone of aggradation). Since gold is much denser than rock-forming minerals, the critical point for gold accumulation lies further upstream than that of other sedimentary materials. Thus there is a zone in which rock material is constantly being removed relative to gold, thereby forming a placer paystreak.

Third, the stream drainage develops by headward migration of the stream profile, and coarse material and gold in the zone of accumulation are gradually buried by finer grained sediments as the critical point moves headward.

Fourth, the shape of the the profile is a reflection of the maturity of the stream drainage. Generally regular profiles indicate mature drainage systems involving long periods of weathering and erosion (and adequate time for major placer formation). Irregular profiles generally indicate immature drainage; however, alternating steep and gentle slopes produce marked changes in stream flow, which may result in local minor placer accumulation.

Stream profiles for all drainages of the Fairbanks district have been plotted (note topographic maps are in nonmetric units, whereas profiles and slopes are expressed in ft/mi). The locations of major placer deposits and tributary confluences were included on the profiles. The profiles were then tested for fit to the function:

$$y = a^{bx}$$

in which y = elevation in feet, x = distance in feet.

The correlation coefficients (R^2) and the values for a and b were calculated. Figures 8-11 are profiles for Fairbanks, Cleary, Dome, and Pedro Creeks and are representative examples of most profiles in the district.

A number of general observations can be made from an analysis of all the profiles:

1) The profiles are very regular with R^2 values greater than 0.90.

2) The maximum stream gradient at the upper limit of major placer formation is 440 ft/mi (for example, a slope of 1/12).

3) The maximum elevation of major placer formation varies systematically, from 1,320 ft on Kokomo Creek in the northeastern end of the district to 600 ft on Cripple Creek in the southwestern extremity. Elevations of major summits and stream divides vary in the same manner.

4) Tributary streams have little effect on the stream profiles.

5) Elevated benches are apparent on some stream profiles, notably at the 1,800-ft elevation on Pedro Creek.

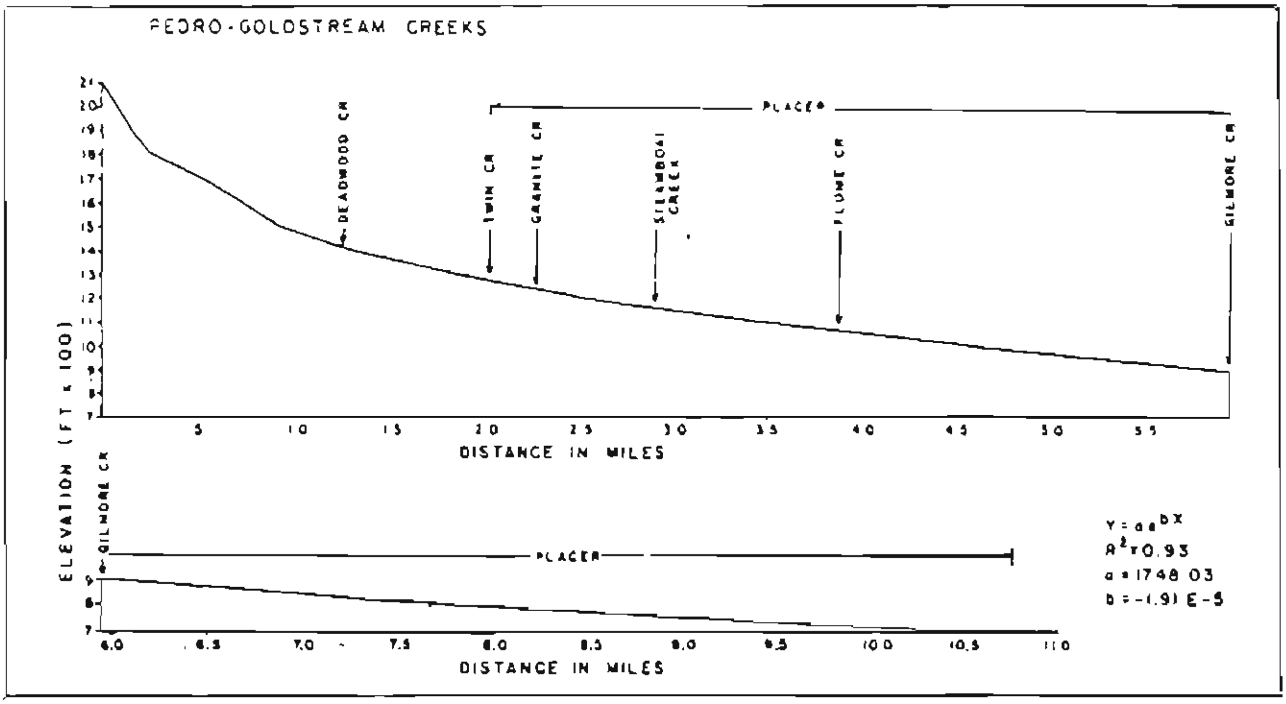

Figure 8. Stream profile for the Pedro-Goldstream Creek drainage system.

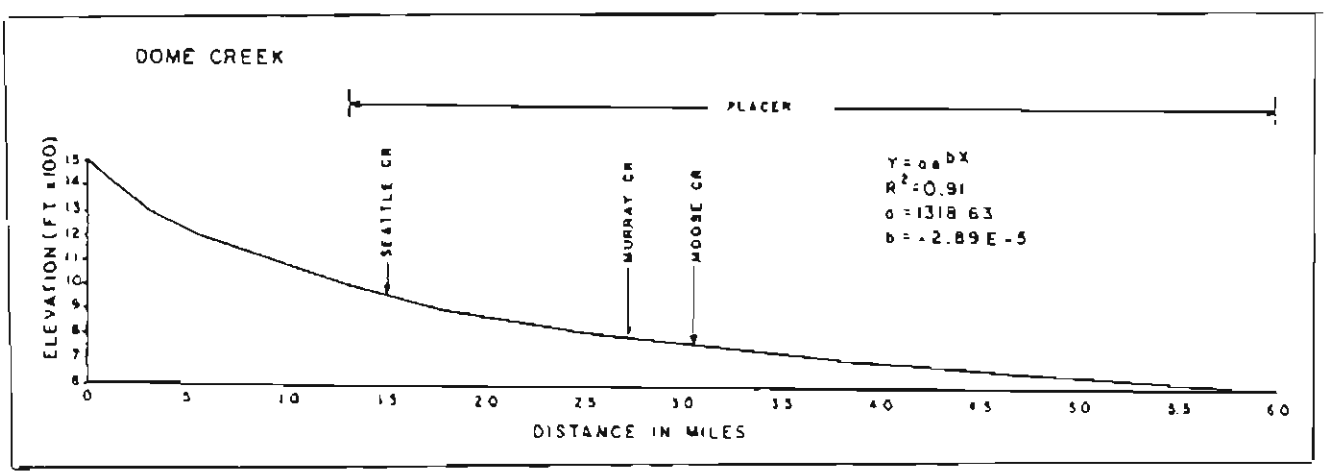

Figure 9. Stream profile for Dome Creek.

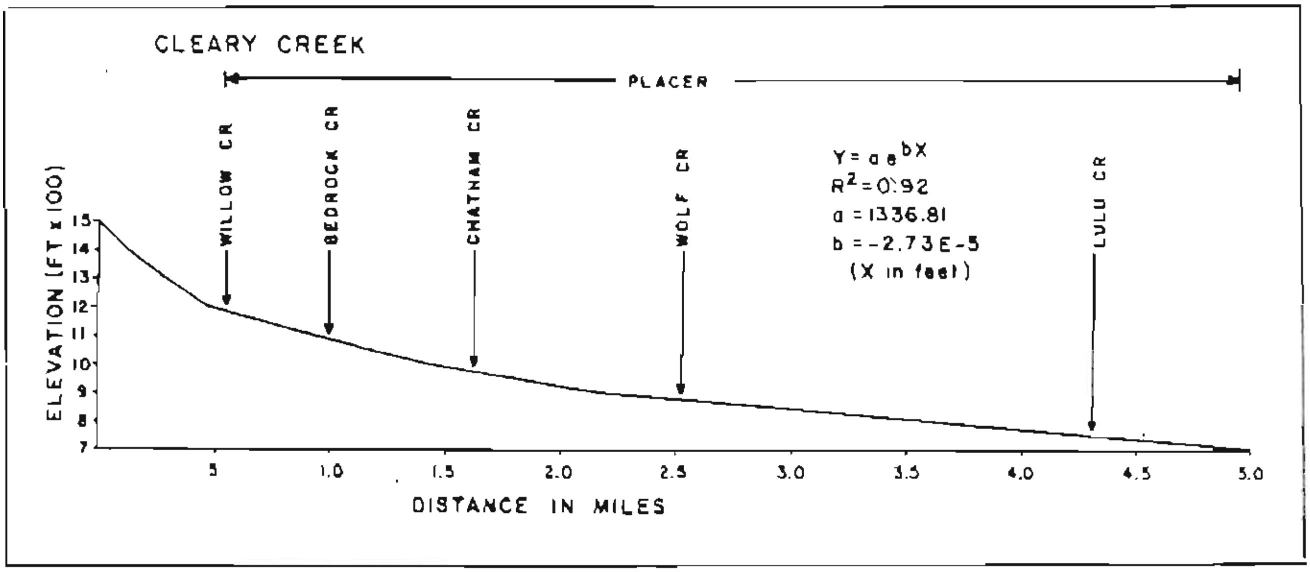

Figure 10. Stream profile for Cleary Creek.

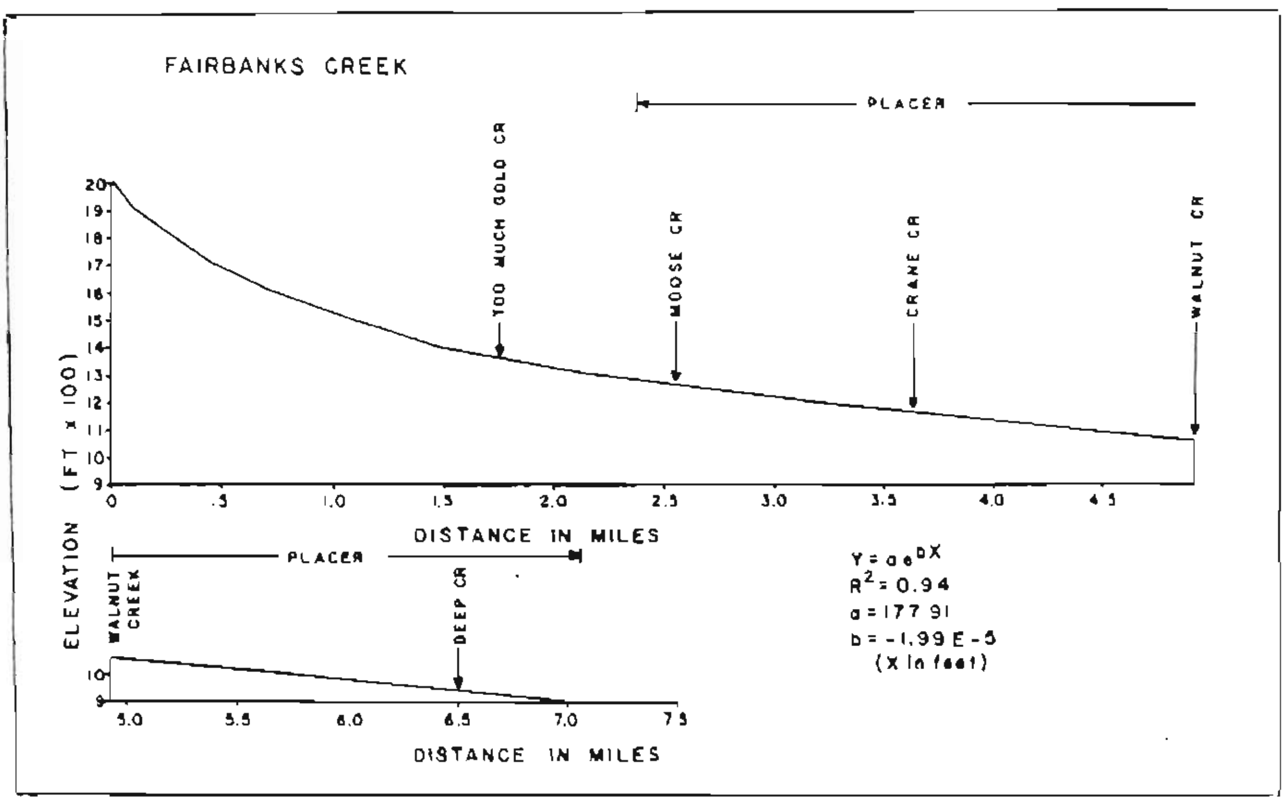

Figure 11. Stream profile for Fairbanks Creek.

6) Stream profiles cannot be used as the sole criterion to predict the presence (or absence) of a major placer deposit. Several streams with profiles similar to productive streams draining areas of gold-bearing metavolcanic rocks do not contain major placer accumulations.

Evidently, the maturity of the stream drainages of the district was one of the necessary conditions for major placer accumulation. This factor should be an essential consideration in the assessment of the placer potential of other mining districts and should be considered when planning future exploration programs in the Fairbanks district.

The maximum elevation of major placers defined in the Fairbanks district is a criterion that should be used with caution in delineating additional target areas. It may be used locally within the district, but the major criterion in assessing placer potential is that of maximum gradient of the stream at the upper point of major placer formation. The reach of the stream at the upper extreme of placer formation probably marks the maximum critical point between degradation and aggradation on the stream. The gradient of 440 ft/mi is a slope of one in 12, which is remarkably close to the average slope adopted by the placer miners for operation of the sluice box.

The sluice has traditionally been the primary gold-recovery system for Alaskan alluvial placer operations. The system is restricted to the recovery of +65 mesh (1 mg maximum, assuming a 4-to-1 shape factor) gold particles. Particles finer than 65 mesh are transported to lower energy alluvial environments just as they are transmitted to tailings in most sluices. Generally, overbank placer deposits on meandering stream and distal beach deposits will have gold particles in the -100/+400 mesh range.

The tributary streams may locally affect the paystreak by shifting it laterally, but their overall effect on the distribution of placer formation is minimal. Tributary channels do not change the stream profiles appreciably at or below the confluence with the main drainage.

The identification of elevated benches in stream profiles is important for two reasons. First, the benches may be important sites of placer formation and second, the main stream channel below benches may be an important site of secondary accumulation.

Because most stream profiles in the district are very similar and because some streams that drain potential bedrock source areas do not contain known major placer accumulations, two inferences can be made. First, such streams may contain major placer accumulations that have not yet been discovered and second, there exist other factors controlling placer accumulation that are not revealed in the stream profile.

Periods of High Discharge and Placer Formation

Without data on contemporary stream discharge rates it is not possible to quantitatively determine whether the present stream profiles formed as a result of stable climatic conditions during the whole period of stream drainage development. Such a determination would add greatly to the interpretation of the paleoclimatic conditions of the late Tertiary and Pleistocene, which were made from the surficial stratigraphic record.

Several of the major placer producing drainages in the district are underfit streams. The best examples are Goldstream and Fish Creeks. Drury's (1965) method of relating valley formative discharge to the meander wavelength-bankfill discharge equation indicates that valley formative discharge for Goldstream and Fish Creeks may have exceeded normal present discharges by 25 to 100 times. Thus, the placer deposits were probably formed during periods of much higher discharge rates.

The Fairbanks mining district was free of glacial ice during the late Pliocene and Pleistocene, yet the placer deposits of the area are associated with the coarse, poorly sorted alluvial Cripple and Fox Gravels. These gravels are much coarser than the material in the current channels (Péwé, 1952). The transport of this coarse material would require significantly higher rates of discharge (Tourtelot, 1968) than those found in modern drainages.

The Cripple and Fox Gravels contain abundant silt, transported as suspended load. Leopold and Maddock (1953) noted the importance of suspended load in the transport of the bed load during flooding, and Cheney and Patton (1967) suggested flooding as a mechanism for placer accumulation on bedrock. Although flooding is difficult to document from the stratigraphic record, Mertie (1937) and Péwé (1975) noted that the late Pliocene and early to middle Pleistocene were periods of rigorous and variable climate in interior Alaska. Changes in discharge rates of streams would accompany this major change in climate. Glaciers north and south of the Yukon-Tanana Uplands produced abundant silt during this time---silt that was transported by wind to the uplands and was subsequently introduced into the stream channels. The scouring of stream channels and the transport of coarse material and gold in the Fairbanks district was probably facilitated by the high discharge rates and abundant silt content of Pleistocene streams.

Alternation of Stream Drainage Patterns and Stream Capture

The existence of older terrace and buried placer deposits led early workers to suggest minor changes in stream drainage due to lowering and raising of the base level. Lowering of the base level was considered to be caused by regional uplift or lowering of sea level; conversely, raising of the base level was attributed to regional subsidence or raising of sea level. Regional changes in base level could account for widespread terrace and buried placers in the uplands; however, it could not account for the following morphological features.

1) Uniform and large local differences in elevation of summits, stream divides, and major placer deposits

2) Asymmetry of stream-valley cross sections

3) Pronounced linearity of some stream drainages

4) Opposing directions of streamflow in adjacent valleys.

As noted previously, the maximum elevation of major placer formation ranges from 1,320 ft (Kokomo Creek) in the northeast to 600 ft (Cripple Creek) in the southwest, a difference of 720 ft exists over a distance of 35 mi. The divides in the Kokomo and Fairbanks Creek area are at an elevation of 2,250 ft; those in the Cripple and Ester Creek area are at 1,500 ft, a difference of 750 ft. Similarly, Twin Buttes in the northeast has a summit of 3,025 ft and the summit of Ester Dome in the southwest lies at 2,364 ft, a change in elevation of 661 ft. These local changes in elevation are twice as large than the inferred changes in sea level since the Pleistocene (Hopkins, 1967).

The asymmetry of stream valleys has been attributed to several factors, including differential slope transport in arctic environments, differential stream erosion due to differences in lithology, and differential headward migration of streams due to increase in bedrock slope (caused in turn by local tectonic activity).

The variation in angle of slope between north- and south-facing cross-sectional profiles in areas of permafrost has been examined by Hopkins and Taber (1962), Curry (1964), Kennedy and Melton (1971), and Kennedy (1976). Since north-facing slopes have less exposure to solar radiation, the slope should be less susceptible to thawing and less subject to erosion by solifluction. South-facing slopes would have increased solifluction because of alternate freezing and thawing, leading to increased erosion and lower slope angle. None of the above investigators were able to demonstrate a simple dependence between slope orientation and slope angle. A simple explanation is even less plausible in the Fairbanks district because, in some major drainages such as Goldstream Creek and the Little Chena River, the south-facing slopes are actually steeper; also, other creeks display little difference in slope angles between north- and south-facing cross-sectional profiles.

Differential stream erosion due to differences in lithology can also be ruled out as an explanation for valley asymmetry in the Fairbanks area. The bedrock in the area is predominantly schist and the regularity of the longitudinal stream profiles indicates the uniform resistance of the bedrock to erosion. (Schist generally exhibits uniform weathering characteristics under varied climatic conditions.)

Cotton (1942) and Lauder (1962) have shown that asymmetry of slope angles can be caused by shifting of stream divides due to local tectonic activity. In extreme cases, the weaker stream drainage will be captured by the more aggressive stream and the result will be a major change in direction of stream flow. Figure 12a shows a stream drainage pattern developed in a schist terrane in a stable craton without major tectonic activity (note the orientation of the arrows indicating stream flow). Figure 12b (after Lauder, 1962) shows the development of a precedent stream along the Wellington Fault, North Island, New Zealand. Tilting of the fault block to the northwest resulted in steepening of slopes down-dip, but the gradient of the precedent stream along the strike of the fault remains constant. The result of the tectonic activity was to increase erosion up-dip, leading to imminent stream capture at P1, P2, and P3. In figure 12c, stream capture is complete at C1, C2, and C3. The former southerly reaches of the precedent stream flow north, but form barbed drainages at B1 and B2 and enter channels that flow to the south (note the orientation of arrows indicating streamflow).

To determine if similar processes were active in the Fairbanks district, figure 7 was examined for the presence of barbed streams. From southwest to the northeast, major barbed drainages include Cripple, Ester Nugget, Sheep, Our, Treasure, Cleary, Kokomo, Fairbanks, and Fish Creeks. The present direction of stream flow is marked by arrows.

Although the Fairbanks area is tectonically active (Gedney and Berg, 1969), the structural control of barbed streams is less obvious in the Fairbanks district than along the Wellington Fault in New Zealand. In the southwest part of the Fairbanks district, a major lineament was identified on Landsat imagery and on aerial photography (Metz and Wolff, 1980). The lineament extends from Cripple Creek along the north side of Chena Ridge to Isabella Creek and French Gulch and on toward Gilmore Dome (fig. 7). The area between Cripple Creek and Isabella Creek is a prominent topographic low. Streams crossing the linear feature from the north are displaced in swampy terrane and do not continue southward to the Chena River.

The divides between Cripple and Rosie Creek and between Isabella Creek and French Gulch suggest wind gaps. Benches along the tributaries of Rosie Creek suggest a larger protodrainage to the west of the current drainage. The earlier drainage system probably included the upper reaches of Cripple Creek and flowed to the southwest, directly into the Tanana River. The valley of Isabella Creek is very wide and swampy, suggesting the presence of an earlier, larger drainage basin. Recent geological mapping has resulted in the discovery of a ½-mi-wide shear zone in a road cut at the divide between French Gulch and Isabella Creek. The shear zone strikes northeast-southwest, is congruent with the Landsat linear feature, is near vertical, and is here designated the Cripple Creek-French Gulch Shear. This shear zone was probably the cause of the capture of both Cripple Creek and Engineer Creek.

Movement along the Cripple Creek-French Gulch Shear caused a relative rapid headward migration of the proto-Engineer Creek, resulting in the capture of the upper reaches of Isabella Creek and reversal in the flow direction of French Gulch. The reworking of the stream sediments resulted in the formation of a very high grade placer in lower Engineer Creek as well as the deposition of the resistant quartz-rich white facies of the Cripple Gravel at the mouth of Engineer Creek.

Metz and Wolff (1980) noted a major Landsat linear feature extending from the east side of Ester Dome in a northerly direction for about 30 mi. This linear feature offsets the northeast-southwest trending regional structure and associated aeromagnetic anomalies and is parallel to a major north-south striking normal fault system in the Yukon-Tanana Uplands, including a large structure 20 mi to the west in the Minto Flats area. The Minto Flats structure dips to the west and has a vertical displacement of over 2,000 ft

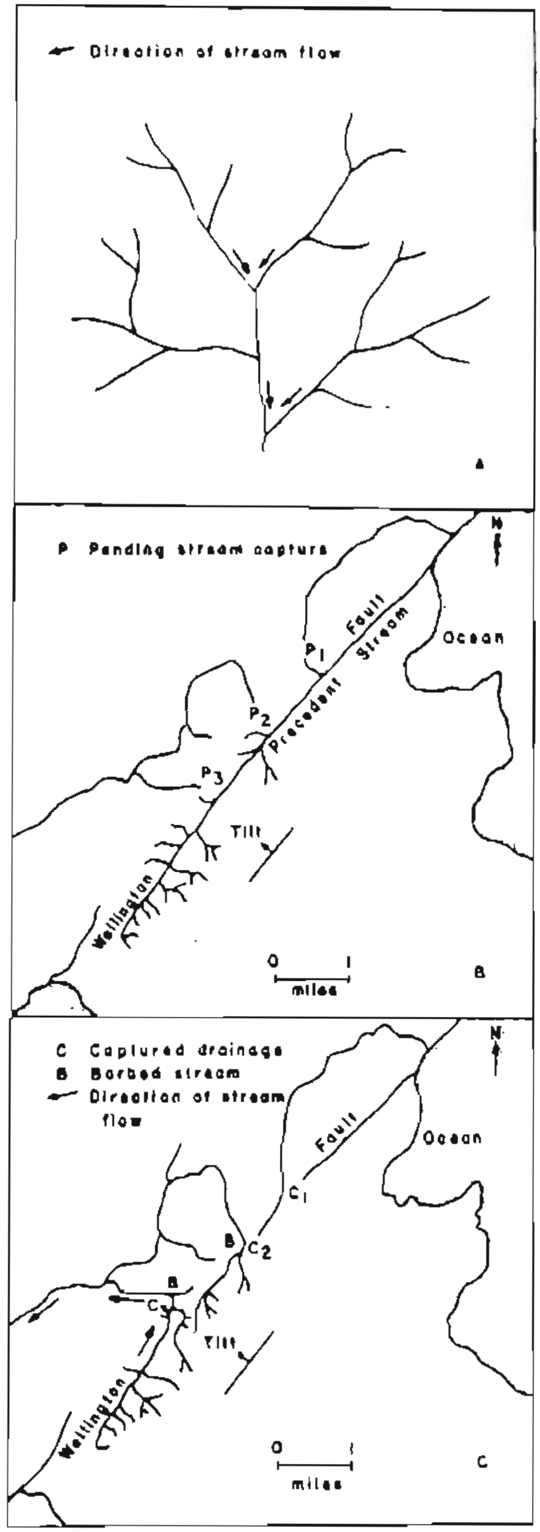

Figure 12. Stream flow directions in (a) hypotentical tectonically stable schist terrane, (b) tectonically active schist terrane before stream capture, and (c) tectonically active terrane after stream capture (figs. 12a and 12b after Lauder, 1962).

(Barnes, 1961). The Minto Flats fault, which can be traced for about 80 mi, appears to offset Pleistocene eolian deposits in the Tanana Valley.

Surface evidence for a major shear zone in the Ester Dome area includes fault breccia in placer tailings on Sheep Creek and the predominance of lower grade metamorphic rocks west of the linear feature. The lack of marker horizons in the schist sequence precludes definitive determination of fault motions in the Ester Dome area. In the Grant mine near Happy Creek, northeast-trending gold quartz veins are truncated by a north-south striking fault zone; however, the extension of the truncated veins have not been located and no offsets can be calculated. Slickensiding suggests both dipslip and strikeslip components. This fault system is congruent with the Landsat linear feature and is designated the Moose Creek Shear (fig. 7).

Placer drill records for the Sheep Creek and Goldstream area define a bedrock high at the confluence of the two creeks. To the east bedrock drops off rapidly, whereas the slope is less steep to the west. This bedrock high is an expression of an earlier stream divide. Movement along the Moose Creek Shear caused increased headward migration of the proto-lower Goldstream Creek to the divide and, ultimately, capture of proto-upper Goldstream Creek, which previously had flowed southward to the Chena River. Moose, Sheep, Happy, St. Patrick, and O'Conner Creeks were all previously flowing to the east by southeast to the Chena River via proto-upper Goldstream Creek. Stream capture not only caused resorting of the Sheep Creek placer but affected the hydraulics of the entire Goldstream drainage, resulting in the formation of the largest single placer in the region.

In the northern and northeastern part of the district, two previously mapped faults (Forbes and others, 1968), the Chatanika Valley Fault and the Cleary Creek Thrust, have affected the drainages of Treasure, Vault, Dome, Little Eldorado, Cleary and Kokomo Creeks. Cleary Creek, which produced 1,000,000 troy oz placer gold, provided the best example of structural control of stream drainage in this part of the district. The upper reaches of Cleary Creek flow to the northeast while the Chatanika River, which is the main drainage system in the area, flows southwest. Willow, Bedrock, and Chatham Creeks, all tributaries of Cleary Creek, originally drained directly into the Chatanika. Relatively faster headward migration of the proto-lower Cleary Creek along the Cleary Creek Thrust caused beheading of Willow, Bedrock, and Chatham Creeks.

Similarly, the Fairbanks Creek and Fish Creek drainage system forms a major barbed reach to the Little Chena River. The tributary streams flow east by northeast, whereas the Little Chena River flows southwest (fig. 7). Recent geologic mapping in the Cleary Creek and Fairbanks Creek area has resulted in the discovery of a major east-west shear zone that appears to control the gold-quartz vein mineralization in this part of the district. The shear zone dips to the south at 45° to 70° and exhibits both strike-slip and reverse-fault movements. The structure has been mapped at several lode deposit localities over a 5-mi strike length, whereas placer drilling in the Fairbanks and Deep Creek area suggests that it may extend for another 4 mi east. This structure is designated the Fairbanks Creek Shear.

A second major structure has been mapped along the valley of the Little Chena River (F. Weber, USGS, personal commun.). This structure strikes northeast-southwest, controls the lower reaches of the Little Chena River Valley, and is visible as an aerial and Landsat Linear for at least 50 mi (Metz and Wolff, 1980).

