PROCEEDINGS OF THE EIGHTH ANNUAL ALASKA CONFERENCE ON PLACER MINING

"PLACER MINING: YESTERDAY, TODAY, TOMORROW"

APRIL 2-5