Movement along these structures apparently gave rise to the relatively rapid headward migration of Fish Creek, which resulted in the beheading of Fairbanks, Bear, and Solo Creeks. Prior to stream capture, these tributaries flowed southeast directly into the Little Chena drainage. These captures caused resorting in the tributaries and the formation of the paystreaks in upper Fish Creek and Fairbanks Creek. The reaches of Fish Creek below the confluence of Solo, Bear, and Fairbanks Creeks should be areas of potential placer accumulation. Several small, recent mining operations on lower Fish Creek below Fairbanks Creek have proven the existence of economic concentrations of placer gold in this area.

Previous descriptions of the relationship between placer formation and stream capture are very limited. Cox (1879) described lode-gold occurrences in the schist terrane adjacent to the Wellington Fault in New Zealand. He also noted the occurrence of placer gold in the streams that Cotton and Lauder later described as captured drainages; neither Cotton nor Lauder considered the possibility of a relationship between stream capture and placer-gold accumulation.

Mertie (1918b) described the change in flow direction of Livengood Creek in central Alaska and noted that the stream system contained the most significant placer deposit in the Tolovana district. During examination of the Goodnews Bay placer platinum deposit, Mertie (1976) provided evidence for the capture of Platinum Creek but did not seem to recognize the significant part played by stream capture in the localization of the placer.

Smirnov (1976) discussed the importance of tectonic processes in placer formation; however, he did not analyze the role of individual processes in the localization of placer deposits.

A consideration of figure 7 demonstrates that most of the major placer deposits in the Fairbanks district lie in stream systems that exhibit barbed reaches. Therefore, the reworking of gravels caused by alteration of drainages and changes in stream-flow direction must be the predominant control of placer formation in the district.

Placer Gold Fineness Value

Metz and Hawkins (1981) discussed the regional distribution of gold fineness values from Alaskan placer deposits and reviewed the significance of local difference in fineness values from placer and lode deposits. Desborough (1970) and Forbes (1980) explained local differences in placer gold fineness values by demonstrating the existence of silver depletion rinds on nuggets from Alaskan alluvial deposits. The existence of such rinds is attributed to the greater solubility of silver relative to gold in the alluvial environment. Increasing the thickness of the rind by increased transport in the alluvial environment (or increasing the relative thickness of the rind by decreasing grain size downstream) is used to explain increased gold fineness away from the lode source.

Gold fineness values (Au/(Au + Ag)) x 1,000 from both Smith (1913b) and this investigation are shown on figure 7. In addition, the fineness values for placers on Pedro-Goldstream, Dome, Cleary, and Fairbanks Creeks were plotted as a function of distance downstream from probable lode sources (figs. 13-16). The figures include the location of the tributary streams and the fineness value of tributary placers. The fineness value-distance plots were then fitted to a polynomial function and regression coefficients (R^2) values were calculated.

In the Pedro-Goldstream system (fig. 12), gold fineness increases 20/mil within the first ½ mi below the Rainbow mine at the site of the first major alluvial workings. The fineness changes by 60/mil over the remaining 10.5-mi extent of the placer workings. The gradients are thus 40/mil per mile up to the alluvial deposits and 6/mil per mile in the placer environment. The regression coefficient for the polynomial is 0.94, indicating a relatively predictable change in fineness.

Fineness values increase below the confluences of Granite, Flume, and Gilmore Creeks with the main drainage system. These increases reflect the higher fineness of the tributary streams and may give a semiquantitative value for the relative contribution of gold from the tributary to the main channel. In the case of Granite Creek, the change may indicate the presence of a significant placer in the tributary or a new lode source of higher fineness.

The Dome Creek plot (fig. 14) shows an increase in fineness of 30 per mil in the three-quarter mile distance from the Soo mine, a change of 40/mil per mile. The change in fineness for the remainder of the placer deposit is 6/mil per mile and is very uniform with $R^2 = 0.97$.

The placer-gold fineness values for Cleary Creek (fig. 15) are much less regular with $R^2 = 0.87$. This is a reflection of the multiple lode source areas on the ridge between Cleary and Fairbanks Creek (fig. 7). The fineness values increase 23/mil in the first ¼ mi below the Cleary Hill mine, or 92/mil per mile, whereas in the rest of the drainage the values increase by an average of 7/mil per mile. The expected value below LuLu Creek is 879/mil, whereas the actual increased value is 861/mil, thus suggesting an alternate lode source area.

The plot for Fairbanks Creek (fig. 16) shows an increase in fineness of 78/mil per mile from the Hi-Yu mine to the first major placer. The fineness only increases 5/mil over the remaining 5 mi of the deposit. The tributaries of lower Fairbanks Creek either have fineness values comparable to the values in the main drainage or have not made a significant contribution to the total volume of placer gold. The change in fineness values from the McCarty mine (814 fine) on upper Fairbanks Creek is 42/mil per mile from the mine to the first placer (fig. 7). The change is similar to the rates of change noted for the other drainage systems.

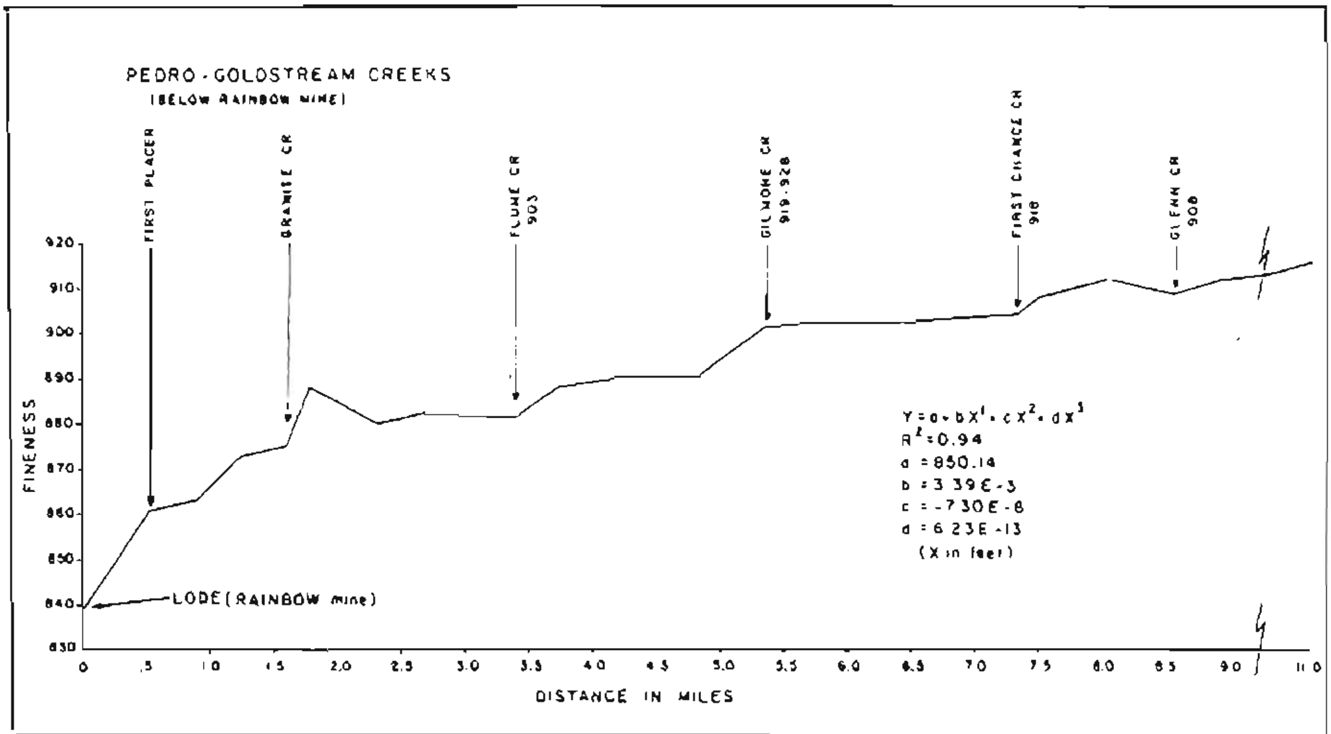

Figure 13. Placer-gold fineness vs. distance plot for Pedro-Goldstream Creek drainage systems.

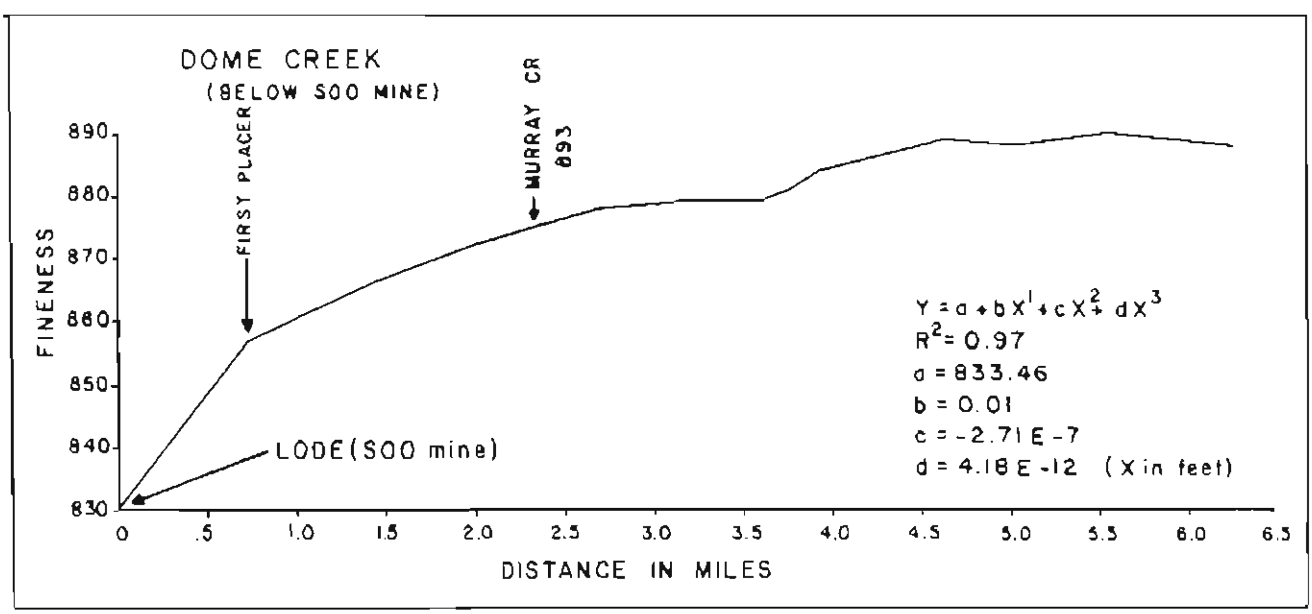

Figure 14. Placer-gold fineness vs. distance plot for Dome Creek.

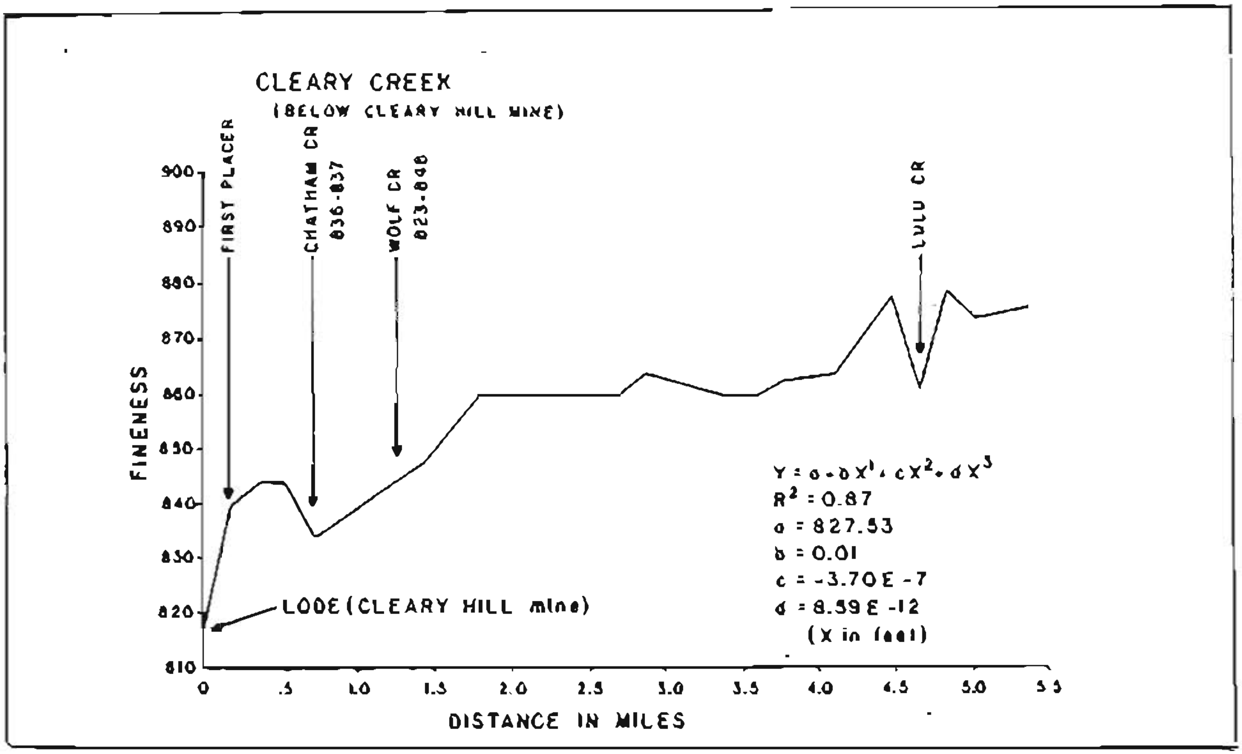

Figure 15. Placer-gold fineness vs. distance plot for Cleary Creek.

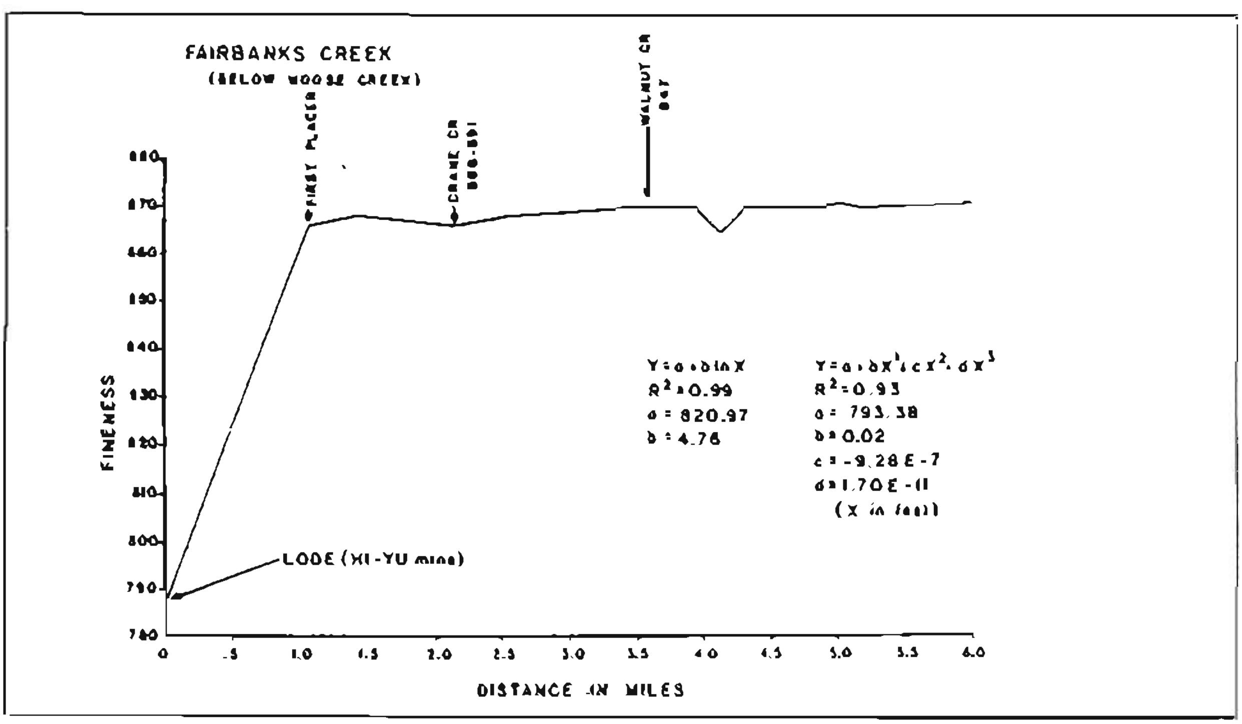

Figure 16. Placer-gold fineness vs. distance plot for Fairbanks Creek.

These data suggest that the predominant change in alluvial gold fineness takes place in the weathering and erosional environment rather than in the placer depositional environment. Because the change in gold fineness value in the alluvial environment is small, these changes may not reflect change in flow direction caused by stream capture. Gold fineness values thus must be used with caution in placer exploration and evaluation.

SUMMARY AND CONCLUSIONS

Placer deposits of the Fairbanks district occur in streams that drain areas of metavolcanic rocks of the Cleary sequence of the Yukon-Tanana Uplands Schist. Two distinct types of lode sources have been identified, each with different gold grain-size distributions. Premetamorphic sulfide lenses contain fine-grained gold (<1 mg), whereas postmetamorphic gold-quartz-sulfide veins contain fine- to coarse-grained gold. Commercial gold placer operations to date have been limited to proximal high-grade gold accumulations. Fine-grained gold from fine premetamorphic bedrock sources would be transported greater distances and may have accumulated in distal terrace or buried placer deposits.

The surficial depositional controls leading to formation of commercial placers include:

1) Deposition in streams with regular gradients reflecting stream maturity and sufficient time for placer formation.

2) Placer values are concentrated on or near bedrock in reaches of streams with gradients less than 440 ft/mi.

3) Placer formation took place under conditions of high discharge rates relative to those found in the present stream systems.

4) Placer gold fineness values increase systematically downstream; however, rates of change are greatest in the weathering and erosional environment rather than in the placer depositional environment.

5) Most of the major placer deposits occur in streams with barbed drainages. Several of these barbed drainages can be related to basement structures. Stream capture and the alteration of stream flow by basement structure was the most important factor in producing economic concentrations of gold in the placer deposits.

ACKNOWLEDGMENTS

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GEOPHYSICAL EXPLORATION FOR PLACERS AND OTHER VALUABLE MINERAL DEPOSITS

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Geophysical techniques can be very beneficial during both placer exploration and mine development. Geophysical techniques can help to determine:

1) Thickness and character of overburden

2) Depth, configuration, and nature of bedrock surface

3) Depth and configuration of groundwater surface

4) The existence, depth and configuration of ground ice

5) The subsurface location of magnetic minerals.

Geophysical techniques have several advantages in delineating subsurface conditions over conventional drilling and digging. Geophysical techniques do not disturb the ground, is fairly portable (about the size of a suitcase), and can survey areas inaccessible to drilling. Geophysical surveys can be performed by a minimum of personnel in a relatively short time (generally within a few days). In addition, geophysical surveys are relatively inexpensive.

The more common geophysical techniques---seismic refraction and reflection, resistivity, magnetic surveys, conductivity, and gravity surveys---are discussed below, along with their applicability to placer mining and average cost.

Seismic Reflection and Refraction

In seismic surveys, a seismic wave is generated in the earth by detonating explosive charges or simply by striking the ground with a sledge hammer. A string of sensors (geophones), placed to record the timing of the propagation of the seismic wave produced, can provide information on the depth to bedrock, the configuration of the bedrock surface, and valleys and channels in the bedrock. Under favorable conditions, these surveys can also provide information on the type of bedrock, type of buried sediments, location of the groundwater table, and even depth and thickness of ground ice. Seismic surveys are hampered by rain and wind, which can create surface noise that can interfere with the survey; permafrost or frozen terrain can also inhibit data interpretations.

A two-man seismic crew can cover 1,000 line ft of survey per day, depending on vegetation cover and topography. Estimated field costs are about $2,000/day; data interpretation costs about $1,000/day. The total cost of a seismic survey averages from $3 to $5 per line foot.

Magnetic Surveys

Ground magnetic surveys, commonly used in placer exploration, measure the magnetic signature of the subsurface materials. The most common magnetic mineral is magnetite, although chlorite, platinum, and even some types of garnets may also register on the magnetic survey. Because gold and magnetite commonly occur in roughly proportionate amounts, magnetic surveys help delineate paystreaks. This conclusion was first documented by Henry Justin of the Alaska Territorial Department of Mines in 1939.

When performing a magnetic survey, the fluctuation of the earth's magnetic field must be monitored and the changes in it must be removed from the data to obtain the exact magnetic signature of the subsurface materials. The earth's magnetic field can be monitored by a stationary self-recording base station.

In some cases, a magnetic decrease can suggest a possible deposit that may, for some reason, have a low magnetic signature.

Magnetic surveys work only if there are magnetic minerals in the placers; they cannot yield information on the bedrock unless the pay channel occurs directly on top of it.

Magnetic surveys usually require only one operator, although 2- or 3-man crews increase the productivity. At 25-ft intervals, 2 to 3 mi of lines can be surveyed per day. In open terrains, up to 15,000 line ft can be surveyed in a day. Thick brush can limit surveys to 2,000 line ft/day. Magnetic surveys cost $600 to $1,200/day.

Resistivity

In galvanic resistivity surveys, two electrodes are placed into the ground and an electric charge is applied to one of them. The resultant measured potential provides information about the properties of the materials that the current is transmitted through.

Because various size fractions of materials have differing resistance to electrical currents, the size of the subsurface sediment particles can be interpreted. Groundwater and ice can also be delineated, making resistivity surveys suitable for permafrost terrains.

Interpretation of resistivity information is difficult, although specially designed computer programs are a great help.

Resistivity studies are time-consuming and relatively expensive because of the length of setup time and the difficulty of interpretation. A resistivity survey can cover 100 to 500 line ft/day at a cost of up to $1,500/day. The time required for interpretation can be almost as long as the length of time required for the survey. However, resistivity surveys provide excellent information, particularly when combined with seismic surveys.

Conductivity (Electromagnetics)

Conductivity, or electromagnetic, surveys involve producing an electrical current through a coil. The coil is run over the ground and another coil with a detector trails it, measuring the potential induced into the ground by the current. Electromagnetic surveys, which have been used in locating mineral deposits located under considerable glacial overburden, can be used to delineate subsurface structures and conductive surfaces in bedrock, overburden, and pay channels.

Electromagnetic surveys are relatively inexpensive (average of $1,000/line mi) and can be used in permafrost terrains.

Gravity

Gravity surveys have been used to delineate the concentrations of high-density materials and groundwater channels. Gravity surveys can be performed quickly and inexpensively, and data interpretation is not as difficult as other geophysical techniques. Gravity surveys have been used successfully in the iron industry, although they are not particularly useful in gold exploration.

Conclusion

Geophysical surveys can provide valuable information not only in exploration for placer deposits but for delineating pay channels, locating targets for drilling, and planning the overall design of the mining operation.

Of the geophysical techniques that have proved useful to placer exploration, conductivity and magnetic surveys are the most inexpensive. However, these require special circumstances (conductive surfaces for electromagnetics and the presence of magnetic minerals for magnetics). Magnetic surveys can be done in all terrains and during all seasons.

Resistivity and seismic surveys are comparatively expensive and involve some seasonal limitations. Interpretation of resistivity data is expensive and generally requires specially designed computer programs. However, when combined, resistivity and seismic surveys can provide excellent information on the subsurface conditions.

COUNTERCURRENT SLUICING ON KETCHEM CREEK

by

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INTRODUCTION

The Ketchem Creek placer mine has been in operation for several years. In 1986, the Department of Environmental Conservation-Department of Natural Resources awarded a placer-demonstration grant to test the efficiency of a countercurrent sluice, or vanner, at this mine.

The stream gravels in the Ketchem Creek Valley overlie weathered granite bedrock. The white decomposed bedrock is readily identifiable from the stream gravel. Several feet of the uppermost gravel is stripped, along with the vegetative cover and silt. A foot or more of the underlying bedrock is sluiced, along with the lower 6 to 8 ft of gold-bearing gravel. Enough bedrock is taken to ensure that gold is not left behind.

As the amount of decomposed granite bedrock increases, the proportion of sand-size material increases greatly. The decomposed granite is almost half sand-size particles. The clay and boulder content of the gravel does not impede washing and mining.

THE TROMMEL

Our trommel, first built in 1985 (fig. 1), was modified from a design presented by Joe Vogler at an earlier Placer Mining Conference. It is 8 ft, 9 in. in diameter and 27 ft long. In the first 17 ft are caterpillar D-9 track rails, which act as retainers. The rails produce an excellent scrubbing action. The lower 7 ft has rectangular 2-in.-high hardened bars spaced 2 in. apart, giving a 50 percent open area.

In the first part of the grant project, before operation of the countercurrent sluice, the trommel had classified to 2 in. To add classification further, a 3/4-in. screen (fig. 2) was attached to it. Welded concentrically around the 2-in. bars, the screen made this a true two-stage trommel classification system.

Oversize rock, from both the 2-in. and 3/4-in. fractions, falls into the same tailings pile. Tailings from both the wash plant and the trommel are bailed out of a small pond. Water flows out of the small pond through the tailings race into a large settling pond.

The trommel is powered by an 85-kw Caterpillar generator. It is driven by a 50-hp electric motor through a Dodge shaft-mounted gearbox with a 25:1 gear ratio. The gear box is mounted between two pillar blocks. The sump

Figure 1. Trommel washing plant and sluice box before Placer Demonstration Grant project, Ketchem Creek mining operation.

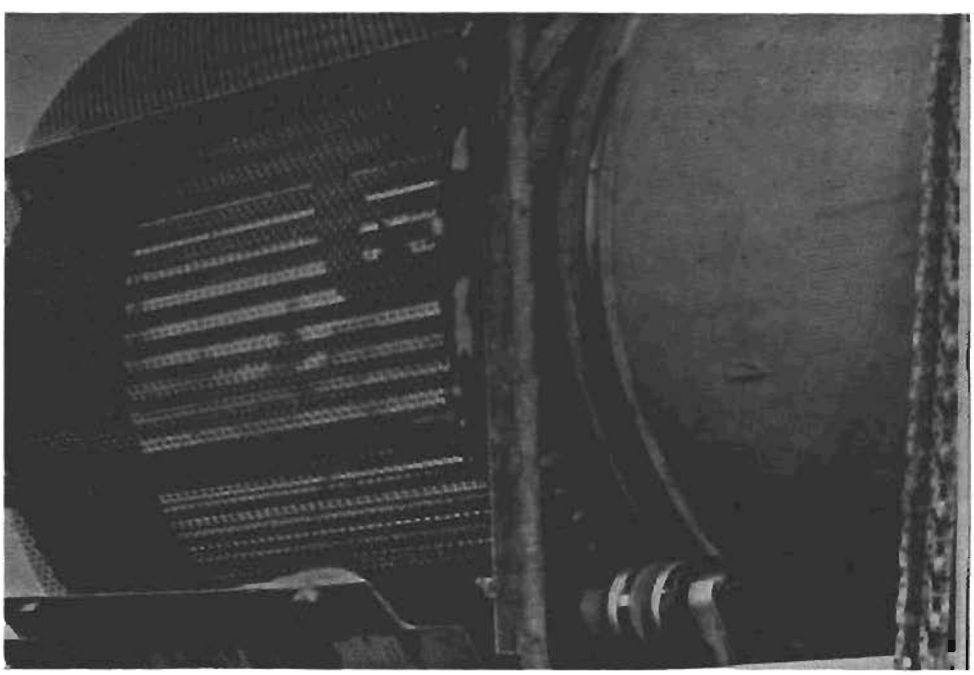

Figure 2. Trommel with second-stage 3/4-in. screen used during grant project.

drive chain is a Caterpillar D-3 sealed and lubricated track chain that surrounds the barrel.

The trommel is cradled on D-9 rollers that ride on a 3-in.-high hardened bar stock. A thrust roller is not needed. The rollers oscillate in a bogie system, doubling the bearing surface.

The trommel revolves at 8 rpm, which we feel is faster than necessary. It will be slowed to 6 rpm with the installation of a slower motor this year.

PLAN OF OPERATION

Water is pumped downstream with an 8-in.-diam Crisafulli pump, which is supplemented with a 6-in.-diam pump that recirculates water from the downstream pond. About 2,000 gpm is used by the washing and sluicing plant.

Stripping and stock piling of gravels is done with a single D-9 Caterpillar. The stockpiled gravel is fed to the washing plant with a 245 excavator. Tailings from the trommel and sluice are bailed from a wet sump, or presettling pond (along with a significant part of the fines), by a Manitowe 3600 dragline, which has 83 ft of boom and a $3\frac{1}{2}$-yd^3 bucket.

The sluice handles 100 to 135 yd^3/hr. With classification to 2 in., the trommel could be fed about 200 yd^3/hr (70 percent minus 2 in.), depending on the amount of fine granular bedrock in the feed. An excess of fines would overload the 64-in.-wide single-section sluice box; sandbars would build up and feed rates would have to be reduced.

With classification to 2 in., the sluice encountered notable riffle packing. With additional classification and washing to 3/4 in. and the use of a hydraulic lift, the riffles stayed open and unpacked much longer.

The riffle packing produced a loss in gold recovery and consumed operating time for cleanups. To solve this, the self-cleaning countercurrent sluice (fig. 3) was designed and built, partly with the Placer Demonstration Grant.

COUNTERCURRENT SLUICE

Gold loss from a sluice box is known to increase as the sluicing time between cleanups increases. The State of Alaska grant program for innovative fine-gold recovery partially funded the development of a continuously self-cleaning sluice.

The countercurrent sluice, or 'vanner,' is a corrugated sidewall conveyor belt that travels upstream while minus 3/4-in. gravel flows down and across the riffled belt surface. Lighter tailings wash down the belt and off into a wet sump (presettling pond). Gold and other heavy minerals are cleaned from the belt by spray jets at the belt turn at the upper end.

The riffle surface for the conveyor was carefully tested in 1985. Unfortunately, it was tested at a reduced (laboratory) scale. Full-scale test-

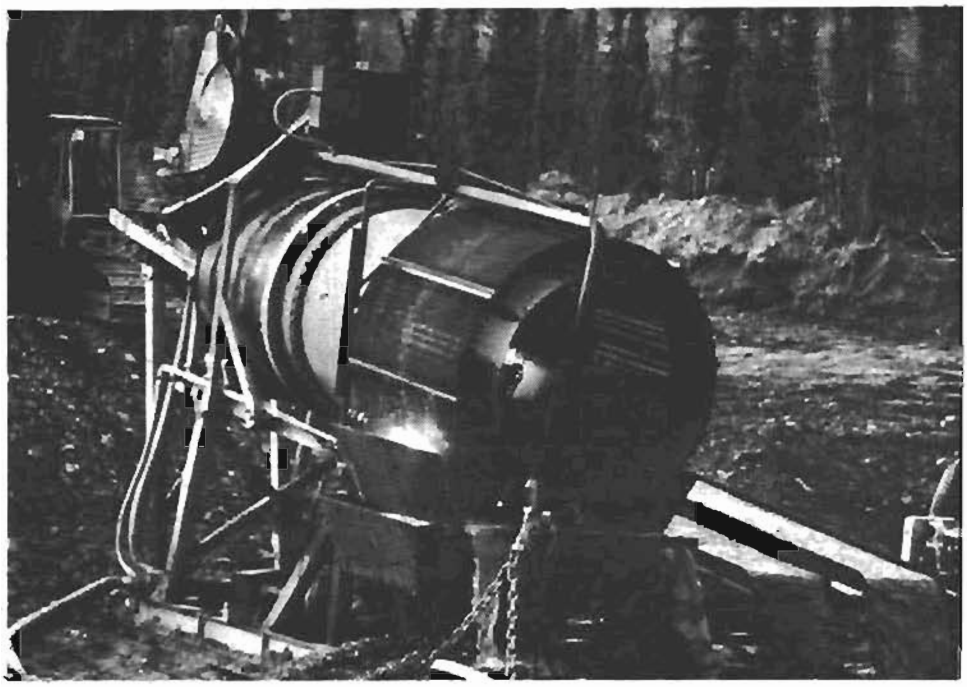

Figure 3. 'Vanner' plant in operation, Ketchem Creek mining operation.

Figure 4. Slick plate with slotted edge feeding vanner belt. Note corrugated belt and rubber riffles.

ing achieved very poor recovery, particularly in the coarser size fractions; this was not expected.

Mechanically functional, though in need of a new riffle design, the vanner was fed through an 8-in.-diam gum-rubber line by a hydraulic lift. The lift was powered by an Ash 6-in.-diam rubber-lined sluicing pump, which may not have functioned perfectly either but was available for the project. The pump injected water, under pressure, into the jetel pump (hydraulic lift) that scavenged minus 3/4 in. gravel from the bottom of a tapered hopper.

The material was split into two outlets, run across a short slick plate and dropped about 1 in. into the rubber riffles of the countercurrent sluice belt (fig. 4).

Once on the belt, the gold was supposed to settle into rubber riffles and then be washed into a hopper as the conveyor belt turned over. Gold-bearing concentrate then passed across one of two Willins jigs. Tailings from the jigs passed through a 12-in.-long tom with indoor-outdoor carpeting as a final recovery check. Concentrates from the jigs could be tapped daily.

The belts were driven by a 3/4-hp variable-drive motor at a speed of 1 to 10 ft/min. The motor was coupled by a worm drive and a chain drive to the head pulleys of the conveyor.

The vanner belts were adjustable 1¼ to 1-5/8 in./ft.

When testing confirmed losses in the coarse gold size fraction, the vanner machine was set aside and the sluice box was returned to operation (fig. 5). This allowed the sluice to be compared to the vanner under identical conditions. Notable reduction in riffle packing were discovered with the additional classification to 3/4 in. and washing from the hydraulic lift.

The most notable feature of the improved sluice box is the individual mounting of the riffles (fig. 6). This sluice box is easy to disassemble and clean. Because of the speed of disassembly, the time between cleanups is reduced, which in turn reduces losses due to riffle packing and loading with heavy minerals.

CONCLUSION

As tested, the countercurrent sluice needs an improvement in its riffle design to become an efficient gold-recovery plant. The sluice box to which the vanner was compared, however, was extremely efficient (over 98 percent recovery). The efficiency of the conventional sluice was attributed to the additional classification from 2 in. to minus 3/4 in., the even feed, and the added washing with the use of a hydraulic lift.

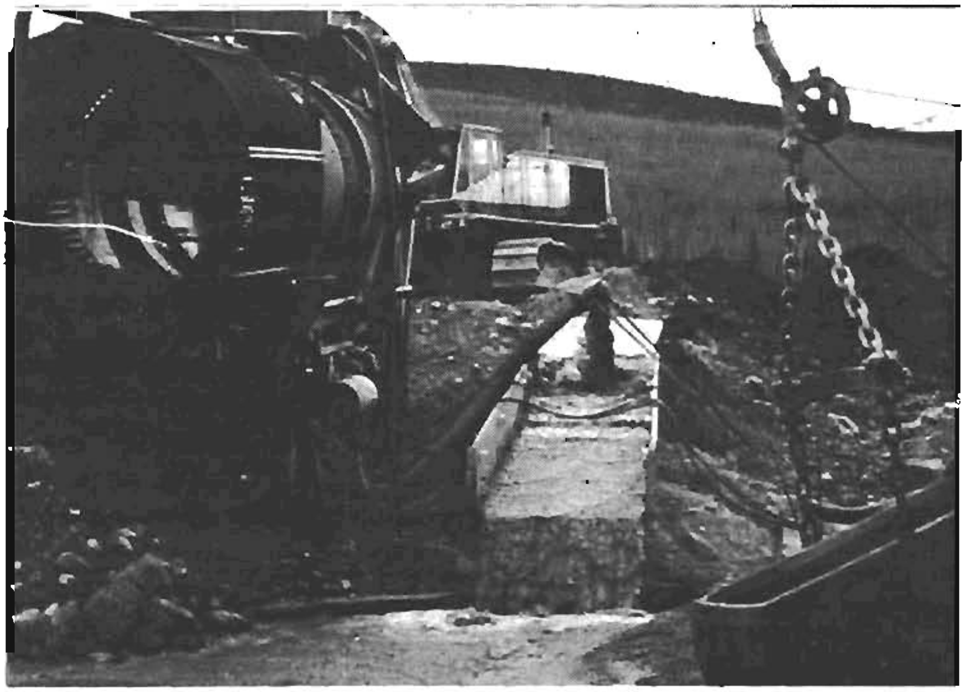

Figure 5. Trommel washing plant and sluice after grant project, Ketchem Creek mining operation.

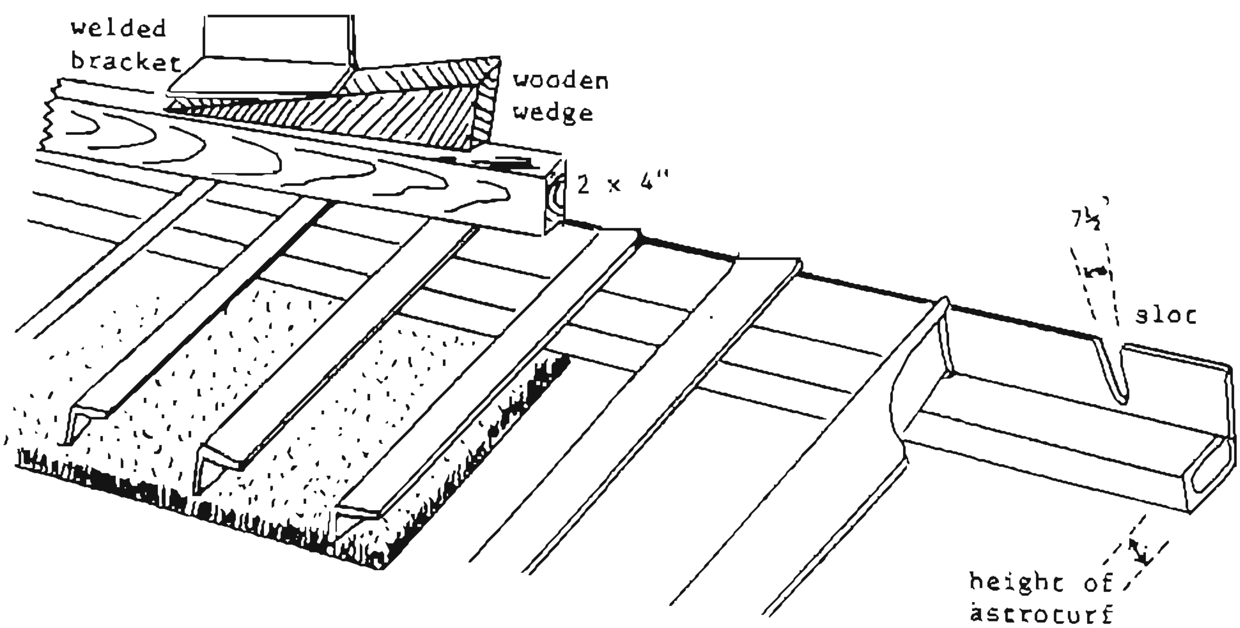

Figure 6. Individually mounted riffles.

DRILLING AND BLASTING IN PERMAFROST AT THE WILBUR CREEK MINE

by

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ABSTRACT

Because of regulatory constraints, the operator on Wilbur Creek is attempting to use modern underground-mining methods at his placer mine, a deeply buried bench deposit. Though drilling and blasting in permafrost has been studied elsewhere, its application to placer mining is not common in this country, and study of the methods may aid in the development of similar deposits. Small-diameter blastholes are drilled with a single-boom hydraulic jumbo. The machine can drill to an 8½-ft depth. Tovex 220 water gel in 1½- by 16-in. sticks is used to initiate a pneumatically loaded ammonium nitrite fuel oil (ANFO) product.

INTRODUCTION

Mining Cycle at Wilbur Creek

The winter mining cycle at Wilbur Creek follows the standard procedure of drilling, loading and blasting, followed by mucking of the blasted material (figs. 1-3). The muck is loaded with a scooptram into a truck and hauled to a stockpile area for spring cleanup.

Though the operator has been developing and mining this ground for a number of years, this method differs substantially from the old method of hydraulic removal of overburden material, followed by sluicing of the pay gravels.

While the new method may prove more costly than the old method prior to enaction of water-standard regulations, it has been shown to be more cost effective than surface 'hydromining' while meeting water-quality standards (Skudrzyk and others, p. 89).

Ground Conditions at Wilbur Creek

In all the underground development at Wilbur Creek the bedrock consists of a graywacke in nearly vertical layers that strike northeasterly (Skudrzyk, 1985). There are two horizons of gravel, each typically 3 to 5 ft thick, separated by a silt layer about 4 ft thick. Gold is produced from the lower gravel and upper few feet of bedrock. The remaining 70 to 120 ft of overburden consists of silt and organic matter and a considerable amount of ground ice and wedges, lenses, and irregular masses (Skudrzyk, 1985).

Figure 1. The jumbo drill, drilling a single hole, Wilbur Creek, Livengood.

Figure 2. Clearing and pumping explosives into drill holes, Wilbur Creek, Livengood.

Figure 3. Underground front-end loader. Rubber-tired equipment was used to muck and haul material from the underground mine.

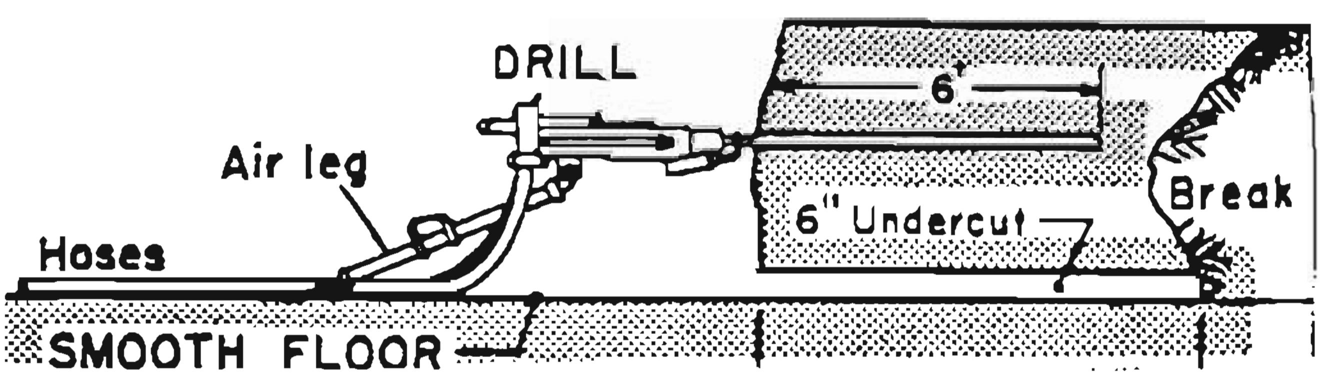

Figure 4. Airleg drill (from McAnerney, 1967).

Choice of Drilling Machines

With the prospect of beginning underground operations, an operator is confronted with a basic choice of two types of drilling machines. A jackleg drill is a hand-operated piece of equipment and represents a smaller capital investment, whereas a hydraulic jumbo offers automatic controls, easy operation, and faster drilling rates, but at a much higher initial cost. (Some experienced operators of jackleg drills feel they are capable of matching or exceeding the drilling rates achieved at the mine at Wilbur Creek; this may well be true, for the jumbo often requires maintenance repairs.)

A jackleg drill (fig. 4) is commonly operated with water used as the drilling fluid to cool and lubricate the drill bit and to remove the cuttings. Jackleg drilling can also be performed dry (Dick, 1970), with frequent stops to blow cuttings from the borehole. However, this creates rock dust in the mine atmosphere, complicating ventilation requirements, as well as health and safety considerations.

With an automatic jumbo, an operator is faced with a second option: use of water or air as the drilling fluid.

Machines using water as the drilling fluid are more accepted by underground miners, are more available, and produce no dust while drilling. However, they create other problems when drilling frozen gravels. The foremost problem with water is that it melts the matrix of the frozen ground, enlarging the size of the borehole and allowing pebbles (often larger than the hole diameter) to block the borehole. (Other problems with water are discussed later.)

In the past, machines using high-pressure air as the drilling fluid have been viewed with jaundiced eye by the miners operating them because of the problem of dust control. However, improved dust-collection techniques may make this type of machine a viable choice for drilling in frozen gravels; such a drill would produce a smoother, cleaner borehole, making it easier to blow clear and load explosives.

The machine in use at Wilbur Creek is a Secoma ATH-12 single-boom jumbo for drilling in small and medium workings (fig. 1). It is equipped with a Secoma RPH-200 hydraulic drifter for rotary or rotary-percussive drilling. It has a 10-ft-long drill steel and drills a maximum depth of 8½ ft. Cross-bits (X bits) with carbide inserts were tested in three sizes; 1-3/4, 2, and 2½ in.

Choice of Explosives

A large variety of commercial explosives and blasting agents are available today. Historically, dynamites and other stick explosives have been used in blasting frozen ground. These have a wide range of density, detonation velocity, and other physical properties, which are not discussed in detail here. Some of these properties are listed in tables 1 to 8 and summarized in figure 5 (Dick, 1968).

Table 1. Properties of straight nitrogen dynamite.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
60	60	1.3	19,000	good	poor	106
50	50	1.4	17,000	fair	poor	104
40	40	1.4	14,000	fair	poor	100
30	30	1.4	11,500	poor	poor	100
20	20	1.4	9,000	poor	poor	100

Table 2. Properties of high-density ammonia dynamite.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
60	52	1.3	12,500	fair	good	110
50	45	1.3	11,500	fair	good	110
40	35	1.3	10,500	fair	good	110
30	25	1.3	9,000	fair	good	110
20	15	1.3	8,000	fair	good	110

Table 3. Properties of low-density ammonia dynamite, high-velocity series.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
65	50	1.2	11,000	fair	fair	120
65	45	1.1	10,400	fair	fair	129
65	40	1.0	10,000	fair	fair	135
65	35	1.0	9,800	fair	fair	141
65	30	0.9	9,400	poor	fair	153
65	25	0.9	8,800	poor	fair	163
65	20	0.8	8,300	poor	fair	174

Table 4. Properties of low-density ammonia dynamite, low-velocity series.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
65	50	1.2	8,100	fair	fair	120
65	45	1.1	7,800	poor	fair	129
65	40	1.0	7,500	poor	fair	135
65	35	1.0	7,200	poor	fair	141
65	30	0.9	6,900	poor	fair	153
65	25	0.9	6,500	poor	fair	163
65	20	0.8	6,300	poor	fair	174

Table 5. Properties of blasting gelatin.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
100	90	1.3	25,000-26,000	excellent	poor	110

Table 6. Properties of straight gelatin.

Weight strength (%)	Cartridge strength %	Sp gr	Confined velocity fps	Water resistance	Fume class	Cartridge count
90	80	1.3	23,000	excellent	poor	105
70	70	1.4	21,000	excellent	poor	101
60	60	1.4	20,000	excellent	good	98
50	55	1.5	18,500	excellent	good	95
40	45	1.5	16,500	excellent	good	92
30	35	1.6	14,500	excellent	good	88
20	30	1.7	11,000	excellent	good	85

Table 7. Properties of ammonia gelatin.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
80	72	1.3	20,000	excellent	good	106
70	67	1.4	19,000	excellent	very good	102
60	60	1.4	17,500	excellent	very good	100
50	52	1.5	16,500	excellent	very good	97
40	45	1.5	16,500	excellent	very good	92
30	35	1.6	14,000	excellent	very good	90

Table 8. Properties of semigelatin.

Weight strength (%)	Cartridge strength (%)	Sp gr	Confined velocity (fps)	Water resistance	Fume class	Cartridge count
63	60	1.3	12,000	very good	very good	110
63	50	1.2	12,000	very good	very good	118
63	40	1.1	11,500	good	very good	130
63	30	0.9	10,500	fair	very good	150

Table 9. Characteristics of pneumatically loaded ANFO in small-diameter blastholes.

Loading devices	Tank pressure (psi)	Jet pressure (psi)	Loading rate (lb/min)	Loading denisty (g/cm^3)
Pressure vessel	10-30	NAp	15-70	0.80-0.85
Ejector loader (jet)	NA	40-80	7-10	0.90-1.00
Combination loader	20	20-80	15-25	0.90-1.00

NA - not applicable.

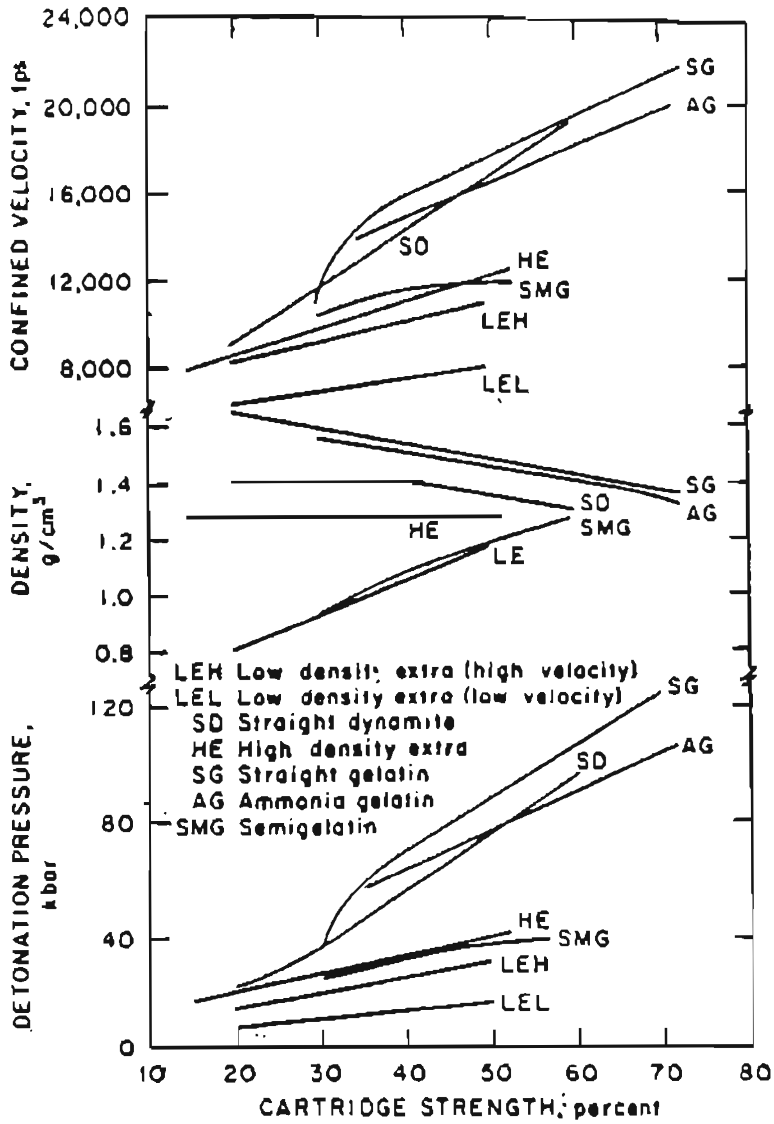

Figure 5. Average confined velocity and density, calculated detonation pressure of explosives.

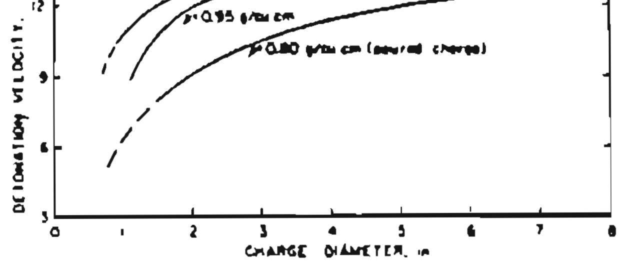

Figure 6. ANFO detonation velocity as a function of charge diameter and density.

One of the chief concerns in any enterprise is cost. An ammonium nitrate-fuel oil mixture (ANFO) is the most cost-effective blasting agent available and is being used successfully at the Wilbur Creek mine.

While ANFO has been tried by some miners in permafrost, results have not always been good, and many miners not experienced or informed in its use have concluded it is not particularly suitable for blasting in frozen ground because of its low detonation velocity (Emelanov and others, 1982).

Frozen ground is, by its nature, more plastic (deformable) than is rock. Transfer of dynamic stresses to the surrounding medium during an explosion depends on the time interval of the energy of detonation. A slower detonation velocity provides a longer interval when energy of the explosion is transferred to the ground; in permafrost, this results in a greater fracture zone (Emelanov and others, 1982).

Detonation pressure is a function of detonation velocity and density of an explosive. A high detonation pressure is desirable when blasting hard rock, but a lower pressure is sufficient for soft ground (Dick, 1968). Table 9 shows the characteristics of pneumatically loaded ANFO and table 10 shows the detonation velocities of ANFO in various borehole diameters. The velocity function is plotted in figure 6 (Dick, 1968). Assuming a velocity of 8,000 fps in a 1-3/4-in. borehole, the detonation pressure can be found from the monograph of figure 7 (Dick, 1968).

Low cost and detonation velocity are not the only properties of ANFO that make it a desirable choice. Its classification as a blasting agent simplifies transportation, storage, and safety considerations. It is easily handled in 50-lb sacks and is quickly and easily loaded into blastholes by a pnuematic 'gun.' Proper drilling, blowing dry, and loading procedures will ensure successful results in frozen ground.

DRILLING IN FROZEN GROUND

Drilling in frozen ground (silts, gravel, and bedrock) has been studied in at least two experiments at the CRREL permafrost tunnel in Fox, Alaska (Dick, 1970 and Mellor and Sellman, 1970). Drilling rates of 3 to 5 ft/min were reported with a jackleg drill and a 1½-in.-diam bit. This is a good penetration rate, but there were problems extracting the steel, as cuttings tended to refreeze behind the drill bit (Dick, 1970). However, it may be that the ground was thawing and filling the hole behind the bit (as was encountered at Wilbur Creek).

With a large rotary rig (Chicago Pneumatic T-650), 8-in.-diam holes were drilled at a rate of 2½ ft/min (Mellor and Sellman, 1970). At Wilbur Creek, holes of this size were not of interest.

Detailed studies of drilling rates have been performed in the USSR. There, drilling rates of 2.0 to 2.3 ft/min. were reported with hand drills and jacklegs (Emelanov and others, 1982). By recording drilling rates over short distances at various depths, an empirical formula for average drilling rate was developed, and from the data, a graph obtained of drilling rate vs

Table 10. Confined detonation velocity and borehole loading density of ANFO.

Borehole diameter (in.)	Confined velocity (fps)	Loading density (lb/ft of borehole)
1½	7,000-9,000	0.6-0.7
2	8,500-9,900	1.1-1.3
3	10,000-10,800	2.5-3.0
4	11,000-11,800	4.4-5.2
5	11,500-12,500	6.9-8.2
6	12,000-12,800	9.9-11.7
7	12,300-13,100	13.3-15.8
8	12,500-13,300	17.6-20.8
9	12,800-13,500	22.0-26.8
10	13,000-13,500	27.2-32.6
11	13,200-13,500	33.0-39.4
12	13,300-13,500	39.6-46.8

Table 11. Drill rates with a 1-3/4-in. drill bit, Wilbur Creek.

Average of:	Rate to 4 ft (ft/min)	Rate to full depth (ft/min)
all holes	2.93	2.66
gravel holes	3.14	2.95
bedrock holes	2.85	2.34

Table 12. Drill rates with a 2-in. drill bit, Wilbur Creek.

Average of:	Rate to 4 ft (ft/min)	Rate to full depth (ft/min)
all holes	2.20	1.79
gravel holes	1.83	1.60
bedrock holes	2.57	1.98

Table 13. Drill rates with a 2-1/2-in. drill bit, Wilbur Creek.

Average of:	Rate to 4 ft (ft/min)	Rate to full depth (ft/min)
all holes	2.93	2.66
gravel holes	3.14	2.95
bedrock holes	2.85	2.34

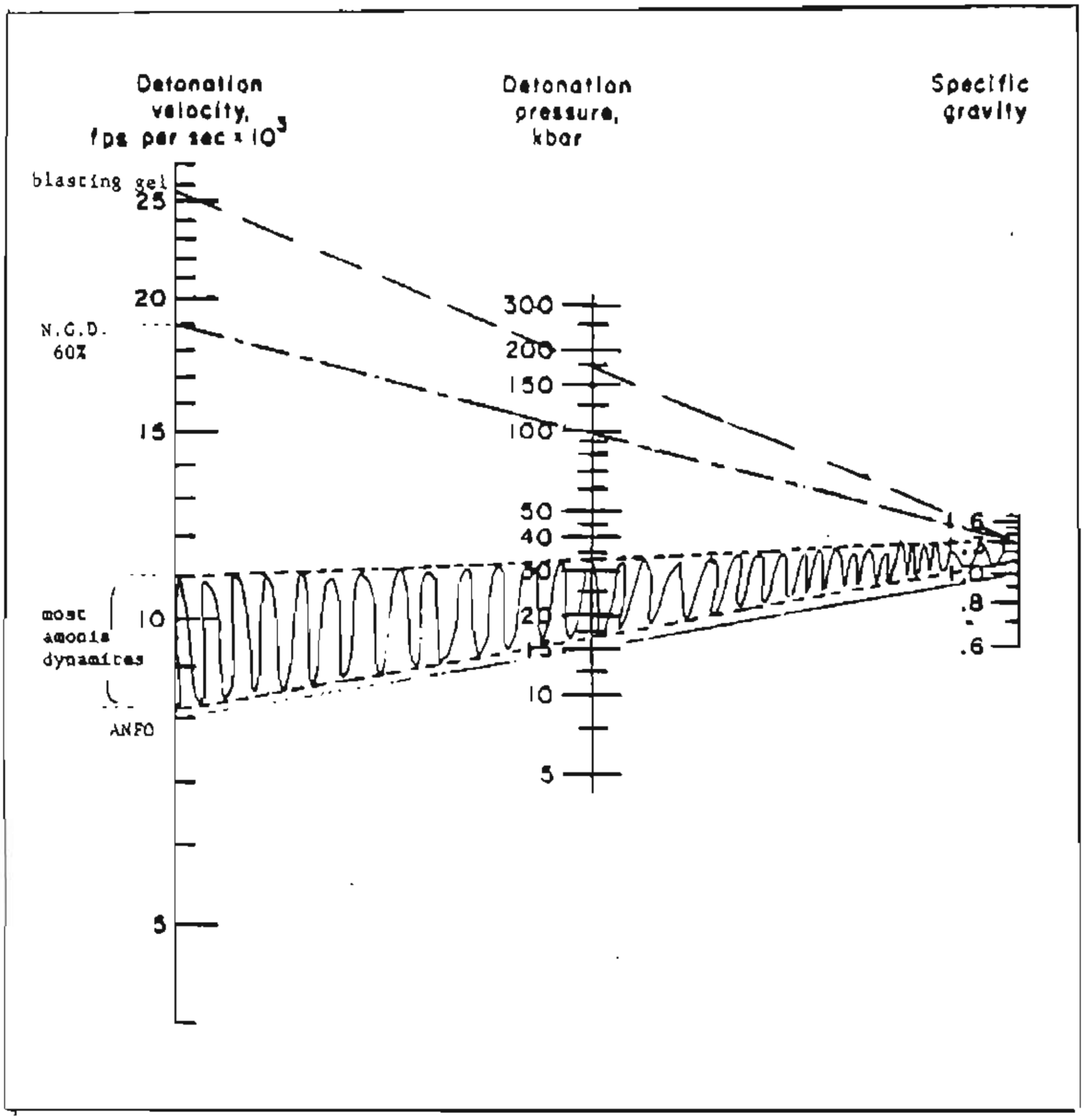

Figure 7. Nomograph for finding detonation pressure.

depth of hole (figure 8). Limited data for the ATH-12 is included. Drilling rates are seen to decrease with increasing depth, as would be expected.

Jumbo Drilling at Wilbur Creek

Because of the production-oriented nature of the work at Wilbur Creek, such detailed study of drilling rates was not practical. However, by using three different bit sizes, data was obtained on drilling rates of the jumbo in the first 4 ft and last 4 ft of drilling (the average hole was 8 ft long; tables 11-13).

The data show that on the average, drilling rates do indeed decrease with depth, though the changes are not high. In fact, in some drillholes, penetration rates were greater in the deeper part. This is easily explained

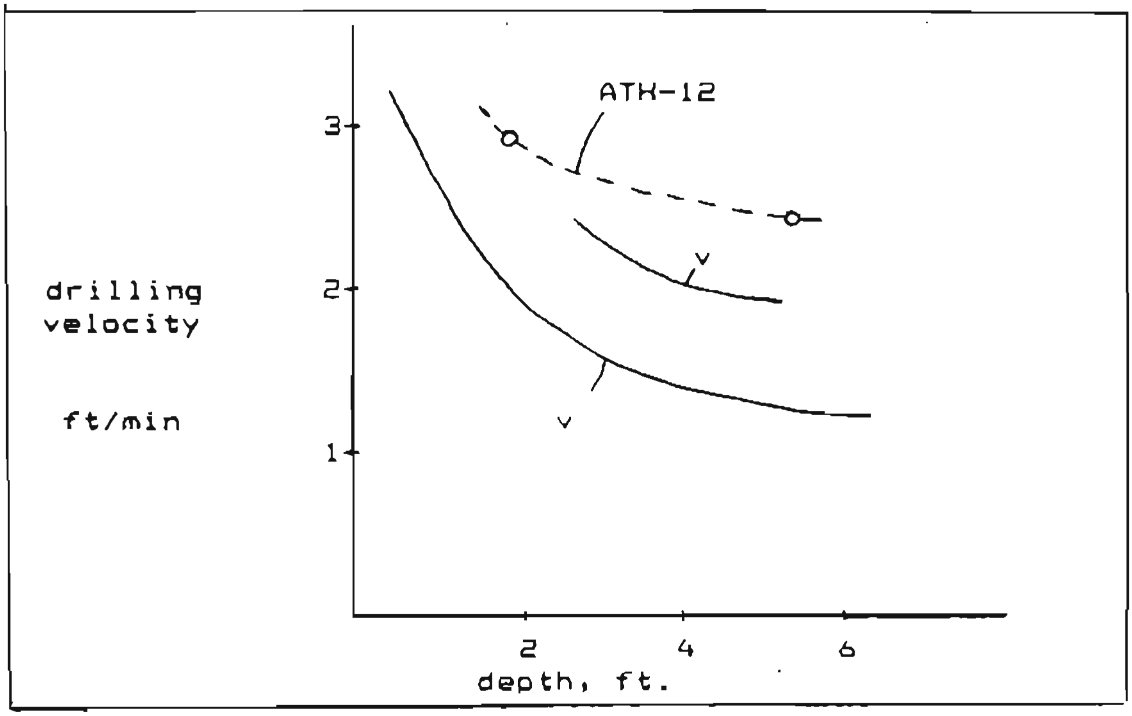

Figure 8. Drilling rate vs. depth of drill (from Emenlanov and others, 1982).

where the material being drilled is gravel, and the bit may be passing through large cobbles of hard rock in the first half of the drillhole.

This phenomeno is more difficult to explain when drilling in bedrock, which appears to be rather uniform in strength. However, the particular drill used is in need of maintenance, and its performance is less than optimum (and decreasing). This fact would explain the drill-rate decrease with depth but also places suspicion on the value of all the drilling-rate data obtained.

Total time to drill a round includes setup time for the drill, and was recorded as the time when the drill left the equipment shed until it was returned. The times ranged from an average of about 1 hr, 15 min for small drift rounds (approximately 9 by 15) to about 1 hr, 45 min for the larger rounds. (This assumes there are no problems with the drill or water line.)

Types of Cut

Success or efficiency in blasting requires that the blastholes be parallel (or nearly so) to a free surface. In developing an underground opening, there is typically only one free face, and the holes are drilled perpendicular to it. Because of this, the first blastholes to fire must create an opening (a second free face) essentially parallel to the remaining blastholes. These first holes are called the 'cut,' and are critical to the efficiency of the remaining holes in the round.

There are two major classifications of types of cuts, burn cuts (parallel hole cuts) and angle cuts. Some research has been conducted at the CRREL permafrost tunnel with both a burn cut and a V cut (Dick, 1970).

The type of cut and delay (long period or millisecond) affected the fragmentation and cleanup times in the experiments at the CRREL tunnel (Dick, 1970), but were not a factor when mucking with the Wagner LHD unit at Wilbur Creek. Figures 9 to 11 illustrate the muck-pile configuration expected for several cuts and delays (Dick, 1970).

Burn cuts typically consist of one or more unloaded holes surrounded by holes loaded with explosives. The empty holes provide space for the fracturing ground to swell into and be ejected. Burn cuts are the easiest to drill, as all the holes are parallel and perpendicular to the working face, but generally require more holes.

Angled cuts like the V cut are drilled at an angle to the working face, and so tend to throw the material out, creating a second free face. They require more care or precision to drill, but generally require fewer holes, thereby saving time and money.

There has also been considerable work done in the USSR on types of cuts. A number of these (fig. 12) and other cuts were tested at Wilbur Creek.

Face Conditions and Roof Stability

The working faces at Wilbur Creek typically consisted of 3 to 6 ft of bedrock topped with 3 to 5 ft of the lower (pay) gravel, overlain by silt layers. Lifters are drilled in bedrock. Near the middle of the face (cut holes, center; trim holes, sides), the holes may be drilled in bedrock or gravel, or a single hole may pass through both. Roof trim holes are typically drilled in gravel just below the silt layers (fig. 13). Occasionally, holes may be drilled in silt layers or lenses.

The rounds commonly break to one of the silt layers separating the two horizons of gravel or all the way to the bottom of the upper gravel. This is especially the case in wide openings greater than 12 to 15 ft.

If the roof fails to break to the silt or upper gravel, horizontal cracks will be present or will develop as roof layers slab off under their own weight. This is because ice between the layers are planes of weakness, and are loosened by the blast (or subsequent ones). These slabs are DANGEROUS and must be 'barred' or blasted down.

The roof must be checked after each round is fired, and appropriate safety procedures must be followed. Slabs that cannot be barred down or are not suitable for blasting down should be watched closely and removed when conditions allow. It is important to cultivate a habit of watching the roof when working underground!!

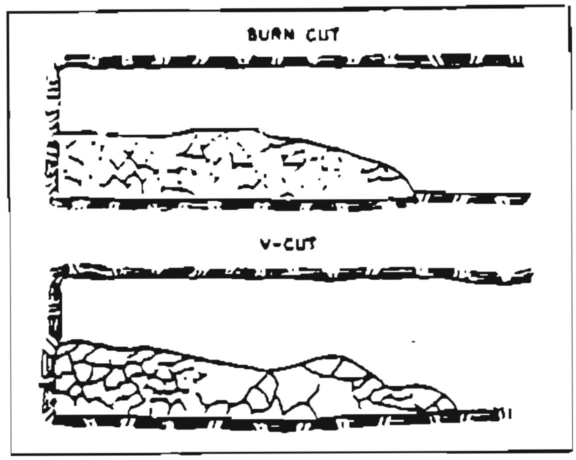

Figure 9. Fragmentation and shape of muckpile as a function of type of cut.

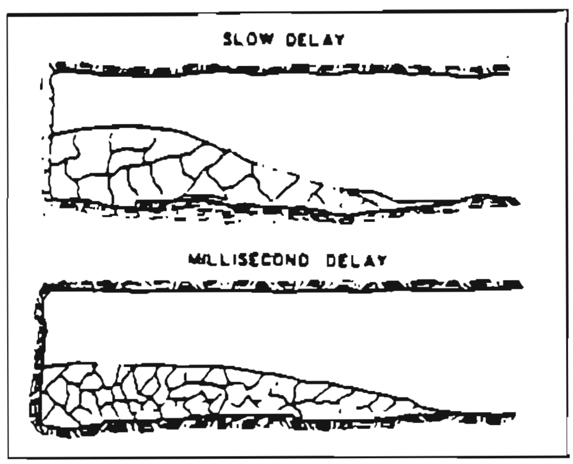

Figure 10. Fragmentation and shape of muckpile as a function of delay.

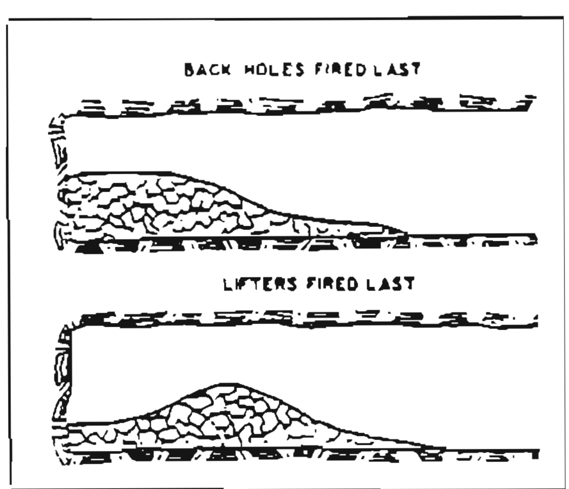

Figure 11. Shape of muckpile as a function of order of firing.

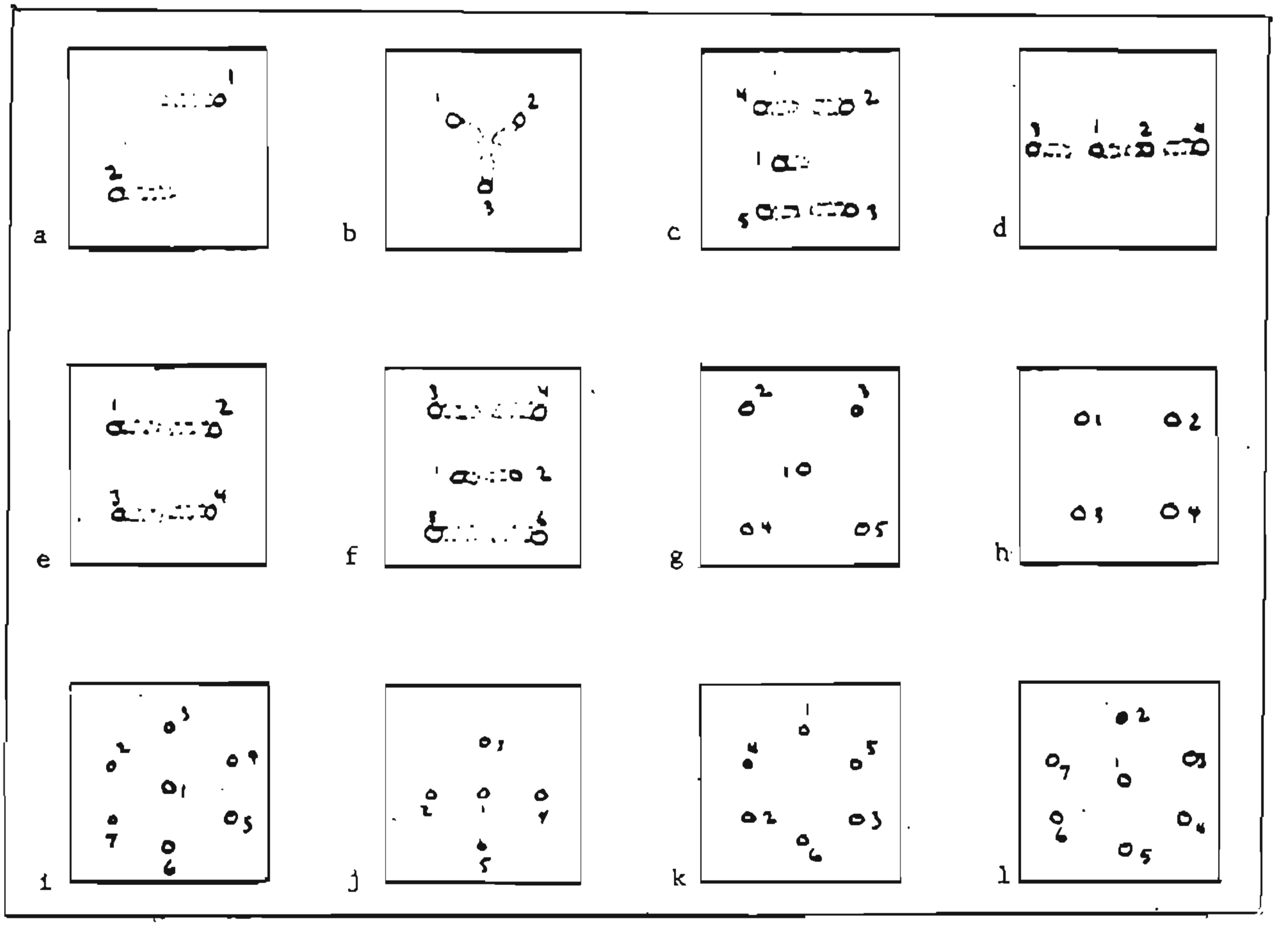

Figure 12. Types of cuts: a - knife; b - pyramid; c - wedge cut with central hole; d - double wedge (baby V); e - four-hole wedge; f - double wedge with six holes; g - prismatic (burn cut) with loaded central hole; h - four-hole prismatic cut; i - six-hole prismatic cut with loaded central hole; j - four-hole prismatic cut with unloaded central hole; k - layered cut; 1 - spiral cut (with increasing depth).

Experimental Rounds at Wilbur Creek

At Wilbur Creek a burn cut with a single unloaded hole yielded poor results. Very likely because of the high swell factor, the space of one empty hole is not sufficient to make a successful cut; the round 'freezes.'

Drilling many more holes is time consuming, water consuming, and more costly, so a round was adopted which has been called a 'modified burn' cut (Skudrzyk and others, 1987). A short hole (4 to 4½ ft deep) is drilled and is the first to fire. This hole is less than 'critical depth' and craters out to the face. Full-depth blast holes (8 to 8½ ft) to each side fire next, and usually break nearly full depth. The next delays fire holes above and below. Rows of holes to either side now have a free face to break to (fig. 14).

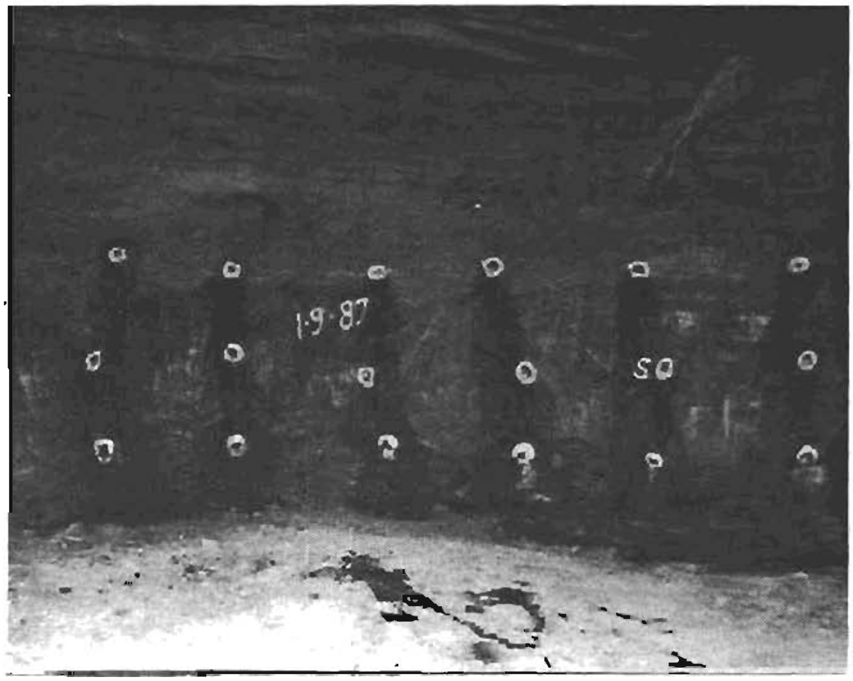

Figure 13. Typical drilling pattern. Upper row of holes at base of gravel-bedrock contact, lower holes in bedrock. Layers of muck in upper part of photo.

All holes are drilled essentially parallel and nearly perpendicular to the face. All holes should be angled up slightly to facilitate drainage of water and cuttings (except the lifters, which are angled down slightly). Rib (side) trim holes are angled out slightly.

A V cut with a 'buster' hole or a knife cut with buster also produced good results with a minimum of drilling (fig. 15).

A cylindrical or prismatic cut produced excellent results, but requires a few extra holes and primers. In this cut, short holes are loaded to within 6 in. of the collar, and adjacent long holes are loaded only in the back, as shown in figure 16 (Skudrzyk and others, 1987). This type of cut also worked well when the number of holes was reduced to four (two long, two short in a diamond configuration).

PROBLEMS ASSOCIATED WITH DRILLING

Problems with Water

As mentioned previously, drilling with water in frozen gravel thaws the ground, enlarging the borehole and causing large pieces to fall into it. This is a problem in the decomposed bedrock as well, particularly if the previous round has left bootlegs or misfires, which means drilling in shattered ground. Cold water, fresh from the river, may thaw the ground less, but introduces another problem.

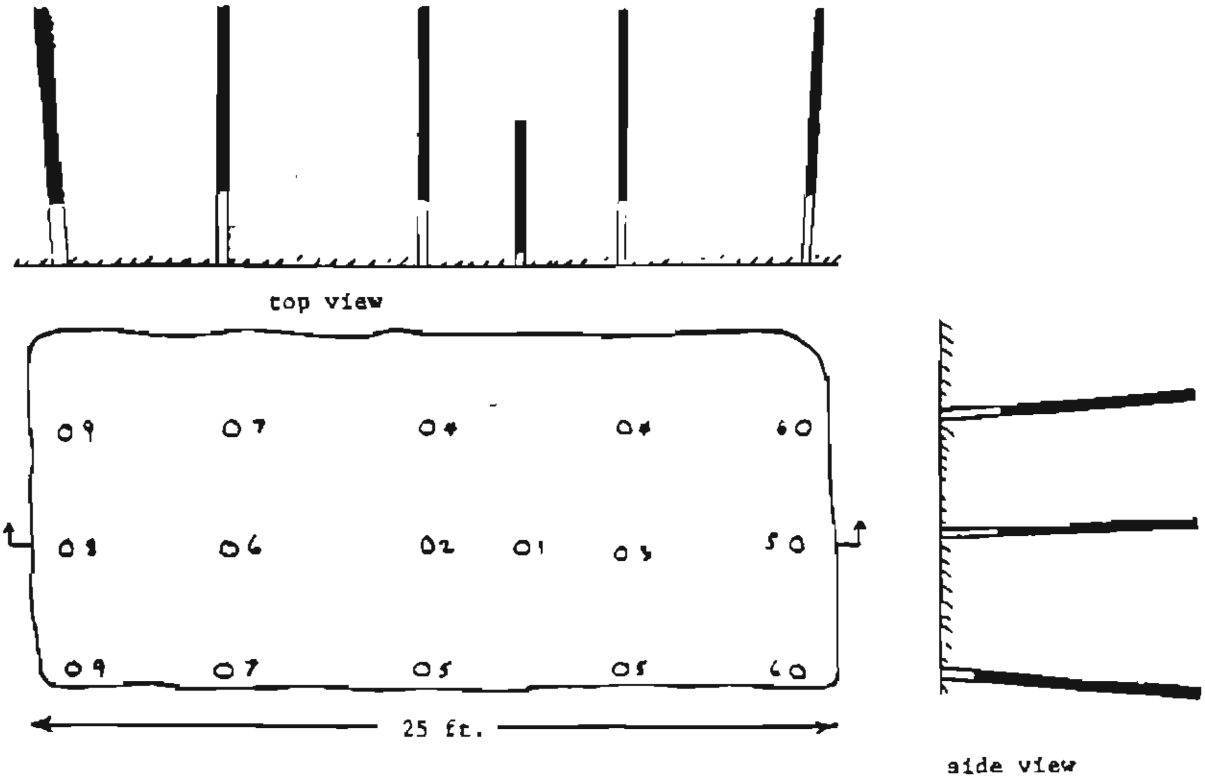

Figure 14. Modified burn cut.

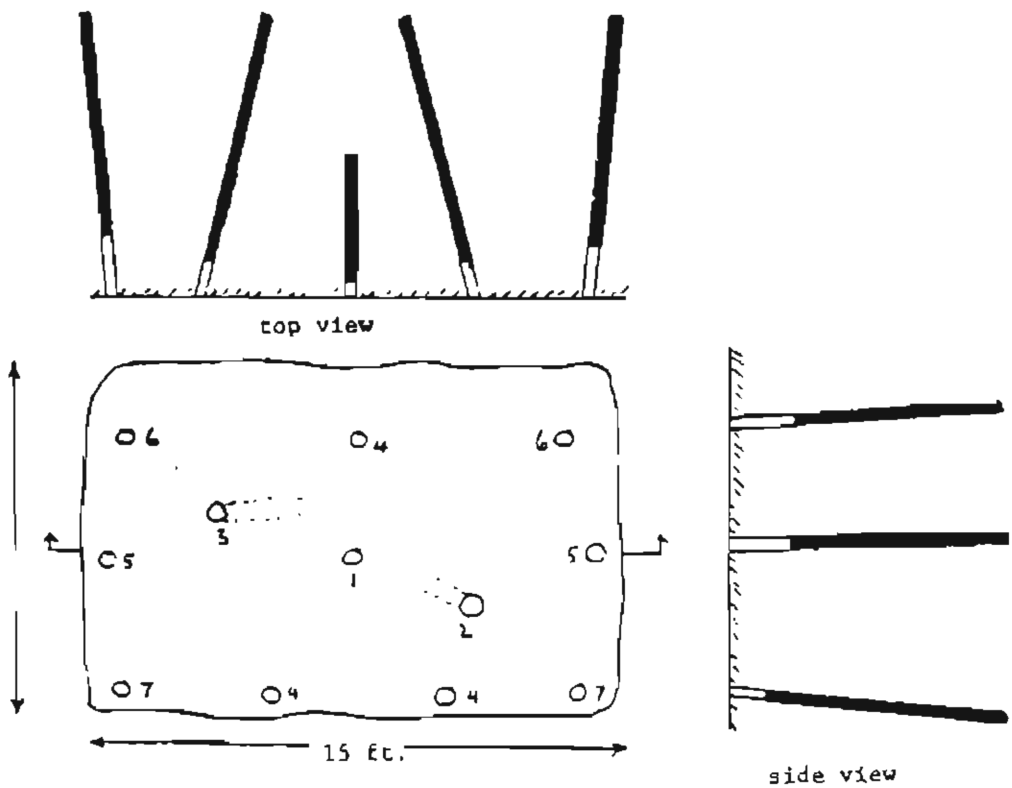

Figure 15. V cut.

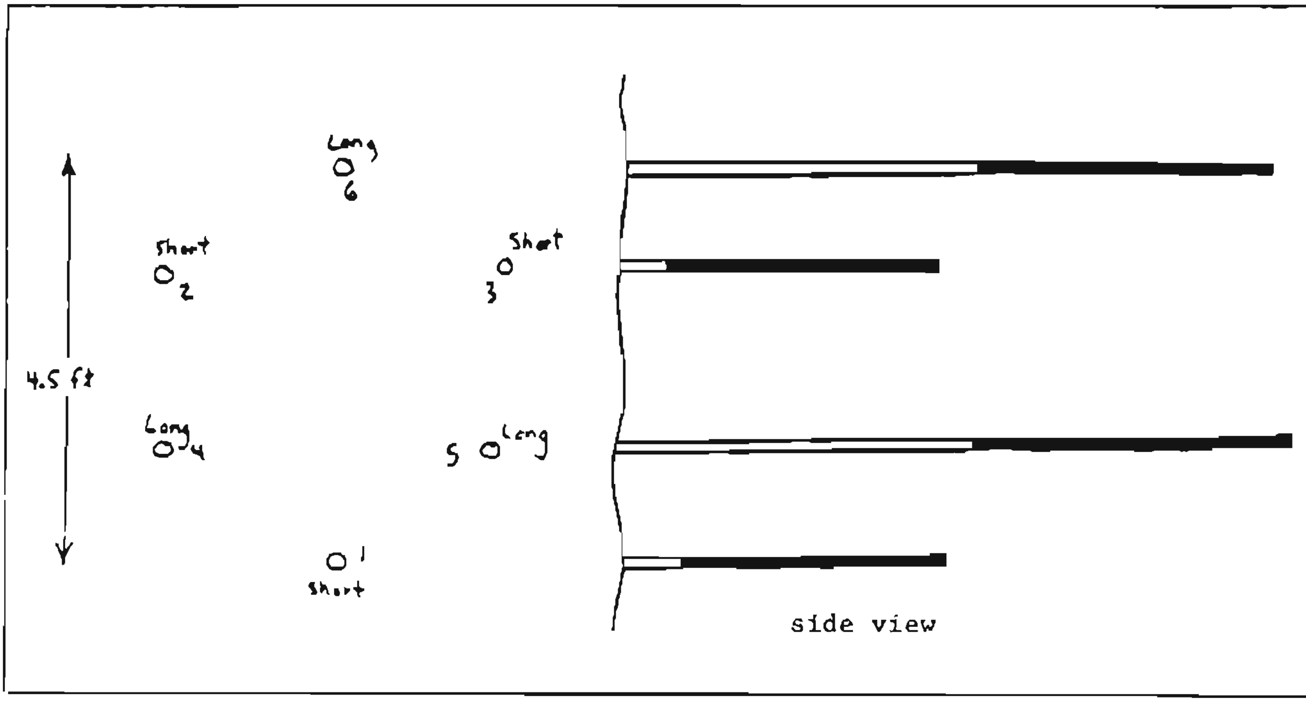

Figure 16. Layered prismatic cut (from Emelanov and others, 1982).

The water is pumped from a tank in the equipment shed, through a heavy-duty garden hose, to the drill underground. If the air temperature is very cold, the line will freeze up (more quickly if the water is still cold) if the water is not kept flowing. If the line freezes and too much time is spent thawing it, the water in the lines of the jumbo will freeze.

This 'garden-hose system' also presents a problem with water pressure. The farther the face is from the water tank, the lower the pressure that reaches the drill. (The drill has a safety feature that slows the drill if water pressure drops below a set level.)

These problems of temperature and pressure could be eliminated by replacing the garden-hose system with a jack tank; a small tank of water to be hauled with the drill and pressurized by the compressed-air line used to blow the holes clear before loading. The tank should hold enough water to drill a full round, plus a margin of insurance.

However, the first problem with water is getting it. The operator at Wilbur Creek has been fortunate that his first underground mining season has proved to be a mild Alaskan winter. Water has been available from the nearby Tolovana River, though the 1,000-gal tank has taken longer to fill at some times than at others. If water had not been available from the river, trips to Livengood or another source would have been necessary.

Spacing and Burden

Spacing blast holes too close, particularly in frozen ground, will result in 'deadpressing' when using ANFO. Propagation of the detonation front in ANFO depends on pore spaces, which create 'hot spots' in the explosive column as the detonation front advances. If the surrounding ground is compressed by an adjacent blasthole firing, the ANFO column is compressed, eliminating pore space (Dick and others, 1983).

If a round fails to pull well, a natural inclination is to drill more holes, closer together, to load more powder. When using ANFO in weak ground such as permafrost, this ameliorates the problem, not the solution. Studies have shown that holes placed closer than about 2 ft will completely deadpress, and the recommended minimum spacing should be about 2½ ft (Emelanov and others, 1982).

Bootlegs and Misfires

Bootlegs occur when the explosive column has fired, but had too much ground (burden) to break to the free face along the full depth of the blasthole. In frozen gravel, the ground is compressed by the explosion, and the remaining length of blasthole expanded to a diameter of 6 to 8 in.

Misfires occur when the primer fails or (more commonly) the primer fires but fails to initiate the ANFO column for some reason. In the latter case, the primer expands the back of the hole, as in a bootleg, but the hole may still be blown out with compressed air, reloaded, and fired.

Bootlegs can be a problem when drilling because they may be tightly packed in the front with muck by the mucker and not be visible when drilling. (All good miners---live miners---know it is unsafe to drill a bootleg because it may contain live powder.) If the drill inadvertently hits a concealed bootleg, the broken ground creates problems in extracting the steel and in blowing and loading the hole. Misfires present the same problem.

Optimum Bit Selection

The drill steel used on the jumbo was 1 in. in diameter. Three bit sizes were tested; 1-3/4 in., 2 in., and 2½ in. Although the larger holes are easiest to blow clear (with a ½-in.-ID blowpipe) and load, the larger annulus around the steel makes it more difficult for the cuttings to flow out, particularly when drilling the lifter holes. This makes extracting the steel difficult, which also is detrimental to the drilling machine.

Because of this, the 1-3/4-in. bit appeared to be the best selection for this machine and drilling steel. A different drill may be matched best with a larger or smaller bit.

Drilling with the larger bits was insufficient to evaluate their performance with respect to powder factor, fragmentation, and swell.

As a final note, operator experience, as with any machine, is essential for accurate drilling of an efficient blasting round.

LOADING EXPLOSIVES

Blowing Holes Clear

Before loading any explosives, you must first blow the holes clear of water, cuttings, and any pebbles and rock that have fallen in from the wall of the drillhole. It is extremely important to blow all of the water from the hole when blasting with ANFO, as ANFO has no water resistance.

The ammonium nitrate, which constitutes 94 percent of the ANFO mixture, is a salt and will melt the ice in the matrix of the permafrost. This amount of moisture may slow the detonation velocity of the ANFO column slightly, but should have no appreciable effect, provided the round is fired shortly after loading.

However, water in the holes (particularly in lifters, because they are drilled on a downgrade) will be immediately absorbed by the ANFO drills and likely result in a failure to detonate. Again, drilling holes other than lifters on a plus grade facilitates removal of water.

The blowpipe used should be longer than the deepest holes to be drilled (about 2 ft is sufficient). The blowpipe should be offset with another 3 to 4 ft of pipe with a valve. The offset allows the operator to stay somewhat clear of the muck, which is forcibly ejected.

About 100 psi air pressure is required. Substantially lower pressure will not do an adequate job. A ½-in.-ID steel pipe is suitable for small holes; smaller pipe may lack the strength and durability required, larger sizes will block the material to be blown out. It is recommended that the 90° joint between blowpipe and offset be reinforced.

A small raking tool is an invaluable accessory in removing rocks from the drill holes. Such a tool can be purchased or easily fashioned from a steel rod. The tool in use at Wilbur Creek appeared to be made with a 4½-ft-long 3/8-in.-diam steel bolt. The head was removed and the end flattened and bent over 90° (fig. 17).

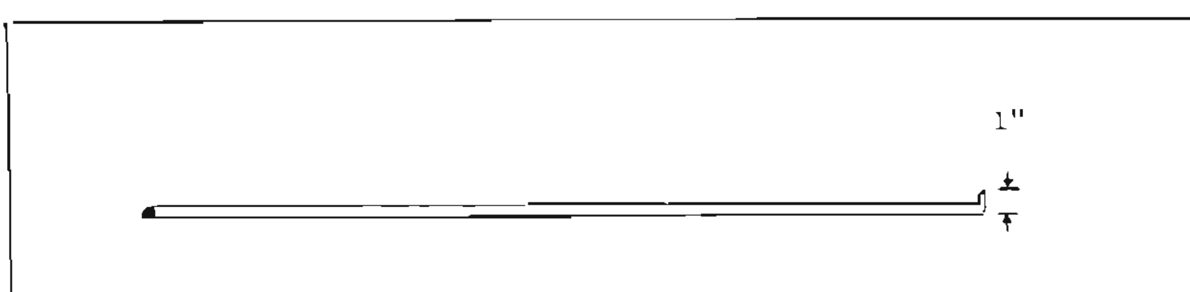

Figure 17. Raking tool.

Loading Primers

A primer is required to initiate ANFO. Often, the primer chosen is a stick of dynamite or other cartridged, cap-sensitive powder. At Wilbur Creek, 1¼ by 16-in. sticks of Tovex 220 (a water gel) were initiated with LP electric blasting caps. All holes were bottom primed. Caps are inserted in the primer and secured with a half-hitch in the leg wires, and the primers are inserted so that the bottom of the cap is directed toward the collar of the blasthole (DuPont, 1978).

In a few cases, a rock or pebble was found to be wedged in the hole after blowing it clear, but before the Tovex could be loaded. When the obstruction couldn't be blown clear, it was usually possible to insert a Detaprime booster past the obstruction and then load ANFO.

Detaprimes are small, high-energy boosters manufactured specifically for initiating ANFO in small boreholes (DuPont, 1978). They were found to perform well, and several rounds were tested using Detaprimes in all but the lifters. Use of Detaprimes can significantly reduce the cost of priming a round, but you must ensure that the holes are blown dry. As an added precaution, a small amount of ANFO can be loaded in the back of the hole before loading the cap and Detaprime.

Half sticks of the 16-in. Tovex were also tested to reduce priming costs. These also performed satisfactorily, but careful insertion of the cap is required, along with careful loading of the assembled primer.

Loading ANFO

At Wilbur Creek, ANFO was loaded with a pneumatic loader referred to as an ANFO 'gun,' which is actually a venturi-type loader. This device is very simple and easy to use. Other types of loaders are available (DuPont, 1978).

Some points to remember when loading ANFO:

1) ANFO has no water resistance; be sure the holes are dry

2) ANFO thaws ice; fire the round as soon after loading as is safe and practical

3) Be sure the discharge hose is long enough for the holes being loaded

4) Be sure the end of the discharge hose is near the back of the hole; mark the hose with tape or a spot of paint

5) Learn to withdraw the discharge hose at a rate equal to ANFO filling the blasthole; too fast may create air pockets in the ANFO column, too slow will build up ANFO behind the end of the discharge

6) Be sure to have the proper air pressure recommended for the type of loader in use

7) Be sure ANFO is stored properly in a cool, dry place that meets regulatory standards. ANFO has a shelf life; excessive shelf life or improper storage may cause evaporation of its fuel oil.

Loading Time

Holes are typically blown clear while drilling the round by either the driller or a second man, and so does not require additional time. However, a few holes may require additional work before loading the primer, and it is best that the blowpipe be left attached to the air line until all primers have been successfully loaded.

Preparing and hand loading the primers, and pneumatically loading the ANFO typically requires about 1 hr for a round. There was little time difference between large or small rounds because setup and preparation account for much of the time. Preparing and loading a primer requires a minute or less; loading ANFO in a single hole takes only slightly longer.

Powder Factor

In the small drift rounds a powder factor of close to 3.0 lb/yd^3 was calculated. This was based on measurements of the drift before and after blasting and on an estimation of the depth of ground the round pulled. The yardage figure obtained was checked on the basis of the amount of loose yards mucked divided by an average swell factor of 1.7 (obtained from data previously collected at the site).

Larger drift rounds averaged about 2.5 lb/yd^3 from similar calculations. Powder factors were sometimes higher, resulting from bad rounds (misfires or excessive bootlegs for various reasons), spillage during pneumatic loading, and waste due to excessive loading (blastholes other than cut holes need be loaded only to within 1½ to 2 ft of the collar).

The best powder factor resulted from a slab round from the rib of an existing opening. The powder factor was calculated to be 1.6 lb/cu yd, but not enough shots of this type were fired to calculate a good average value.

Drilling and Blasting Costs

Costs at Wilbur Creek have been covered in detail in another discussion (p. 89). For drilling, it will suffice to state that from other analysis, owning and operating costs appear to be less than $.15/yd.

Explosives costs are probably the highest of all equipment and materials costs in the operation. The figure calculated is \$1.85/yd^3. Use of Detaprime boosters could reduce this to about \$1.50/yd^3.

CONCLUSIONS AND RECOMMENDATIONS

The drilling and blasting procedures in use at Stan Rybachek's Wilbur Creek property have been quite successful in the underground mining of a frozen placer. Some of the practices can be improved to eliminate small problems such as freezing of the water supply line in subzero temperatures.

Using mechanized drilling with air and using a machine with a drilling capacity that is not stressed by using bit diameters of 2 to 3 in. or larger should be studied. Ground with a much thicker pay horizon would be more suitable for this type of work, as ore dilution of about 1:3 is being encountered at Wilbur Creek.

ACKNOWLEDGMENTS

The author is grateful to Dr. Frank Skudrzyk, University of Alaska, Fairbanks, for his advice, supplying of materials and equipment, and translation of literature.

Gratitude is also due Stanley Rybachek, who provided board and living quarters while at the mine, and Michael Roberts, whose years of underground experience helped keep the mine a safe place to work.

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COST-ESTIMATION HANDBOOK FOR SMALL PLACER MINES

by

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This U.S. Bureau of Mines publication presents a method for estimating capital and operating costs associated with the exploration, mining, and processing of placer deposits.

NEED FOR HANDBOOK

The handbook was the idea of George Gale, metallurgist at the USBM Western Field Operations Center. Gale recognized that placer mining is one of the few forms of mining where an individual can still make a living with a small-scale operation. Because placer operations typically operate on a limited budget, little money is available, and Gale felt that a cost-estimation system similar to the U.S. Bureau of Mines 'STRAAM' method (below) would be a valuable tool for placer miners and those charged with evaluating placer deposits.

U.S. BUREAU OF MINES COST ESTIMATING SYSTEM

The Bureau of Mines first published a system for prefeasibility cost estimation in 1975. This first handbook, developed by STRAAM Engineers, Inc., under contract to the Bureau, allowed the user to estimate reliable costs for numerous mining and milling methods. The STRAAM approach revolved around equations used to estimate capital and operating costs based on daily mine capacity. These equations were derived by performing a geometric regression on actual and estimated costs representing similar unit processes (raising, drifting, crushing, etc.).

To update the technological and cost data used in the STRAAM Handbook, WFOC was assigned to rewrite the part dealing with underground mining in 1984; this updated version is now called 'The BuMines Cost Estimating System,' (or CES). In carrying out this task, we subsequently developed this, the 'Cost Estimation Handbook for Small Placer Mines.'

HANDBOOK DEVELOPMENT

MINE VISITS

Because our own view of placer mining at WFOC was typically limited to reconnaissance of worked-out deposits, our first task was to visit as many operations as time allowed to determine current placer practices. Operations in Nevada, Oregon, Idaho, and Montana were visited, and information was collected on mines in Alaska and California. The main goals of this information gathering period were to:

1. Study equipment types, capacities, and use

2. Determine supply needs

3. Estimate labor requirements in relation to production rates, and to determine work-force characteristics (family, day laborers, owner-operator)

4. Estimate time and work spent on activities not directly related to production (reclamation, equipment downtime, etc.).

EQUIPMENT SELECTION

We realized that most of these operations revolved around a few key pieces of equipment. All costs (labor, fuel, supply, etc.) were related to the types and capacities of the equipment used. Consequently, we based our cost-estimation system on equipment types rather than unit processes, as in the case with the CES. We limited ourselves to equipment that could be obtained commercially, even though it might be easily made on site.

EQUIPMENT PARAMETER DEFINITION

The next task was to set a range of capacities for each type of equipment and to define characteristics affecting it, mainly from equipment manufacturers. All costs used to determine cost-estimation equations in the handbook are based on these equipment capacities mainly, and on actual operating characteristics. A typical list of 'base parameters' includes:

Bulldozers - mining

1. Straight 'S' blade
2. 50-ft cutting distance
3. 300-ft dozing distance
4. No ripping
5. 2,300-lb/LCY (linear cubic yard) density
6. 50-min/hr efficiency
7. Average operator ability
8. Even, nearly level gradient.

COST ESTIMATION FOR BASE DATA POINTS

After the parameters were established, costs were estimated for each aspect of an operation at varying capacities. For example:

Bulldozing - (Operating costs, 80-LCY/Hr, D-6 Caterpillar)

Equipment:	Parts	-	$ 3.15/hr
	Fuel	-	4.42/hr
	Lubrication	-	.32/hr
	G.E.C.	-	3.35/hr
Total		=	$ 11.24/hr

Equipment cost		=	$\frac{\$\ 11.24/\text{hr}}{80\ \text{LCY/hr}} = \$\ 0.14/\text{LCY}$
Labor:			
	Operator	-	1.000 hr @ \$ 15.69/hr = \$15.69/hr
	Maintenance	-	0.128 hr @ \$ 15.69/hr = \$ 2.01/hr
Total			= \$17.70/Hr
Labor cost		=	$\frac{\$17.70/\text{hr}}{80\ \text{LCY/hr}} = \$0.22/\text{LCY}$

Equipment operating and capital costs are typically based on information supplied from equipment manufacturers.

COST DERIVATION

Cost estimations such as those above were repeated for varying capacities to provide the base data points for the regression analysis. For instance, in the case of the bulldozer operation, costs were estimated at 10, 50, 80, 100, 150, 200, and 250 LCY/hr. The regression analysis then provided an equation for estimating costs at various capacities. It is these equations that appear in the handbook.

Best-fit equations were usually obtained with a geometric regression. This produced an equation in the form of $Y = A(X)^B$, with X representing the capacity variable and A and B provided by the regression analysis.

With the above procedures, equations were determined for each exploration, mining, milling, and supplemental function included in the handbook. In most cases, more than one equation is required. Equations calculated include:

- Equipment operating cost
- Labor cost
- Supply cost (where applicable)
- Capital cost.

THE HANDBOOK

In its final form, the hand book is divided into two sections, one on the equipment and methods typically used in placer mining and with the cost equations and adjustment factors needed to estimate placering costs. With the handbook, the user will be able to estimate costs for the following:

Exploration	Development	Overburden removal
Planning	Access roads	Bulldozers
Churn drilling	Clearing	Draglines
Bucket drilling		Front-end loaders
Trenching		Rear-dump trucks
General reconnaissance		Scrapers
Camp costs		
Rotary drilling		
Helicopter rental		

Mining	Processing	Supplemental systems
Backhoes	Conveyors	Buildings
Bulldozers	Feed hoppers	Employee housing
Draglines	Jig concentrators	Generators
Front-end loaders	Sluice	Lost time and general services
Rear-dump trucks	Spiral concentrators	Pumps
Scrapers	Table concentrators	Settling ponds
	Tailings placement	
	Trommels	
	Vibrating screens	

SECTION I - EQUIPMENT DESCRIPTIONS

Section I of the handbook is designed to familiarize the user with equipment and methods typically used in placer mining. It contains detailed descriptions of most equipment included, and some general discussions on exploration, mining, and milling practices. A typical description of equipment appears below:

Bulldozers

The bulldozer is the most popular tool in placer-deposit extraction. It can be used for overburden removal, pay-gravel, excavation, bedrock cleanup; overburden and pay-gravel transportation; road construction; and tailings placement. Other than the dragline, this device is the only one capable of handling all tasks required for placer mining in a practical manner and must be considered if capital is scarce.

Although bulldozers can handle all placer-mining functions, they are not necessarily the most efficient machine for any one task. With its ripping capacity, the bulldozer can clean up bedrock (although the backhoe is more selective and efficient). The bulldozer is used to transport gravel, but trucks, scrapers, and front-end loaders can often do the job cheaper. Bulldozers are not well suited to move large volumes of gravel or dig deep, like draglines can.

But, the bulldozer can excavate, transport, and load the mill all in one cycle, eliminating the need for expensive rehandling. Dozer capacities for excavating and hauling range from 19.0 loose cubic yards per hour for a 65-hp machine up to 497.5 loose cubic yards per hour for a 700-hp dozer (based on a 300-ft transportation distance). Capacity depends on ripping requirements, operator ability, cutting distance, haul distance, digging difficulty, and haul grade.

Dozers are best suited for situations where deposit and overburden thicknesses are not excessive, large obstructions are few, and haul distances are less than 500 ft.

The equipment description usually contains:

- Applications: How the machine is best used, and how it compares to other equipment used for the same functions.

- Capacities: Lists sizes of equipment most often used and expected productivity, along with factors that will alter productivity.

- Advantages and disadvantages: States 'best-use' scenario and lists reasons that machine could be used in certain situations.

Operation Design

All equipment included in the handbook is categorized into one of four major mining functions:

1) Exploration
2) Mining
3) Milling
4) Supplemental systems.

The main principles of each of these functions are briefly discussed in section I, to help the evaluator to come up with a viable method of mining his deposit.

SECTION II - COST ESTIMATION

Cost Equations

Costs are calculated by simply inserting the maximum amount of material handled hourly by a specific machine into the cost equation. The maximum amount of material handled (pay gravel, overburden, and waste) is used so that the machine is properly sized.

Costs are calculated in dollars (for capital cost) and dollars per cubic yard (operating costs).

Cost-data Adjustments

Individual components of each cost are stated (for example, 47 percent parts, 53 percent fuel and lubrication) so that costs may be updated from the January 1985 base. This is done by using indexes available from WFOC. To obtain the updated cost, the ratio of the current index divided by the January 1985 index is multiplied by the representative percentage of cost.

Site-adjustment Factors

These factors (which in the bulldozer example include distance, gradient, ripping, used equipment, and digging difficulty) adjust the costs for a specific set of circumstances and are probably the most important alterations to the base costs. These factors make each cost estimate unique for each deposit.

Labor-adjustment Factors

Labor costs are based on a rate of \$15.69/hr for mining functions and \$15.60/hr for milling. These are averages of rates we found in the western

U.S. (including Alaska) and include a 24-percent burden. As expected, we found that wage scales at small placer-mining operations were highly variable and that our averages were best used only as a base. Labor costs are easily adjusted by dividing the expected wage by the base wage and then multiplying the calculated labor cost by that ratio.

Capital-cost Equations

After all the necessary data and labor adjustments have been made to the base costs, they are entered, along with the calculated site-adjustment factors, into the total cost equation at the end of each cost page. This equation will produce the total adjusted capital cost for that piece of equipment or function. This figure is then entered on a cost-summary page for the final capital-cost calculation.

Operating-cost Equations

Again, once the necessary data and labor adjustments have been made to the base costs, they are entered, along with the calculated site-adjustment factors, into the total cost equation at the end of each cost page. This equation will produce the total adjusted cost per cubic yard of material handled by that particular piece of equipment for that specific function. This figure is then multiplied by the total amount of material handled annually by that piece of equipment for that specific function and the product is entered on a cost-summary page. The total cost per cubic yard is then calculated by dividing the total annual operating cost by the annual amount of pay gravel mined.

FINANCIAL ANALYSIS

Now, a cost estimation is not a financial analysis; capital and operating costs represent only two variables in a total financial evaluation. The evaluator must also consider:

1) Recoverable value of commodity
2) Local, state, and federal taxes
3) Capital depreciation
4) Depletion allowances
5) Desired return on investment
6) Cost and method of project financing
7) Inflation
8) Escalation
9) Environmental intangibles.

In short, a prospect is not economically feasible simply because the apparent commodity value exceeds the accrued capital and operating costs calculated from this manual.

SUMMARY

Use of the Handbook

The handbook is best used as a prefeasibility cost-estimation tool. With the information it contains, the user should be able to put together a rough design of an operation suitable for cost estimation. However, the validity of the cost estimate is directly related to the detail of the design. (The more detailed and representative the design, the more valid the resultant figures.)

The envisioned use of the handbook is that an evaluator or operator with a few basic facts about the deposit (such as location, topography, depth of gravel, some idea as to thickness of overburden, water availability, and a rough estimate of total volume) will be able to estimate:

- Exploration work required
- Equipment used for mining
- Average mine-to-plant haul distances and gradients (if applicable)
- Concentration method and equipment supplemental functions required (settling ponds, employee housing, etc.).

With this information, capital and operating costs can be estimated using the equations in section II of the handbook. Capital costs are reported directly in dollars, and operating costs are summarized in dollars per year. The average operating cost per cubic yard pay gravel is then calculated by dividing the annual operating cost by the total amount of pay gravel mined annually.

VALIDITY OF ESTIMATED COSTS

Actual cost figures from active placer operations are difficult to obtain mainly because many miners fail to account for and categorize costs in a way that lends itself to a dollar-per-cubic-yard cost or a lump-sum capital cost. In addition, costs and production rates may vary considerably from year to year. Consequently, we have not yet gathered enough actual cost information to calculate a statistically meaningful accuracy range for the handbook. However, because costs are estimated on a more finely divided basis than are those of CES (types of equipment as opposed to unit processes containing several types of equipment), we believe they may be more representative.

To obtain a copy of the cost-estimation handbook, contact the author.

DISTRIBUTION, ANALYSIS, AND RECOVERY OF PLACER GOLD FROM THE PORCUPINE MINING AREA, SOUTHEASTERN ALASKA

by

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INTRODUCTION

The U.S. Bureau of Mines (Bureau) and Alaska Division of Geological and Geophysical Surveys (DGGS) completed an evaluation of the placer resources of the Porcupine mining area near Haines, Alaska in 1985. The Bureau sampled and evaluated the placer deposits and the DGGS mapped the geology and summarized the glacial geologic history of the area. This report summarizes the results of the study. More complete information has been published in open-file reports by Bundtzen (1986) and Hoekzema and others (1986).

ACKNOWLEDGMENTS

The authors thank the following miners in the Porcupine mining area for their direct help with this study: Jo Jurgeleit and Earl Foster on Porcupine Creek, Jim McLaughlin on McKinley Creek, and Wes Childers on Nugget Creek. We also thank the Haines Library for supplying historical information concerning the discovery and development of the Porcupine Mining Area.

STUDY AREA

The Porcupine mining area is located in southeastern Alaska, about 30 mi west-northwest of Haines (fig. 1) in the Juneau mining district. The mining area encompasses about 200 mi^2 and is bordered on the south and east by the Tsirku and Chilkat Rivers, on the west by the Canadian border, and on the north by township line 27S (fig. 2). Land access is provided by the Dalton Highway, which runs from Haines, Alaska, to Whitehorse, Y.T., and by numerous logging and mining roads. Access to the upper reaches of Tsirku River is by either helicopter or airboat.

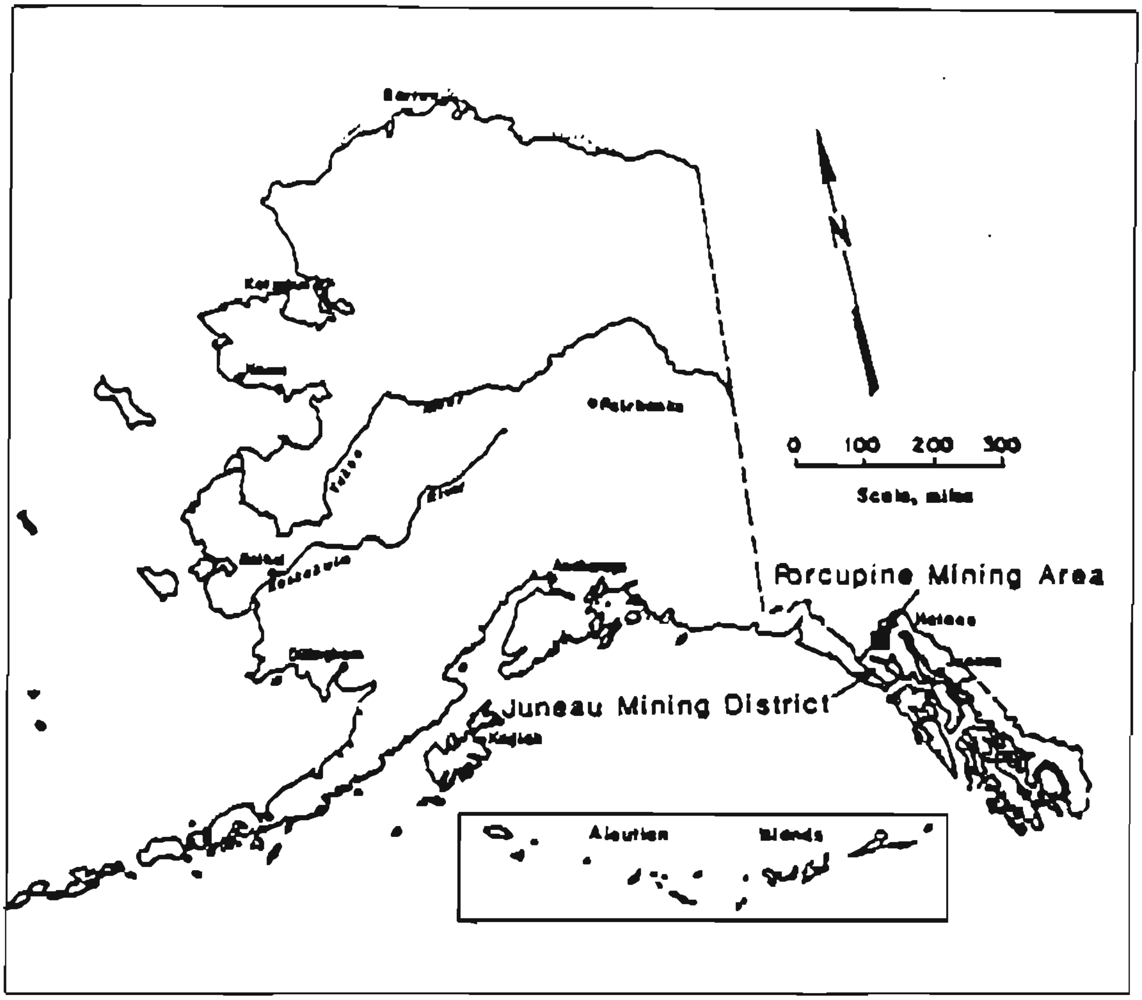

Figure 1. Index map of Alaska showing Porcupine mining area.

Land status of the area is complex (fig. 2) (Roberts, 1985). Much of the mining area is currently managed by the Bureau of Land Management (BLM) and is open to mineral entry. According to BLM records, the area had five patented and 444 unpatented placer claims as of July 22, 1985. The approximate location of current (1985) mining claims is also shown on figure 2. BLM land-status plats should be checked for detailed site-specific information.

MINING HISTORY AND PRODUCTION

In the spring of 1898, packers on the Dalton trail panned gold from the gravels of the Klehini River. Shortly after the discovery, most of the streams in the Porcupine mining area were staked; however, many claims were subsequently dropped because of the low quantities of gold found. Gold-producing drainages in the Porcupine mining area include Porcupine, McKinley, Cahoon, Nugget, Cottonwood, and Christmas Creeks. Production records for the Porcupine mining area are sparse. Minimum estimated production through 1985 was 79,650 oz, based on Bureau records and reports by Wright (1904), Roppel (1975), and Beatty (1937) (table 1).

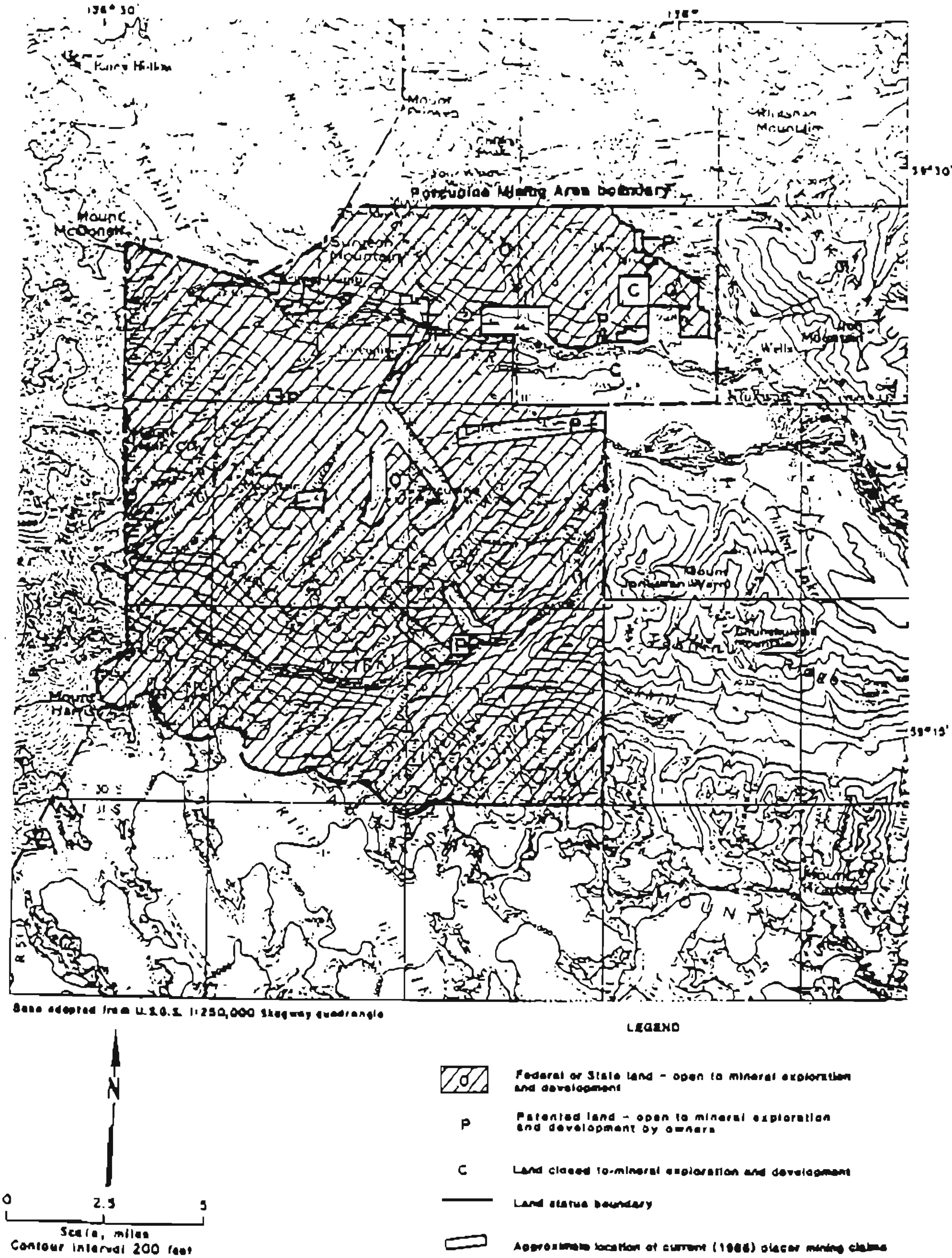

Figure 2. Land status and placer claim map of Porcupine mining area.

Table 1. Reported placer-gold production from the Porcupine mining area, 1900-85.

Drainage	Active years	Source	Quantity (oz)
Christmas Creek.....	1900-1985	(estimated)	200
Nugget Creek........	1902-1909	Beatty (1937)	350[a]
	1909-1985		100
Porcupine, Cahoon, and McKinley Creeks	1898-1903	Wright (1904)	27,000[a]
	1904-1915	Eakin (1919)	43,000[a]
	1916-1925	Beatty (1937)	6,000[a]
	1916-1936	Greatlander (1977)	[b]
	1936-1975	Roppel (1975)	500
	1975-1985	(estimated)	2,500
		TOTAL	79,650

[a]Based on placer gold evaluated at $17/oz.
[b]The Greatlander Shopping News reported that 78,000 oz of gold were produced during this period. However, this quantity is unsubstantiated by any other source of information available to the authors. Some additional production is likely.

GEOLOGIC SETTING AND MINERALIZATION

Bedrock in the Porcupine mining area consists of metamorphosed sedimentary rocks (slates, phyllites, and marbles), which have been intruded by igneous rocks of the Coast Range complex. The area has been extensively glaciated, and glaciers are still present at the headwaters of many drainages.

BEDROCK GEOLOGY

Possibly the oldest rock unit in the area is the Middle to Late Palezoic Porcupine Limestone (or marble), which forms prominent outcrops of carbonate along the access road on the south side of the Kleheni River and along canyon walls of Porcupine Creek.

Overlying the Porcupine Limestone is the Porcupine slate, which may range from Mississippian to Pennsylvanian (Redman and others, 1985). The slate, sandstone, and siltstone of the Porcupine Slate form a complex antiform throughout the central portion of the study area. Auriferous lodes cutting this 'slate belt' are believed by many previous workers to be the main source of placer gold in the Porcupine mining area.

The western and northeastern parts of the study area are underlain by pillow basalt, carbonate, and volcaniclastic sediments that may be Triassic, based on fossils collected in 1985 (Ken Dawson, oral commun., 1985).

GLACIAL GEOLOGY

A knowledge of the glacial history of the area is important for understanding the placer deposits. Virtually all the land forms are Wisconsin or younger (< 100,000 yr B.P.). Pleistocene glaciers advanced, retreated, and readvanced, resulting in at least three bedrock-incised channels or terrace levels in the valleys of Porcupine, Cahoon, and McKinley Creeks (Molnia, 1986; Péwé, 1975). Apparently, the remnants of these channels avoided ice scour and, except for deposition of glacial drift and erratics, were unaffected by later events. The oldest recognized terrace level occurs 250 to 300 ft above the modern canyon levels of McKinley and Porcupine Creeks; these are followed downstream by levels at 140 to 200 ft, 50 to 75 ft, and a final youngest level that is 25 to 40 ft above the modern drainages (fig. 3). The oldest terrace level may be a composite of fluvial material and drift not incised into bedrock.

Radiocarbon samples were collected from an exposed mine cut directly on the base of the 'dry channel' (fig. 3) located on the east and west sides of lower Porcupine Creek, as described by Beatty (1937). The dates obtained are shown on table 2, and show that the modern stream incision is relatively recent.

Table 2. Summary of radiocarbon analyses of channel gravels from Porcupine mining area.

Lab no.	Field no.	C-14 age	Remarks
Beta 11090	85BTC2.....	2,190 ± 90 BP	Woody material in dry channel near waterfall.
Beta 11091	85BTC3.....	2,640 ± 100 BP	Wood from base of dry channel, western side of Porcupine Creek.

PLACER GEOLOGY

Heavy-mineral placer deposits in the Porcupine mining area formed during multiple glaciofluvial cycles and occur in bench deposits in incised bedrock channels and glacial till, alluvial fans, and modern stream gravels.

Stream gradients indicate that the Porcupine mining area is a very high energy fluvial environment. The average stream gradient of the study area is 500 ft/mi compared with averages of 80 to 150 ft/mi in many interior Alaska placer districts.

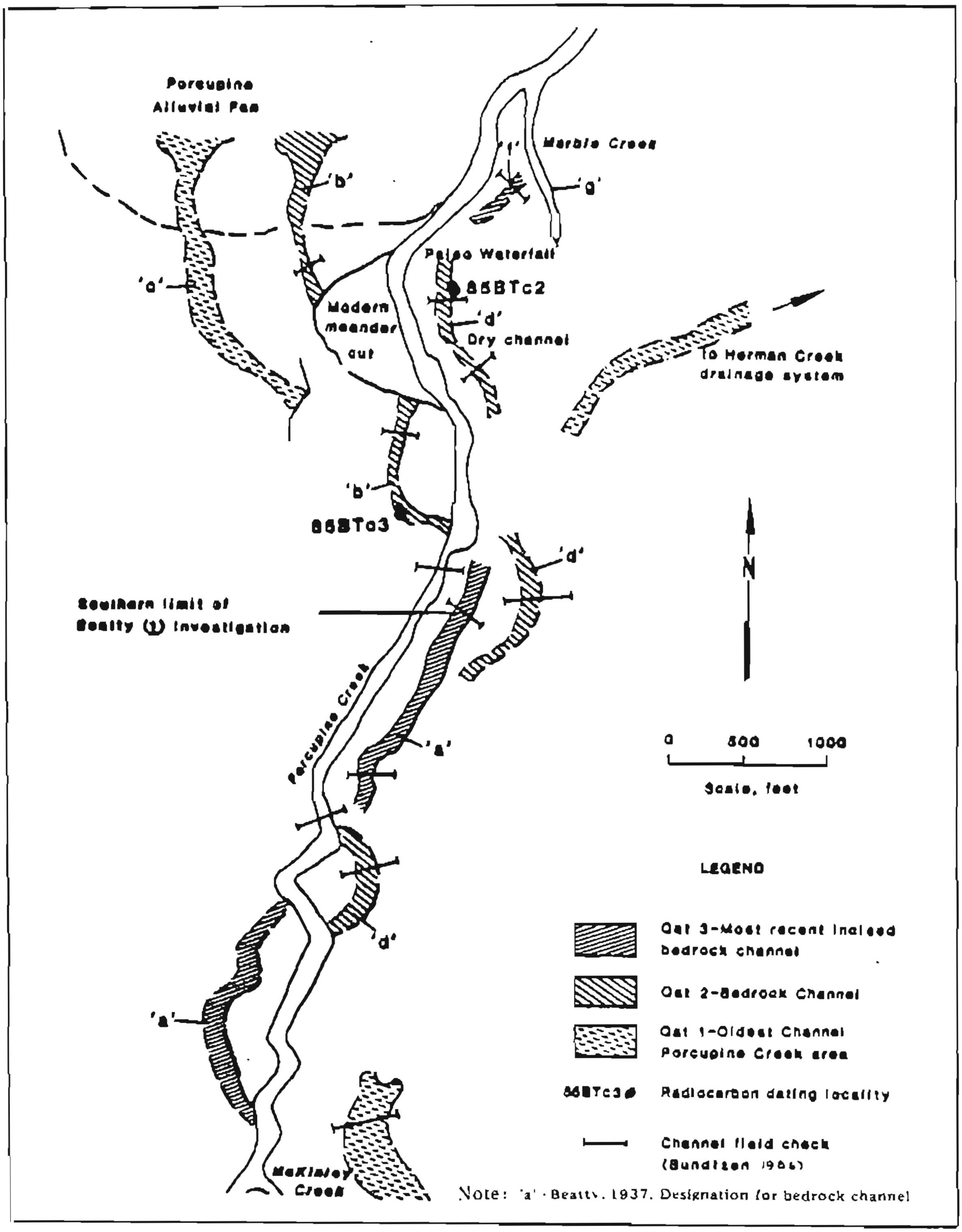

Figure 3. Abandoned channels on lower Porcupine Creek (from Bundtzen, 1986).

Bedrock sources of the placer gold have been identified by Eakin (1919), Beatty (1937), Still and others (1984), and Bundtzen and Clautice (1986). The most likely bedrock sources are crosscutting quartz-sulfide-gold fissure veins associated with altered mafic dikes that cut Porcupine Slate in the McKinley and Cahoon drainages. Pyritiferous zones in the Porcupine Slate also contain anomalous gold values ranging up to 2 ppm gold (Jan Still, oral commun., 1985).

Table 3 summarizes trace-element and gold fineness of placer gold collected during the Bureau-DGGS investigations. Gold fineness is expressed as a ratio of gold to silver + gold as suggested by Boyle (1979) and Metz and Hawkins (1981).

The average fineness of Bureau samples from the Porcupine mining area, using the method of Boyle (1979), is 837, close to the 820 reported by Smith (1941), who used records from four locations on the Porcupine Creek drainage for his analysis. The range of fineness in the Porcupine mining area is also consistent with those reported by Moiser (1975) for epithermal and lower mesothermal temperatures of formation. Bullion was analyzed for the trace metals copper, lead, zinc, and antimony. Significantly, samples containing detectable copper were found in McKinley and Cahoon Creeks, perhaps suggesting recent association with lode sources. The gold-to-copper ratio is much too high for typical gold placers of any temperature range, but the presence of antimony in single samples on Cahoon and Porcupine Creeks suggests formation in epithermal or lower mesothermal temperature ranges (Moiser, 1975).

Placer gold from McKinley, Porcupine, Nugget, and Christmas Creeks was miscroscopically examined to delineate characteristics of transport and origin of the bullion that was mined. Consistently, two distinctive types of gold are present in the analyzed concentrates: well-worn, rounded, bright 'nugget' gold that shows evidence of fluvial transport, and small wirelike grains with quartz and undetermined gangue mineralogy that show little evidence of stream transport. More than one lode source may be present or proximal lode gold and 'nugget' gold has been transported by fluvial mechanisms.

Both Beatty (1937) and the authors noted a general lack of fine gold (100 mesh or smaller) in the Porcupine mining area. The extremely high-energy nature of placer formation in the area suggests that virtually all fine gold has been flushed down the streams and possibly out of the study area. However, the Glacier, Porcupine, and Nugget alluvial fans represent significantly lower energy fluvial environments than those of the main feeder streams entering into the lower valleys, which suggests that alluvial fans may have accumulated part of the fine-gold fraction absent in the main production streams.

BUREAU OF MINES INVESTIGATION

In 1985, the Bureau collected 78 reconnaissance, 53 channel, and four site-specific bulk placer samples. All the major streams in the mining area were sampled, with at least one sample taken from each drainage. All site-specific bulk samples were taken from lower Porcupine Creek.

Table 3. Trace-element and gold-fineness analyses of placer gold from Porcupine mining area[1]. Elements measured in parts per thousand (ppt).

Field no.	Drainage basin locality (creek)	Sample weight (mg)	Gold (ppt)	Silver (ppt)	Copper (ppt)	Antimony (ppt)	Other (ppt)	True 2/ fineness	Remarks
9047....	Porcupine.......	21.64	794	140	15	50	1	850	Channel sample 0.1 yd^3, Porcupine Creek.
9096....	Porcupine.......	64.01	902	90	ND	ND	8	909	Channel sample 0.1 yd^3, but below channel.
9081....	Porcupine.......	35.36	817	145	ND	ND	38	849	Channel sample 0.1 yd^3, modern Porcupine channel.
9043....	Porcupine.......	34.75	812	144	ND	ND	44	849	Channel sample 0.1 yd^3, bench upstream from cabin.
9002....	Porcupine.......	64.94	822	155	ND	ND	29	841	3 pans on bedrock from bench west side of creek.
9037....	Porcupine.......	67.18	838	107	ND	ND	55	886	Channel sample 0.1 yd^3.
9119....	Porcupine.......	50.70	838	115	ND	ND	47	879	0.5 pan, dry channel, east side Porcupine Creek.
9112....	McKinley........	65.82	811	187	ND	ND	2	813	Channel sample 0.1 yd^3, on bedrock.
9109....	McKinley........	4.97	779	170	ND	ND	51	820	Channel sample 0.1 yd^3, boulder layer under colluvium.
3/ 9106....	McKinley........	33.74	669	259	22	ND	50	721	From sulfide vug, 'ladder vein'.
84BT313.	McKinley........	16.15	855	136	9	ND	0	859	3 pans, modern flood-plain, boulder-rich.
3/ 84BT317a	McKinley........	8.15	780	219	ND	ND	1	780	From Golden Eagle vug vein.
9054....	Cahoon..........	70.10	738	201	37	11	13	786	Channel sample 0.1 yd^3, on and in bedrock cracks.
9005....	Glacier.........	36.60	855	136	ND	ND	9	863	Channel sample 0.1 yd^3, 6 in gravel on bedrock.
85BT25..	Christmas.......	9.01	835	129	ND	ND	36	866	3 pans from auriferous till on bedrock.
9061....	Nugget..........	60.09	722	236	ND	ND	42	754	Channel sample 0.1 yd^3, fluvial gravel and till.
85BT29..	Nugget..........	28.40	756	207	ND	ND	37	785	3 pans, modern flood-plain, not on bedrock.
85BT28..	Cottonwood......	18.30	769	193	ND	ND	38	799	3 pans, modern flood-plain, not on bedrock.

ND - not detected.
1/Raw placer gold derived from channel and grab samples collected by Bureau and ADMG. All elements presented in parts per thousand; gold and silver determinations by commercial laboratories in Vancouver, BC, Lakewood, CO, and ADMG Mineral Laboratory in Fairbanks, AK. Zinc and lead were looked for but not detected.
2/'True Fineness' as defined by Boyle (2, p. 197) is the ratio of gold to gold plus silver times 1,000 or $\frac{Au}{Au + Ag} \times 1000$
3/Gold panned from 'hardrock' quartz-sulfide vein near Golden Eagle prospect (15).

The procedure for collecting reconnaissance placer samples consisted of processing, on the average, 0.1 yd^3 of gravel through a portable aluminum minisluice box. Where use of the box was not feasible, pans were used. (Sixteen slightly heaped 16-in. gold pans equal 0.1 yd^3 gravel.)

The procedure for channel placer sampling consisted of digging an approximate 1 ft x 1 ft channel from the top of a gravel section to bedrock (whenever feasible). The gravel taken from the channel was processed in 0.1 yd^3 increments through a hydraulic concentrator.

The procedures for taking site-specific bulk samples were dry screening, with 1-, 2-, and 4-in. mesh screens, 560 to 690 lb of gravel in the field. The +1, +2, and +4 mesh-size fractions were weighed, washed through a hydraulic concentrator, and discarded in the field. The -4 mesh fraction was bagged and shipped to the Bureau's processing lab in Anchorage. The samples were then dried and screened to -200 mesh size. Free gold in the +100 mesh and greater size fractions was separated by using a sluice and pan. Each mesh fraction of the gravel and recovered gold was weighed.

The concentrates from all the samples were saved and examined with a binocular microscope to identify heavy minerals present and the character of the gold. The concentrates have been retained for future chemical analysis.

RESULTS OF RECONNAISSANCE AND CHANNEL SAMPLING

Of the 78 reconnaissance and 53 channel samples collected, 35 contained values greater than 0.005 oz/yd^3 Au.

Results from reconnaissance and channel sampling were used to rate each stream's mineral-development potential for placer gold: 'high,' 'moderate,' 'low,' or 'unknown' (table 4). These ratings are estimates based on an evaluation of grades and extent of mineralization and other factors such as depth of overburden, presence of large boulders, and stream configuration.

A deposit of high mineral development potential would, by definition, have high grades (0.01 oz/yd^3 Au) and probable continuity of mineralization. A deposit of moderate mineral development potential would have either a high metal content or continuous mineralization identified but not both. A deposit with low mineral development potential would contain uneconomic grades or show little evidence of continuity of mineralization. For example, a placer deposit with grades below 0.001 oz/yd^3 Au would rank as low. Similarly, deposits containing less than 5,000 yd^3 would rank low unless their grade was very high. Unknown mineral development potential has been assigned to placer occurrences with little or no available geologic information.

Resource estimates were made for streams having moderate or high potential for placer-gold mineral development and for the Nugget and Porcupine creek fans. Resource estimates were derived by multiplying the length of the deposit being evaluated by the average width (as identified from available maps or from tape and compass traverses) by the average depth of the gravel. Average depths were based on trenching and historical data except in the case of the Porcupine and Nugget Creek fans, where assumed depths are used owing to lack of information. The results of these estimates are listed on table 4.

A summary of the sample results obtained from each drainage follows. Detailed information, sample data, and sample locations can be found in Hoekzema and others (1986).

Table 4. Mineral-development potential ratings and identified[a] resource estimates for drainages in the Porcupine mining area.

Drainage	Mineral development potential				Identified resources (yd^3)
	High	Moderate	Low	Unknown	
Big Boulder..........			X		ND
Cahoon...............		X			10,000
Christmas............		X			42,000
Cottonwood...........				X	ND
Glacier..............			X		ND
Klehini..............				X	ND
Little Boulder.......			X		ND
Little Salmon........			X		ND
McKinley	X				20,000
Nugget Channel.......		X			3,000
Alluvial fan..				X	2,000,000
Porcupine (lower)					
Channel..........		X			500,000
Bench............	X				152,000
Alluvial fan......				X	6,000,000
Porcupine (upper)....			X		ND
Summit...............			X		ND
Tsirku...............				X	ND

ND - not determined.
[a]Identified resources include auriferous gravels identified by the Bureau in 1985. Additional hypothetical resources probably exist.

Porcupine Creek

Porcupine Creek is a steep, rapidly downcutting drainage, with an average gradient of 350 ft/mi. Since 1898, over 77,000 oz gold has been produced from the creek and its tributaries. Reportedly, little gold was produced from Porcupine Creek above its junction with McKinley Creek (Beatty, 1937); this study, in six reconnaissance samples taken above the junction contained nondetectable to 0.0004 oz/yd^3 Au. The following discussion pertains to lower Porcupine Creek (below McKinley Creek).

Three categories of placer deposits occur on lower Porcupine Creek: abandoned channel and bench deposits, recent stream gravels, and an alluvial fan. Bureau sampling identified the highest grades in the abandoned channels and bench deposits. A much larger potential resource occurs in the alluvial fan, but grades are unknown.

Abandoned Channels and Bench Deposits

Figures 4 and 5 identify five gravel resource areas on lower Porcupine Creek blocked out on the basis of channel samples collected by the Bureau in 1985. These areas consist of abandoned channel and bench gravels, some of which correlate with old channels identified by Beatty (1937) and Bundtzen (1986) (fig. 3). These gravels are apparently quite young, as wood obtained from Beatty's 'dry channel' was dated at about 2,200 yr B.P. (table 2, sample 85BTc2; fig. 3).

The Bureau collected 12 samples from channels labeled as b, d, f, and g on figure 3. These samples contained from a trace to 0.021 oz/yd^3 Au. Another 38 channel samples were collected from abandoned channels and bench deposits located farther upstream in an old channel on figure 5 (area 5), and from bench deposits (44-49, 53-59) in areas 1 and 2 on figure 4. These samples contained from a trace to 0.058 oz/yd^3 gold. Concentrates contained 5 to 70 percent magnetite, up to 10 percent pyrite, and less than 1 percent zircon, garnet, and scheelite. The balance of the concentrates consists of rock fragments and quartz.

Samples collected indicate a collective identified resource in the five resource areas of about 152,000 yd^3 grading 0.0106 oz/yd^3 gold (table 5). These values are likely to be lower than actual values, as bedrock was not reached at all channel sample sites.

Additional resources are known to exist along upstream parts of Porcupine Creek but were not evaluated as part of this study. Some of these deposits, such as at Bear Gulch, have been mined, but unmined deposits that warrant further evaluation also remain.

Recent Stream Gravels

Present-day stream gravels consist of poorly to moderately well sorted gravels containing appreciable silt and boulders weighing up to several tons. These gravels have been worked with apparently good results.

The Bureau collected five samples from recent gravel deposits. These samples, which contained from a trace to 0.004 oz/yd^3 Au, are representative of surface values only. Because gold values in the Porcupine mining area are concentrated on bedrock, higher values should be anticipated at depth. The concentrates consisted of from 15 to 35 percent magnetite, 5 to 45 percent

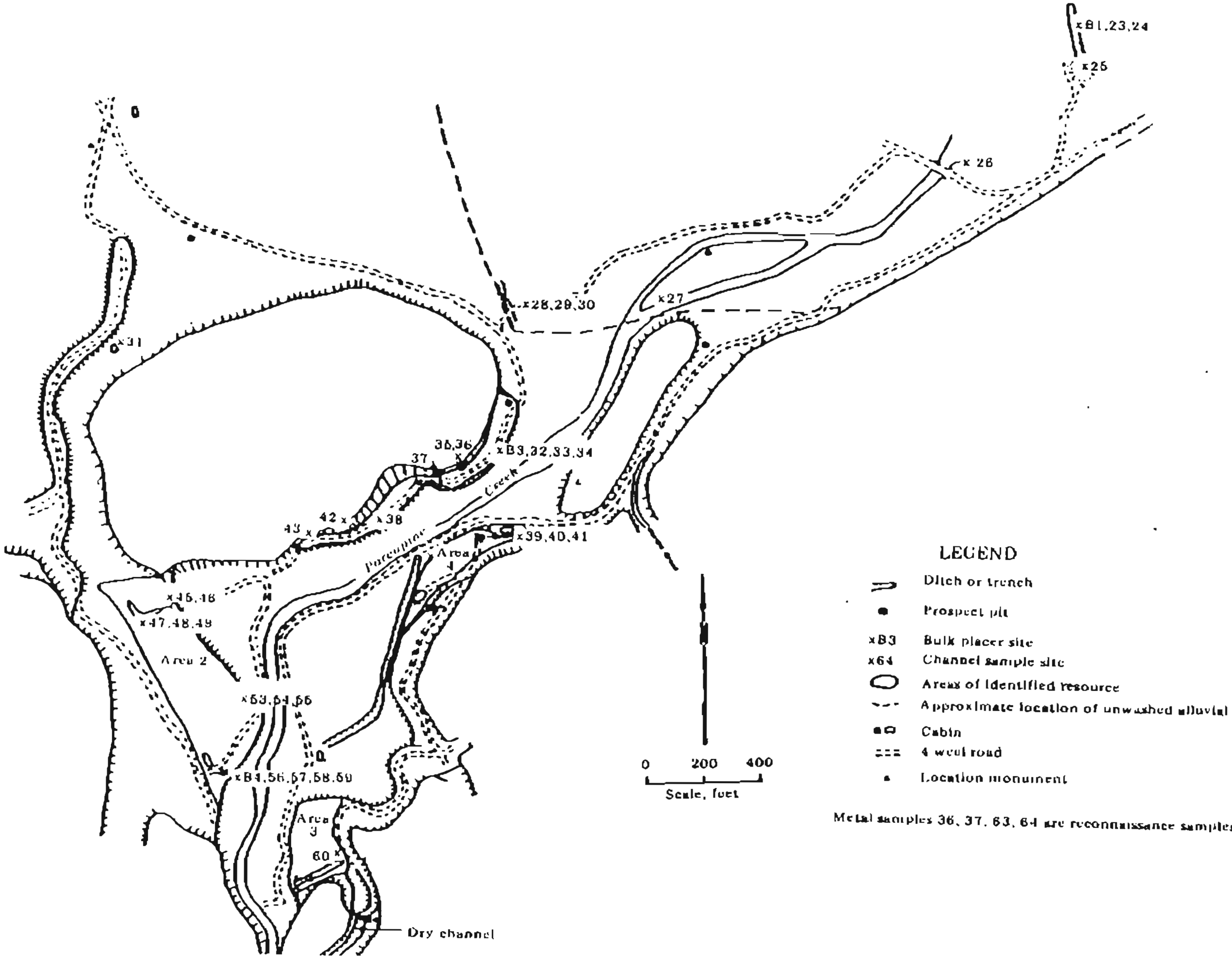

Figure 4. Placer sample locations, lower Porcupine Creek area.

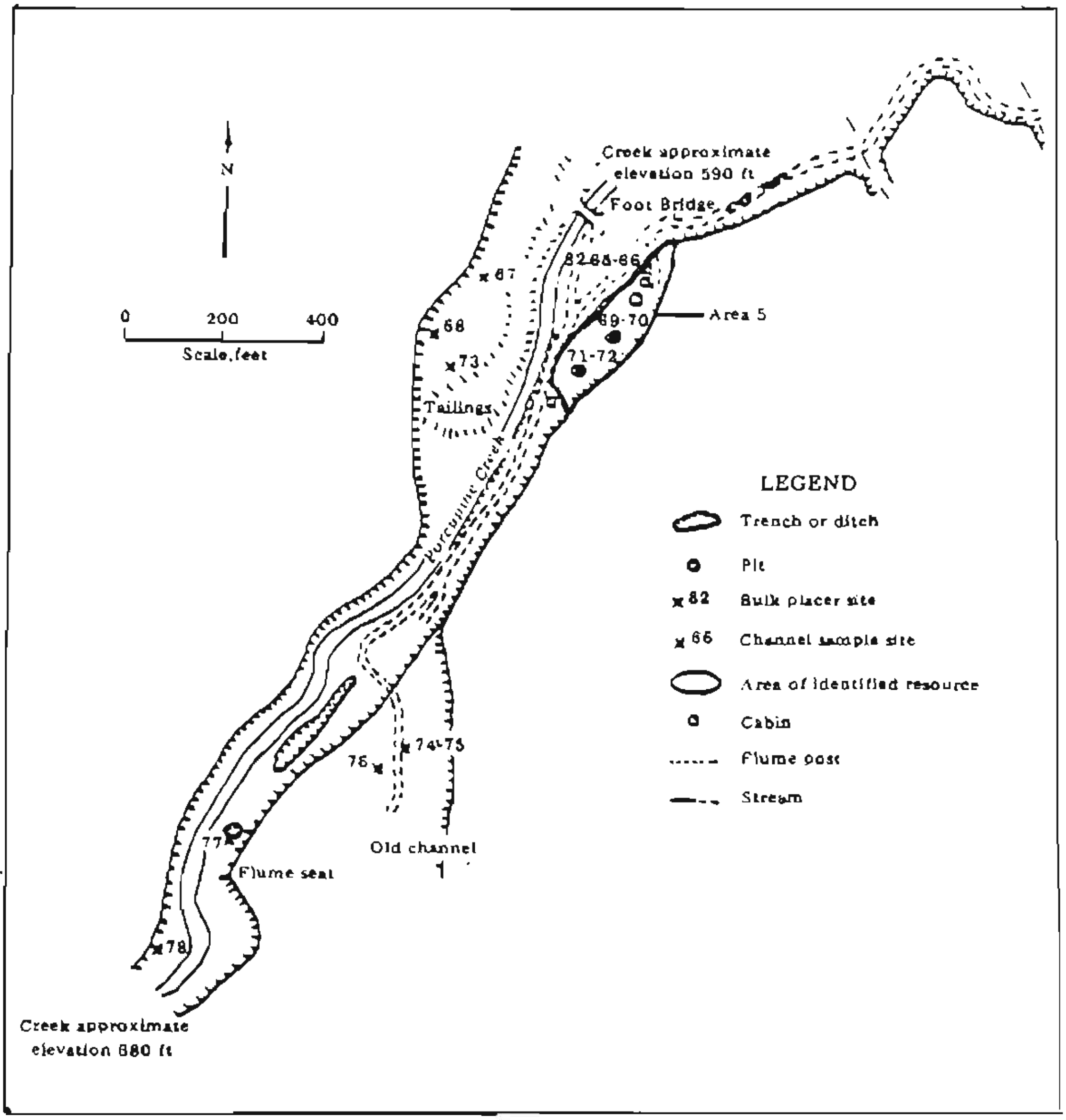

Figure 5. Placer sample locations, middle Porcupine Creek.

pyrite, and minor percentages of zircon, garnet, and scheelite. Results indicate that gold continues to be transported by Porcupine Creek during flood stages. The best values are concentrating just below McKinley Creek, which is the acknowledged source of most of the Porcupine Creek placer gold. The McKinley Creek junction area of Porcupine Creek has been mined several times in the past. Apparently, placer gold in this area reconcentrates periodically, depending on flood intervals. However, little gold appears to have been transported downstream to the fan area in recent years. Several thousand feet of stream bed, beginning about 1,000 ft below McKinley Creek and extending to the southern limit of the Beatty (1937) (fig. 3) investigation, have not been mined completely. This section is virtually inaccessible to heavy equipment, but suction dredging might be possible. The channel gravel of lower Porcupine Creek has an identified resource of at least 500,000 yd^3 of

Table 5. Identified resources in bench and abandoned channel deposits, Porcupine mining area.

Area	Figure	Volume (yd^3)[a]	Grade (oz/yd^3 Au)[b]	Samples
1	5	21,000	0.0215	B3, 32-43
2	5	75,000	0.0087	B4, 44-59
3	5	23,000	0.0106	60-63
4	5	20,000	0.0038	39-41, 50, 51
5	6	13,000	0.0145	B2, 65-72
Total		152,000	0.0106	

[a]Volumes were calculated by multiplying the surface area of the block by a thickness based on field information (if available). Thickness figures used tended to be minimum values.
[b]Grades were calculated by averaging the grades determined for each channel. No weighting factors were used. These values are likely to be lower than the actual values, because bedrock was not reached at every sample site. However, gold values are distributed throughout the gravel. Best values are correlated with coarse gravel layers.

unknown grade based on an average thickness of 18 ft and an average width of 90 ft (table 4). Actual thickness of mined section is reported to have exceeded 40 ft in some locations (Roppel, 1975).

Alluvial Fan

The alluvial-fan gravels consist of 12 to 15 ft of recent stream gravels lying over an unknown thickness of older gravels. Old channels correlative with older abandoned channels along Porcupine Creek are believed to occur beneath the fan. To date, these potentially gold-bearing channels have not been identified. Some drilling reportedly occurred in the early 1900s, but results are unknown. Rumors suggest that bedrock was encountered at a depth of 70 ft in at least one hole.

The Bureau collected eight samples on the alluvial fan. However, these are mostly representative of recent surface gravels and with older channel deposits that may exist at depth. However, the results were encouraging, for the samples recovered from a trace to 0.011 oz/yd^3 Au. The concentrates contained magnetite (up to 40 percent), garnet, zircon, and minor pyrite and scheelite.

The Porcupine fan contains over 6 million yd^3 of potential resources based on a length of 2,400 ft, width of 1,800 ft, and depth of 40 ft (table 4). Much of this volume will likely prove to be uneconomic to mine. However, high-grade channels might exist at depths of less than 100 ft. Additional evaluation of this resource is warranted.

McKINLEY CREEK

McKinley Creek, the largest northwest-flowing tributary of Porcupine Creek, has a gradient of 500 ft/mi. A lode-gold deposit is located next to the creek at an 1,800-ft elevation about 2 mi above its junction with Porcupine Creek. Free gold can be panned from the sulfides in the lode deposit.

Bureau reconnaissance samples collected above the lode deposit contained from less than 0.0004 to 0.0056 oz/yd^3 Au. Samples taken below the lode deposit contained from less than 0.0004 to 0.0539 oz/yd^3 Au. The concentrates contained up to 30 percent magnetite and 10 percent pyrite, with minor zircon, garnet, and scheelite.

Identified resources consist of narrow point-bar deposits and channel deposits of from a few hundred to 2,000 yd^3 each. About 20,000 yd^3 grading from 0.001 to 0.054 oz/yd^3 Au occur on McKinley Creek along a 1.75-mi stretch above its junction with Cahoon Creek. Additional resources exist below Cahoon Creek, but this section has been mined several times and grades of the remaining gravels are unknown.

CAHOON CREEK

Cahoon Creek is a steep (gradient, 650 ft/mi), northeast-flowing tributary to McKinley Creek. Very little gravel is present in the creek channel; much of the stream flows on bedrock. Cahoon Creek has been recognized by miners as a source for the gold on McKinley and Porcupine Creek. The lower 0.5 mi of the creek has been extensively worked.

Steep terrain and the presence of large amounts of brush precluded sampling of the lower 1 mi of the creek. Sampling was also impeded by a lack of gravel. The nine samples taken indicate that the gold concentration increases as the junction with McKinley Creek is approached. The samples contained from less than 0.004 to 0.045 oz/yd^3 Au. The concentrates contained greater than 70 percent magnetite, with minor pyrite, zircon, and garnet.

Channel gravels in Cahoon Creek are limited (table 4). Some potential for abandoned channels or bench deposits may exist, but these have generally been covered or diluted with colluvium and avalanche debris. The channel gravels might be successfully mined with suction dredges, especially along the lower 1.5 mi of the creek. An abandoned channel of Cahoon Creek, which joins McKinley Creek about 0.25 mi upstream from the current junction, should be investigated.

NUGGET CREEK

Nugget Creek flows south into the Tsirku River at an average gradient of 900 ft/mi. Placer deposits are present as alluvium or colluvium in the stream bottom, as abandoned channel deposits at high elevations on the east side of the creek, and as an alluvial fan at the mouth of the creek. Alluvium in the lower canyon of the creek is from 12 to 20 ft deep. Gold is found on or near bedrock, with little gold found in the overlying gravels.

The Bureau collected 11 reconnaissance samples from Nugget Creek and its alluvial fan. The best value (0.0138 oz/yd^3 Au) was in a sample collected at the mouth of an abandoned channel of Nugget Creek adjacent to the Tsirku River. Only minor amounts of gold (trace to 0.0007 oz/yd^3) were found in the creek itself. A sample collected from a hydraulic cut at 2,550 ft elevation on the east side of the creek contained 0.0006 oz/yd^3 Au. Concentrates contained from 25 to 70 percent magnetite, from less than 1 to 70 percent pyrite, and minor percentages of zircon, garnet, scheelite, and galena.

Gravel resources in the existing stream channel are minimal but have been shown to contain coarse gold by recent suction-dredging operations. The alluvial fan contains an estimated 2,000,000 yd^3 of material, but the grade remains unknown. Only parts of this volume would be minable because high grades would likely be restricted to channels.

COTTONWOOD CREEK

Cottonwood Creek is a steep (gradient, 750 ft/mi), southeast-flowing tributary of the Tsirku River, located about 1 mi west of Nugget Creek. Encouraging amounts of gold have been found in the creek, but no extensive mining has been done.

The Bureau took three reconnaissance samples from Cottonwood Creek and found from less than 0.0004 to 0.0005 oz/yd^3 Au. Concentrates contained from 10 to 20 percent magnetite, up to 10 percent pyrite, and minor percentages of garnet, zircon, and scheelite.

Gravel resources in the creek channel are very limited owing to the steep gradient and narrow bedrock canyon. A significant volume of untested alluvium exists in the alluvial fan at the mouth of the creek. This fan coalesces with the Nugget Creek fan. Abandoned channels have been identified in the fan between Cottonwood and Nugget Creeks and should be investigated.

GLACIER CREEK

Glacier Creek is a northeast-flowing tributary of the Klehini River and is located about 2 mi west of Porcupine Creek. The creek is not as steep as most of the creeks of the area, with an average gradient of 250 ft/mi.

The Bureau's reconnaissance sampling of Glacier Creek found no significant recoverable gold values in seven samples collected. The concentrates contained up to 70 percent sulfides (mostly pyrite), 10 percent magnetite, and minor garnet, and zircon.

Glacier Creek contains a significant gravel resource. However, no evidence of recoverable gold values in these gravels exists. Christmas Creek is the only auriferous tributary to Glacier Creek identified to date.

CHRISTMAS CREEK

Christmas Creek is a small, steep (gradient, 1,000 ft/mi) north-flowing eastern tributary of Glacier Creek.

The Bureau collected four reconnaissance samples from gravels exposed in the mining cut near the junction of Christmas and Glacier Creeks. The gravels are clay rich, unsorted, and are interpreted to be in part, glacial till. Results indicate that there is a relatively equal distribution of gold through 8 ft of gravel. The grade of the gravel averages 0.0065 oz/yd^3 Au. The concentrates contained magnetite, zircon, garnet, minor pyrite, and scheelite.

Identified resources are largely restricted to the lower 0.5 mi of the creek. The lowermost section of the creek near the workings, contains about 12,000 yd^3 of identified resource grading 0.0065 oz/yd^3 Au. An additional resource of up to 30,000 yd^3 is estimated to occur farther upstream (table 4).

RESULTS OF SITE-SPECIFIC BULK PLACER SAMPLING

Four site-specific bulk placer samples (B1-4) were collected from previously unworked gravels on Porcupine Creek for analyzing gravel and gold particle sizes. Because of the disseminated nature of most placer gold within a gravel deposit, the gold from the channel samples taken at the site-specific sample locations was also screened and weighed. The weights of the gold recovered from the site-specific samples were added to the weights recovered from the site specific samples to reflect a larger sampling volume (table 6). Because of this, the totals in table 6 cannot be used to calculate grades. Histograms of the percentages of gravel and gold in the mesh sizes are shown on figures 6-10.

A 604.05-lb sample (B-1) was taken from a 10-ft-thick interval of alluvium on the Porcupine Creek alluvial fan (fig. 4). Over half of the gravel is greater than +1 mesh in size. Gold was found in mesh sizes between -14 and +100, with over 88 percent in the -14 to +50 mesh sizes (fig. 6).

A 584.25-lb sample (B-2) was taken from a gravel bench along Porcupine Creek (fig. 5). The sample was taken from 12 ft of alluvium resting on slate bedrock. Over half of the gravel is coarser than +1 mesh in size. Over 90 percent of the gold was from -10 to +30 mesh (fig. 7).

A 562.4-lb sample (B-3) was taken from 16 ft of alluvium on an abandoned channel of Porcupine Creek (fig. 4). Nearly 65 percent of the gravel is coarser than +1 mesh. Over 95 percent of the gold was -10 to +50 mesh (fig. 8).

A 689.9-lb sample (B-4) was taken from alluvium along Porcupine Creek (fig. 4). The sample was taken from 13 ft of gravel. Nearly 58 percent of the gravel is coarser than +1 mesh. Over 90 percent of the gold was -10 to +60 mesh (fig. 9).

Figure 10 is a graph of the cumulative results for all four site-specific samples. The graph indicates that over 90 percent of the gold is -10 to +50 mesh and that over 55 percent of the gravel is greater than +1 mesh.

Table 6. Results of site-specific bulk placer samples collected from lower Porcupine Creek.

Sieve size (mesh)	Sample B-1 Gravel weight (lb)	Sample B-1 Gold weight (gm)	Sample B-2 Gravel weight (lb)	Sample B-2 Gold weight (gm)	Sample B-3 Gravel weight (lb)	Sample B-3 Gold weight (gm)	Sample B-4 Gravel weight (lb)	Sample B-4 Gold weight (gm)
+1........	308	0	300	0	360	0	395	0
+2........	33	0	40	0	22	0	42	0
+4........	70	0	78	0	54	0	79	0
+6........	17.25	0	10.75	0	10.5	0	14	0
+10.......	41	0	35.75	0	28	0	42.8	0
+14.......	20	0	17.6	0.0989	12.75	0.0824	19.4	0.0405
+20.......	20	0.0025	17.5	0.0208	11.8	0.0206	18	0.0314
+30.......	18.75	0.0060	15.75	0.0475	10.75	0.0654	16	0.0322
+40.......	16.5	0.0049	13	0.0078	9.75	0.0294	13.4	0.0117
+50.......	16	0.0028	8.5	0.0094	8.5	0.0202	11.25	0.0163
+60.......	6.75	0.0007	4.8	0	4	0.0051	4	0.0038
+70.......	5.25	0.0004	3.6	0.0002	3.2	0.0018	3.25	0.0010
+80.......	4.8	0.0006	3.5	0	3	0.0025	2.75	0.0026
+100......	5	0.0005	4.25	0.0004	3.5	0.0005	3.25	0.0016
+200......	11.75	0	17.25	0	11.4	0	13.4	0
-200......	10	0	14	0	9.25	0	12.4	0
Total	604.05	0.0184	584.25	0.1850	562.40	0.2279	689.90	0.1411

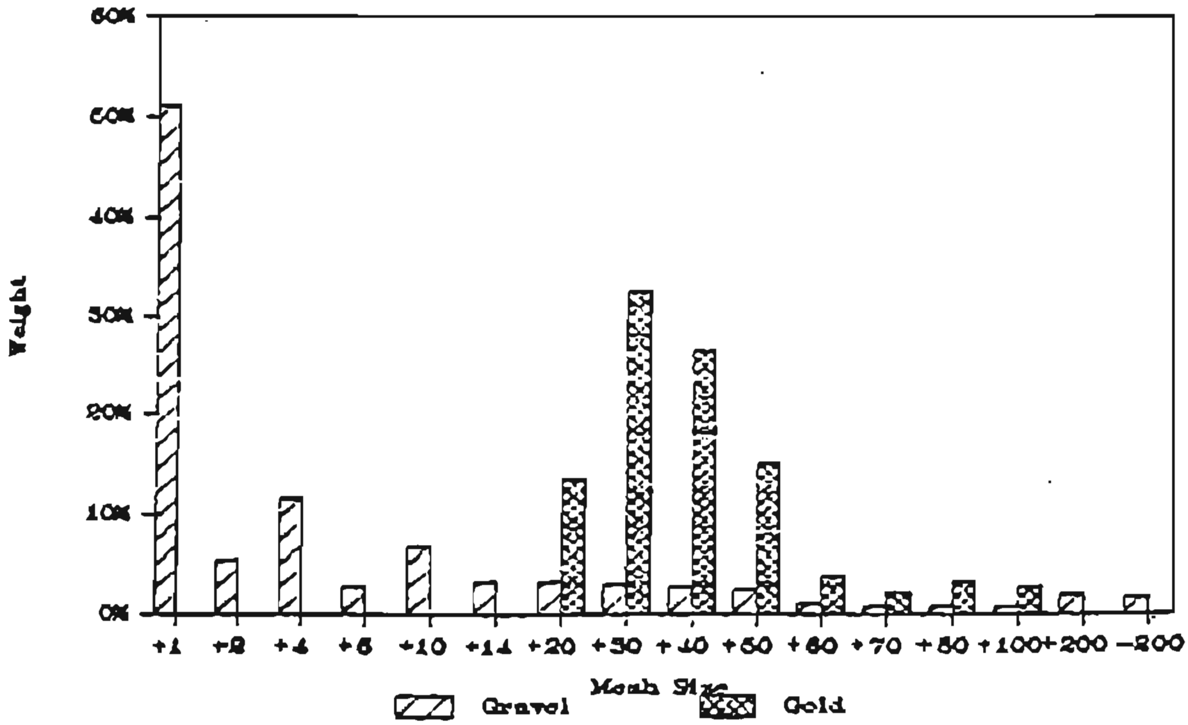

Figure 6. Histogram of sample B-1.

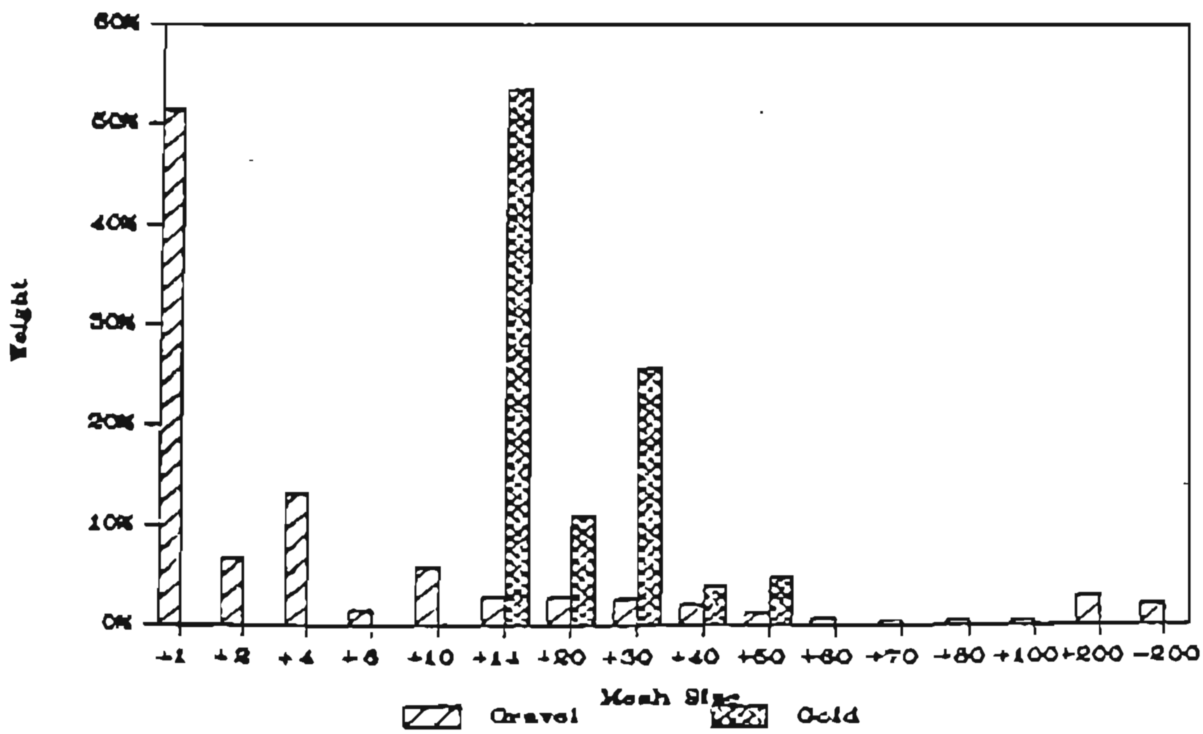

Figure 7. Histogram of sample B-2.

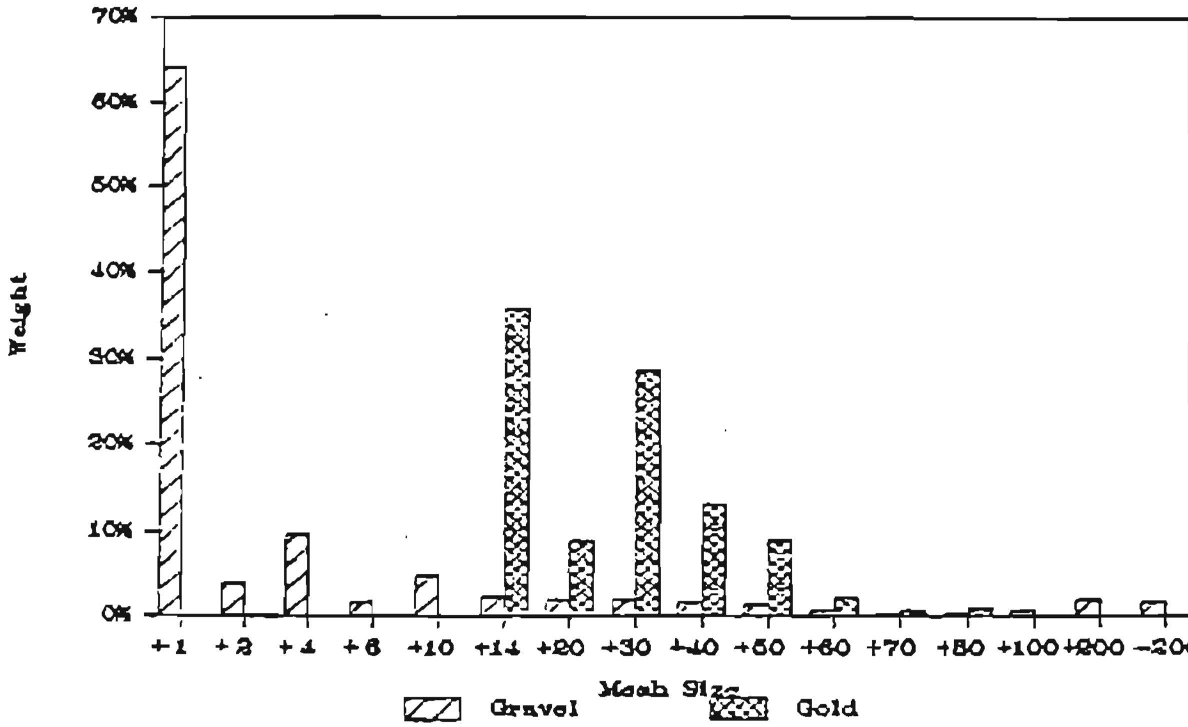

Figure 8. Histogram of sample B-3.

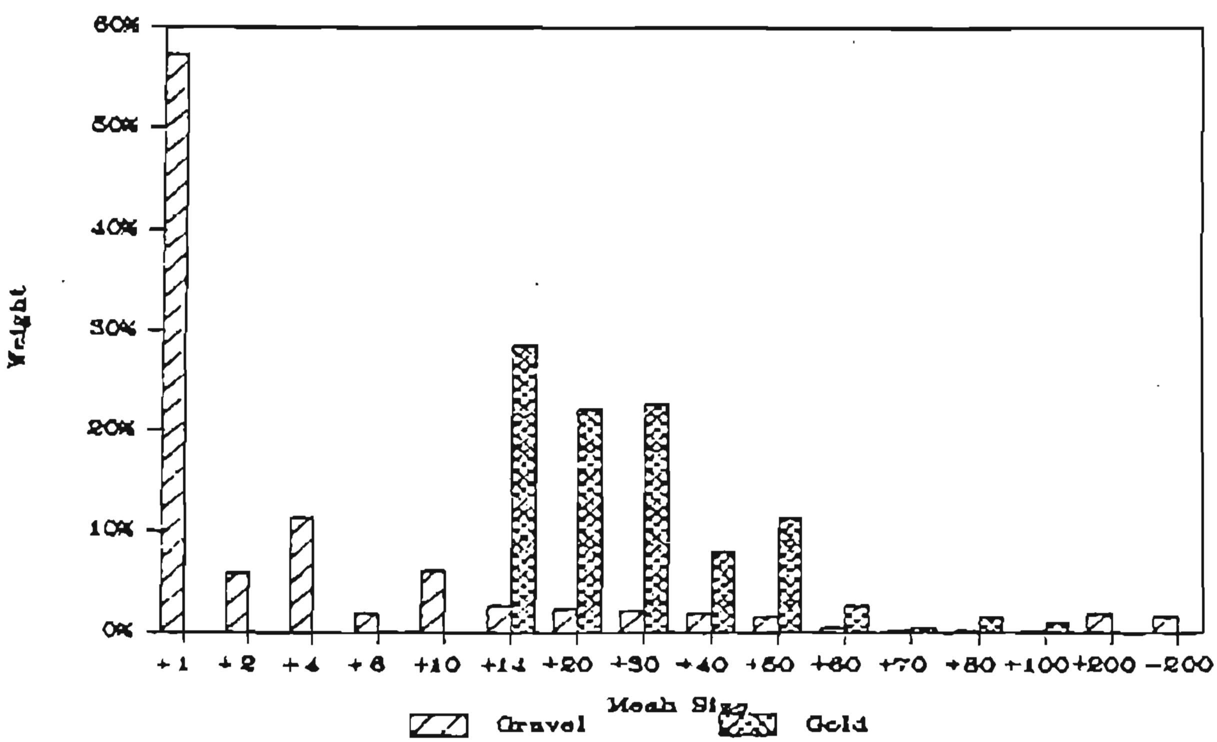

Figure 9. Histogram of sample B-4.

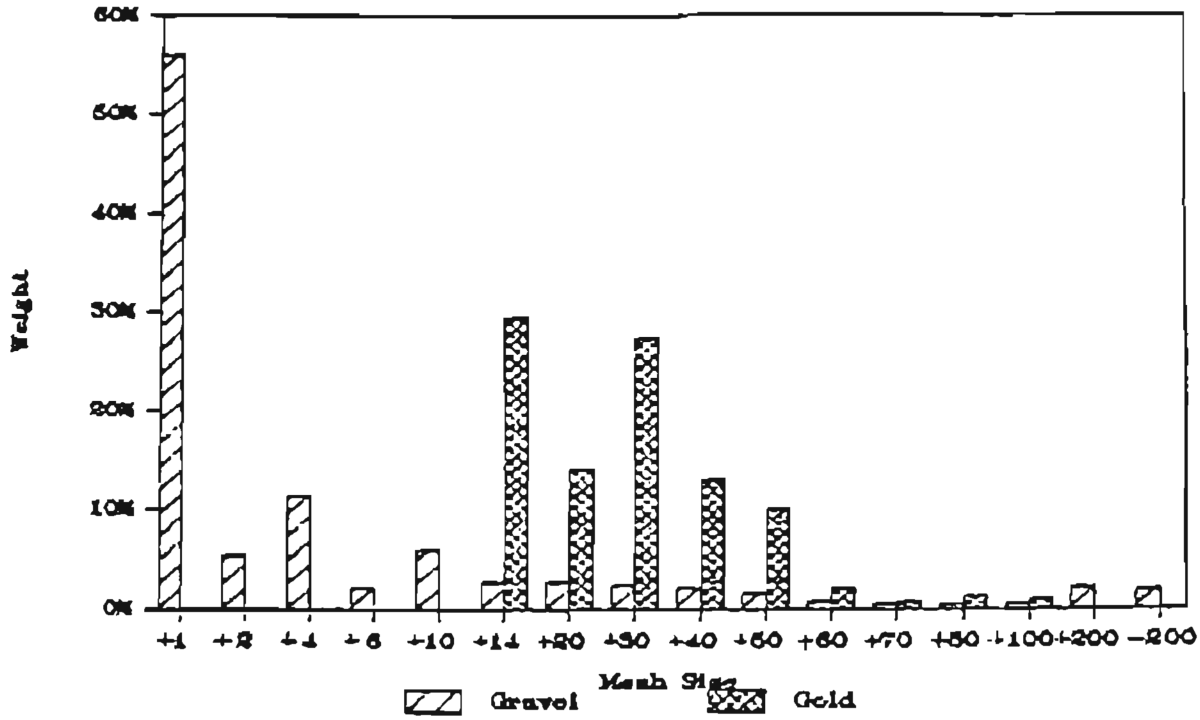

Figure 10. Cumulative histogram of bulk samples.

SUMMARY

The Bureau conducted reconnaissance and site-specific bulk placer sampling in the Porcupine mining area in 1985. This work resulted in identifying gravel deposits having moderate to high mineral-development potential on lower Porcupine, Cahoon, Christmas, McKinley, and Nugget Creeks.

Abandoned channel and bench deposits on lower Porcupine Creek have the best potential for supporting a small to medium-sized (500 to 1,000 yd^3/day) heavy-equipment placer operation. However, the prospective developer should identify a resource having average grades nearly double those identified by this study (i.e., 0.01 oz/yd^3 Au) before making a substantial investment. Bureau records indicate that successful operators in Alaska during 1980-85, using heavy equipment to mine at the above rates, mined ground averaging 0.015 oz/yd^3 Au. A 1-mi-long section of McKinley Creek above Cahoon Creek has high mineral-development potential for small placer operations using suction dredge and hand-placer techniques. Moderate development potential for small heavy equipment (50 to 500 yd^3/day) or hand-placer operations exists on Christmas and Nugget Creeks. The greatest potential for future mining on a large scale in the area depends on the results of exploring the alluvial fans of Porcupine and Nugget Creeks, which together conservatively contain over 8,000,000 yd^3 of gravel resources. Site-specific samples collected from lower Porcupine Creek indicate that washing plants should screen to -1 mesh and be designed to recover gold down to +80 mesh.

The DGGS investigated and mapped the Quaternary geology and placer deposits of the Porcupine mining area and identified the fineness values of gold samples collected from the study area. The average overall fineness of placer gold from the Porcupine mining area is 837. Dating of organic material collected from bench deposits indicate that the Porcupine placers are less than 3,000 yr old. Glacial features suggest four stages of glacial advance within the past 13,000 yr.

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POSTER TOPICS AND SUMMARIES

"Poster sessions allow you to use innovative ways to get your message across"

Cartoon from Geolog (Newsletter of the Geological Association of Canada), Winter 1987, v. 16, pt. 1, p. 20.

This section includes a brief summary of the poster displays presented at the Ninth Annual Placer Conference held on March 25-27, 1987 in Fairbanks, Alaska. Those posters which were previously published are listed at the end of this report as selected publications.

MINING AND THE UNIVERSITY OF ALASKA MUSEUM: CONTRIBUTIONS TO ALASKA'S FOSSIL HERITAGE

by

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and

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In 1980, the University of Alaska Museum was dedicated to Otto William Geist. This German immigrant was a self-taught archaeologist, paleontologist, and naturalist. His noteworthy contributions in paleontology, through his exhaustive collecting efforts, were instrumental in establishing Alaska as one of the richest Pleistocene mammal fossil locales in North America.

Fifty years ago, the Fairbanks Exploration Company had extended its mining activities along many of the creeks in the Fairbanks area. The hydraulic giants used by the miners often exposed Pleistocene fossils. In 1937, Geist directed fossil collecting in the Fairbanks area for the University of Alaska and the Frick Laboratory in New York. Geist and his assistants collected many tons of fossils each mining season. During the winter months he classified, repaired, and preserved thousands of specimens at the university before shipping many of them to the Frick Laboratory.

Geist collected more Pleistocene fossils in the subarctic and arctic than any other American, and his discoveries changed vertebrate paleontology in North America. His relationship with the mining community in interior Alaska established the important connection between mining and vertebrate paleontology. This historic cooperation between the University of Alaska Museum and miners across Alaska has been the basis for some of the most important discoveries in vertebrate paleontology.

In 1979, Walter and Ruth Roman discovered the mummified carcass of a Pleistocene bison at their placer mine on Pearl Creek (Fish Creek). The bison was exposed as it eroded from the muck deposit. The Romans contacted the University of Alaska and a field crew directed by Dr. Dale Guthrie excavated the carcass. Analysis of this specimen indicates that the bison, *Bison priscus*, roamed the Fairbanks region 36,000 yr ago. The bison, now on display at the UAF Museum, is the only restored Pleistocene bison carcass in the world.

In 1982, Ron Rosander and family discovered a set of mammoth tusks protruding from a muck deposit at their mining claim on Colorado Creek. After carefully excavating the tusks, Rosander realized he had discovered the entire skull with intact tusks of a woolly mammoth, *Mammuthus primigenius*.

Rosander notified the museum, and a field crew was sent to Colorado Creek to prepare the skull for shipment to Fairbanks (see photograph on frontispiece).

The following summer, a field crew excavated the site and found many articulated bones belonging to the mammoth. In addition, bones from a second mammoth were discovered. This material, which is 15,000 yr old, is being analyzed at the university and will provide another glimpse into the past. The Colorado Creek mammoth skull is also on display at the UAF Museum.

Indoubtedly, many fossils are still embedded in the muck deposits throughout Alaska. In the future, the relationship between the mining community and the University of Alaska Museum will provide the field of vertebrate paleontology with many more exciting discoveries.

The University of Alaska Museum extends heartfelt thanks to the mining community for its contributions to vertebrate paleontology and the preservation of Alaska's fossil heritage.

SUPERGENE GOLD IN GOLD PLACERS

by

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PROPOSAL

Recent work suggests that part of the gold in gold-placer deposits is of supergene origin. Theoretical and experimental work have established that Cl-, CN-, SO-, and other anions will form complexes with gold under certain Eh (oxidation potential) and pH conditions at surface temperatures and pressures. Freeze-thaw cycles also play a role in this process. When soil begins to freeze, solutes are exuded into the unfrozen fraction of the soil pore water. This concentrates the complexing agents and favorably affects both the pH and the Eh. These processes may ultimately explain the abundance of placer deposits in the subarctic.

I propose an integrated study of a natural alluvial placer to determine if much of the gold is of nonalluvial origin. The study will take advantage of the benches developed on gentle slopes of asymmetrical valleys, a geomorphic feature common in central Alaska. These benches, which contain alluvium deposited by the axial stream, are the result of sustained, simultaneous downcutting and lateral migration of the axial drainage. The higher benches are successively older, providing a semicontinuous geological record at least 1 million yr.

I propose to map and sample the late Cenozoic deposits on these benches during a mining season. The provenance and sedimentology will be mapped for sample control and future correlations. The surface morphology of the gold grains will be studied by scanning electron microscope (SEM) to determine the relative importance of mechanical deformation, chemical solution, and chemical deposition. These grains will then be examined with an electron microprobe to determine the internal trace-element zoning pattern. A small number of samples will be analyzed to determine the hydraulic equivalence of the gold grains relative to the other mineral grains. These results will be used to determine the relative amount of gold in each sample originating from alluvial detrital transport, from recent addition of residual gold grains from underlying bedrock, and from chemical deposition from groundwater.

This study, undertaken as Master of Science thesis research, will begin in June 1987. Results should be available in the spring of 1988.

POSTMETAMORPHIC QUARTZ-STIBNITE-GOLD LODES

by

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Quartz-stibnite-gold mineralization is commonly found worldwide in greenschist-facies metamorphic rocks. These deposits occur in belts ranging from 100 to 150 km long and 30 to 60 km wide between the pumpellite-out and biocite-in metamorphic isograds. The deposits show no direct relationship to an igneous source and are probably a product of metamorphism.

The veins, always postmetamorphic, cut metamorphic fabrics. The deposits can be divided into two types, schist-marble contact deposits and discordant vein types. Common features of the schist-marble contact deposit are:

1) Mineralization localized at the intersection of high-angle faults and schist-marble contacts

2) Formation of both breccia and jasperoid prior to mineralization

3) Multiple periods of mineralization

4) Mineralization consisting of quartz-stibnite-gold with little or no base metals present.

Common features of discordant vein type deposits are:

1) Host rocks of schist or quartzites

2) Mineralization localized in postmetamorphic high-angle faults or shear zones

3) Mineralization consisting of quartz-stibnite-gold with little or no base metals present.

In Alaska, many districts show quartz-stibnite-gold mineralization, including the Nome district, the Sukakap Mt. area of the Brooks Range, and the Nolan-Wiseman district. Several areas within the USSR also contain this type of mineralization (the Kadamzhai deposit, the Terek deposit, the Dzhizhikurt deposit, and the Sarylakh deposit). In Alaska a systematic exploration program could result in new discoveries of postmetamorphic quartz-stibnite-gold lodes.

POTENTIAL OF DIGITIZED AERIAL-PHOTOGRAPH ANALYSIS FOR PLACER-GOLD EXPLORATION

by

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Before the 1960s, aerial photography was the sole system used in remote sensing. Since the advent of the NASA space program, technology has been moving from the traditional photographic method to computer-aided image processing. Today, many of the major advances of satellite digital image-processing techniques are widely used by scientists and engineers in search of natural resources.

Digital technology is the key factor in nonphotographic remote-sensing systems. The massive amount of information obtained often dictates data use as well as interpretation. There are potential applications of the image scanner (Eikonixscan 78/79) in placer gold exploration. The vidicon camera at the UAF Digital Image Processing Laboratory converts an aerial photograph into digital form with spatial resolution of 2,048 by 1,728 pixels with an eight-bit radiometric resolution. The digitized photograph is then processed by computer and interpreted by visual identification. In our study, a high-altitude aerial photograph was scanned and digitally processed. Several contrast enhancement methods were used to improve the quality of digital data and to extract the needed information.

PLACER MINING AND RECOVERY SYSTEMS

by

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Placer mining is the process of recovering minerals concentrated by erosional processes. It involves moving large quantities of material and then using specific gravity to further concentrate minerals for recovery. This paper discusses a typical placer mining and recovery system observed in Australia.

Overburden, the material overlying the pay gravels, must be moved to allow mining. Once the overburden has been removed (usually by bulldozer), the pay gravels are moved to the processing site.

The pay gravels are fed into a grizzly by using a constant-feed conveyor, which prevents 'choking' of the grizzly and provides a uniform feed to the processing plant. The gravels are washed as plus 6-in. rock passes across the grizzly and onto a conveyor belt that leads to a tailings filter.

The smaller rock is directed to the trommel mouth. As the material moves down the trommel, it will be washed and tumbled, breaking up any existing clay and liberating any attached gold. The resultant plus 3/4-in. gravels are discharged from the end of the trommel and also funneled to the tailings filter.

Residual material passing through the screens of the trommel are then discharged into the three sluice boxes, according to size.

The first size discharged is minus 1/8 in., the second size is plus 1/8 to 1/4 in., and the third is plus 1/4 in. to minus 3/4 in. These sizes were selected to permit a lower mechanical energy by the water moving through the boxes and therefore a greater fine-gold recovery.

UAF Mining Extension students visiting mines in Australia observed placer recovery systems used in semidesert conditions. Water is scarce, and the miners must recover as much of it as possible. Often, Australian miners use their gravel tailings as filters for their mining water. They run their discharge water into a U-shaped settling pond and pass it through the filter pile into a holding pond for further use.

For more information on touring Australian mines or other Mining Extension courses, contact Jim Madonna, UAF Mining Extension, Fairbanks.

EVALUATION OF FINE-GOLD-RECOVERY EQUIPMENT

by

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INTRODUCTION

Often, the most difficult part of placer mining is the recovery of fine gold from black-sand concentrates. Gravity separation in a rapid, efficient manner continues to be a major problem. This project tested the shaker table, the gold wheel, and an automatic panner under controlled conditions to determine which type of gold recovery unit is best suited to the collection of fine gold from concentrates.

TEST SAMPLE PREPARATION

The test sample consisted of actual placer concentrates that had been processed for gold without sizing. These concentrates were screened to -20/+30 mesh and panned to eliminate all visible gold. This mesh size was selected because -20/+30 mesh gold can be separated easily by gravity separation, whereas finer gold (-30 mesh) requires amalgamation or separation by other sophisticated processes.

The sample was sterilized, dried, and sized to -20/+30 mesh. Ten lb of concentrate were weighed, and three dwt (4,665 g) was returned to the sterilized concentrate for machine-recovery testing. The sample was wetted with water for the tests.

THE TESTS

The shaker table was the first machine tested. Shaking tables consist of an inclined deck or table onto which material is fed and washed. The table shakes with an asymmetrical motion at right angles to the flow of material, which vibrates across the riffle and separates by specific gravity. Materials can be collected as tailings, middlings, and concentrate. The shaker table was cleaned by running water through and over it. The angle of the table was set to direct only the highest specific-gravity material into the concentrate bin for further panning. The troughs were channeled through hoses into buckets to save the entire sample for reprocessing through other machines. After washing the 10 lb sample, maximum efficiency was obtained only when black sands (magnetite and pyrite) accompanied the gold directed to the concentrate catch bin. Gold passed into the middling bin, necessitating a rewash. Also, small amounts of concentrate and gold washed up under the table and accumulated on the lip of the middlings and concentrate trough. Cleaning these areas was not a problem, but caution had to be taken to prevent gold loss. The concentrates from the shaker table required final panning because of the amount of black sand in the final concentrate bin. The feed was slow, and all material was rerun for absolute maximum gold recovery; 79.3 percent of the gold was recovered in the concentrate collector.

The second machine tested was the gold wheel. The gold wheel is a portable variable-speed bowl with a dual-lead spiral rotating to a center deposit tube; it has an adjustable spray bar for water feed and wash. Material is slowly fed into the face of the machine, and the rotation and water combine to permit the riffle (spiral) to collect high-specific-gravity material. The highest such material (gold) separates out and travels to the center as the bowl rotates; it is deposited through a central tube into a collection basin. The machine angle can be adjusted to include or exclude material as desired by the operator.

The gold wheel was cleaned and checked for proper operation. The water feed was started and the 10-lb concentrate sample was slowly fed into the face. The machine was simple to run, easy to adjust, and very efficient. The gold in the 10-lb sample could be observed as it traveled to the center of the spiral and into the recovery tube. The gold recovered was very clean and ready to be dried; only a few particles of black sand were collected. The machine was operated for a few minutes after the full 10-lb sample had been processed to ensure complete collection. The 1- to 2-lb load in the machine was then panned to see if gold remained. The machine collected 95.5 percent of the gold.

The third machine tested for concentrate cleaning was the automatic panner. This machine consists of a feed chute with pumped water sprayed directly onto a virbrating 0.5-in. sizing screen and into a series of deep riffles that shake horizontally while the machine operates. The machine was cleaned by hosing out the feed chute, screens, and riffles. Collection receptacles were placed as necessary to catch the material washed through the unit. A screened sample of concentrates similar (-20/+30 mesh) to the 10-lb sample, without gold, was fed into the machine to observe the action of the machine. The automatic panner washed some of the sample out of the +½-in. chute; most of the concentrate was accumulated in the riffles of the sluice box. Very little concentrate washed through the sluice box. The 10-lb sample with gold was not run. The machine was cleaned up. We determined that this machine is actually a large, semiportable power sluice box best suited for a small mining operation to process bench gravels or to clean up large-scale sluice gravels.

CONCLUSIONS

The shaker table reclaimed 79.3 percent of the gold in the -20/+30 sized concentrate sample with an accumulation of black sands that had to be panned out by hand. The machine should be mounted permanently because of the shaking action. A large quantity of water of a consistent flow is necessary for best washing results.

The gold wheel reclaimed 95.5 percent of the gold in the concentrate sample. However, monitoring is necessary as adjustments are sometimes needed for optimum recovery. The gold recovered was very clean and only a slight gold-pan washing was necessary to remove the remaining black sand. The remaining 1 to 2 lb of concentrate material contained 0.042 dwt, or 1.4 percent of the gold sample. This was clearly the most efficient, practical, and portable machine tested.

The automatic panner failed the test as a concentrate cleaning unit. This machine is better described as a power sluice box with water pump. Concentrates would have to be further panned or processed to obtain the gold.

HEAP LEACHING IN ALASKA

by

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This paper discusses a hypothetical heap-leaching operation designed for use in Alaska.

The ore of this heap leach operation will be mined from a 100- by 50-ft mineralized zone by using open-pit excavation methods. The ore is applicable to heap leaching, as it is porous and contains microscopic and submicroscopic gold (Au) and silver (Ag) particles in conjunction with a low-clay and fine-silt content that presents no difficulty in percolation of the leach solution. The optimum ph of 7 for the leachate will be maintained by using sodium hydroxide. Channel assays taken across the mineralized zone hypothetically average 0.165 oz/ton Au and 0.06 oz/ton Ag.

The ore will be mined at 50 tons per day with a Cat D-8 with ripper and a Cat 966 front-end loader. The D-8 will rip the ore-bearing rock and then push it into piles accessible to the 966 loader for loading into a 5-yd end dump truck. The truck will haul the material from the pit to the feed bin, 1/4 mi away. The truck will dump directly into a 10- by 10- by 10-ft feed bin with a holding capacity of about 50 tons. The ore will be fed into a high-volume (36- by 18-ft) jaw crusher by a plate feeder at 2.2 tons per hr.

The crushed ore from the jaw crusher will be passed across a 2- by 4-ft shaker screen with 2-in. openings. The plus 2-in. fraction will be returned to the unit by conveyor for additional crushing. The minus 2-in. fraction will be fed by conveyor into a 3-tph gyratory crusher.

The ore is then conveyed to a 2- by 4-ft shaker screen, where the plus ½-in. fraction is transferred by conveyor to the gyratory. The minus ½-in. material is transferred by conveyor to a stockpile.

During the first year of operation no ore will be leached, but full-scale open-pit excavation will take place from April 1 to October 1; 9,000 tons of ore will be prepared for leaching beginning the following spring.

During the first summer, the heap leach pad will also be built. The base of the pad will be 1 ft of compacted fine gravel overlaying solid rock that has been graded to drain to the center of the pad. The front of the pad will have a 2-percent grade to provide proper drainage of the leaching solution. Next, 20 mil of impervious liner will be placed on the gravel base followed by a drainage net that will allow the fluid to flow easily along it; a second impervious liner of 40 mil will be placed on top the net. Overlying the 40-mil liner will be another layer of drainage net. Above this, a layer of geotextile fabric will be placed; this will prevent the fines from entering and clogging the drainage net ('sliming' the system) and will maintain an open drainage net, thereby providing adequate drainage of the leachate. The layer of geotextile will also provide additional tensile reinforcement to the entire lining system.

The second layer of geotextile lining will strengthen the system and act as a precautionary backup should one of the layers fail.

The lining will be sealed at the seams with a heat gun. This welding process creates a completely homogeneous bond between the molten weld bead and the surfaces of the sheets.

The dimensions of the pad will be 235 ft^2 at the base, and the sideslopes of the pad will have a ratio of 2:1 to prevent the material from sloughing off the pad. The total height of the pad will be 40 ft, which will take 5 yr at 9,000 tpy to complete. Similar pads will be prepared adjacent to the original pads as they are needed to provide continual leaching of the ore at 9,000 tpy.

The holding ponds will be constructed using a double liner of 20 and 40 mil of the same material as used on the pad, and will contain no drainage net or geotextile. The leachate holding pond will be 30- by 30- by 10 ft deep. Normal operating capacity will be 30,000 gal, with a total holding capacity of 60,000 gal. There will be a backup both at holding pond to hold the overflow caused by precipitation. The overflow capacity will be equal to 1 yr of precipitation over the pad area, or 325,800 gal/acre, which will require an overflow pond 75 ft^2 and 10 ft deep with a holding capacity of 420,000 gal.

The leachate solution will contain a biodegradable chemical called BIO-D, sodium bromide, and sodium hydroxide, which are all approved by the EPA as nonhazardous chemicals.

The leaching cycle begins at the leachate holding ponds, where the strength of the solution will be monitored by a control system that automatically checks the solution every 10 sec and adds the needed component chemicals to maintain a predetermined strength. The solution will be pumped to the top of the pad where it will be distributed by rainbird sprinklers. The rates of application of leachate solution varies from operation to operation, but are generally about 10 $l/hr/m^2$ (0.25 gpm/sq ft) of leach area. Using the rate of ¼ gph/ft^2 will require an application rate of 315 gal/min.

As the leachate is applied on the pad it will percolate through the ore, leaching the precious metals as it travels, and the resulting 'pregnant solution' will flow through the geotextile fabric, which will filter most of the fines and slime from the solution. As the solution passes through the geotextile fabric it will encounter the top impervious liner and drain to the front of the leach pad through the channels of the netting.

From the front of the leach pad the solution will flow through a slime ditch 10 ft long, 5 ft deep, and 4 ft wide that is doubly lined in the manner of the holding ponds. The slime ditch will act as a settling pond for the slime. The leachate solution is then returned through a plastic pipe by gravity feed to the leachate pond, thereby completing the cycle. Total leaching time for the system is 120 days.

The next step involves extracting the precious metals from the pregnant solution by the use of ionic resins. The pregnant solution will be pumped at 30 gpm from the leachate holding pond into a resin cell of 6 ft^3 (1 cu ft of resin weighs 40 lb and holds 150 oz of precious metals). As the pregnant solution flows through the resin, the gold and silver ions are exchanged for ions in the resins. The stripped leachate solution is then returned by gravity feed to the leachate holding pond.

Daily assaying of the leachate solution as it enters and leaves the resins for values of gold and silver will make it possible to compute the precious-metal recovery at the flow of 30 gpm. When the absorption rate for the precious metals begin to decline it will serve as an indicator of resin saturation.

The saturated resins will then be removed from the system while switching the pregnant solution flow to fresh resin. The saturated resins will then be smelted at the site, assayed, and then sent on to the refinery.

Expected percentage of recovery from the heap-leach operation is 60 percent. At 9,000 tons per year the operation will produce 891 oz gold and 324 oz silver per year.

THE DELPRAT LEAD-ZINC MINE
BROKEN HILL, AUSTRALIA

by

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The Broken Hill Mine was discovered in 1883 by Charles Rasp, a sheep range rider who discovered some unusual black minerals and subsequently staked claims and sunk a shaft. The early ores included cerrucite, a lead carbonate, and oxidized gossan zones containing substantial native silver.

During the early period of extensive drilling, one of the major problems of the mine was the lack of strong timber to support the underground workings; native trees (including the gum) and imported New Zealand pine quickly decayed. Today, the Delprat mine hosts a growing city of 21,000 (the third largest city in New South Wales) and uses imported Oregon timber for its head frame, cribbing, and timbers. The ore mined today, an unoxidized sulfide ore that includes galena, magnetite, and sphalerite, averages about 4 oz/silver. It contains about 30 percent metal and 70 percent gangue material. When the ore arrives at the smelter, up to 51 metals are extracted, including a considerable amount of cadmium. Broken Hill, one of the largest producers of cadium in the southern Hemisphere is also one of the largest gold producers in New South Wales. Some of the other metals include beryllium, nickel, and copper.

Today, miners are brought down to the working level in a cage. Upon leaving the cage, the drive follows along a footwall. Each 100-ft level contains an ore chute. Most miners use air-legger machines or jacklegs to drill a 12-ft block of ore. Drill holes are located to ensure maximum safety and ore extraction. After each hole has been loaded they are connected to a series circuit. The explosives used today are electrically detonated by a 'millisecond' unit. When all holes are completed, the shift foreman removes all miners from the level being fired. Once the dust has settled, the next shift is allowed to enter the mine. The miners then shovel the ore into a small 1-ton truck and push the truck to the ore chute. Clydesdale horses once pulled the ore carts to the surface, but have been replaced by 3-ton ore cars powered by banks of lead-acid batteries.

At Broken Hill, underground mining is done on an incentive scale or contract. The more ore a team brings out, the more money they earn. A good party (a four-man team) can earn around $100/shift by removing an estimated 10-ft slice into the ore body, or roughly 1,300 tons.

Once on the surface, the ore is taken to the mill, crushed to a fine powder, and then taken to a flotation room. There, flotation cells (developed at Broken Hill in the early 1900s) separate the ore from the gangue by placing the finely powdered concentrate and reagents into a large wash trough that runs through flocculation cells. The ore floats to the surface and is scooped off and directed to filtering pads for drying. The concentrates are then shipped to the smelter at Port Pirie in South Australia, where the individual metals are separated.

ALLUVIAL SAPPHIRE MINING IN AUSTRALIA

by

Allan W. Coty

Peter Brown's Sapphire Mine is located about 9 mi from the town of Inverell, in New South Wales, Australia.

The operation requires two men to mine and process 80 yd^3/day of alluvial gravels excavated from a bench about 100 ft from the present stream. Mr. Brown digs test pits with a HY-MAC Excavator Backhoe to outline areas with sufficient sapphire content to be mined economically. Then 4 to 9 ft of overburden is removed and stacked to the side of the trench. Access is left open at both ends of the trench. A lensate 1- to 6-ft-thick layer of gravel pay is removed by the backhoe and loaded into a 4-yd dump truck for a 100-yd trip to the processing plant, where it is dumped into a receiving hopper, washed with water, and turned into a slurry.

The gravels are then washed from the receiving hopper into the rotating trommel mouth.

An 18-hp diesel engine drives the pump that supplies both the water for the hydraulic monitor and the hutch water on the jigs. The pump is located at the fourth of four settling ponds, which are in a closed circuit. The overflow method is used to transfer water from one pond to the next. The overflow outlets are switched to opposite sides from pond to pond to keep the water from running straight through, thereby giving the solids a longer settling time. The last settling pond contains only seepage water that passes through the gravel tailings barrier. The pond barriers were built with tailings and oversize from the mining operation.

An 18-hp diesel engine also provides the working power required to turn the trommel. The trommel in turn spins a wheel to which a sprocket is bolted. A chain from the sprocket drives a rod-and-cam system that operates the primary and secondary jigs.

The slurry from the receiving hopper is then tumbled and thoroughly washed in a 10-ft section of blinded trommel before it crosses a 5-ft section of 1-in.-thick steel mesh.

The oversize material, after being washed, falls out the end of the trommel directly into the dump truck, where it is transported back and used to fill in the mined portion of the trench.

The minus 1-in. material that passes through the steel mesh is directed across the primary jig. The minus 1-in. plus 3/16-in. sapphires work through the bedding and concentrate at the interface of the gravels and jig screen. The minus 3/16-in. material then works its way through the bedding and passes through the 3/16-in. jig screen and out the spigot at the bottom of the duplex. From the spigot it falls into a trough that leads to the secondary jig, which has a 1/16-in. screen. Anything that goes through the screen is directed to the first pond, along with the tailings from the primary jig. The minus 1/16-in. sapphires are not kept because they are too small to be faceted economically.

After shutting down to clean up, the bedload of natural stones were removed to expose the plus 3/16-in. sapphire concentrate, which is loaded into a bucket for final sorting. The bucket is dumped into a washing screen, which is vigorously rotated in the water to concentrate the sapphires at the interface of bedding material and screen. The screens are flipped over onto a clean, flat surface. The sapphires are thus exposed on top of the pile, where they are easily visible and can be hand picked.

SELECTED PUBLICATIONS

This section lists selected reports that were discussed or displayed at the Ninth Annual Placer Conference that may be of interest to members of the mining community. Contact the author(s) or the appropriate agency for information regarding the cost and availability of these reports.

Alaska Division of Geological and Geophysical Survey, 1985, Reports and maps issued by the Alaska Division of Geological and Geophysical Surveys: Alaska Division of Geological and Geophysical Surveys Information Circular 11, 83 p.

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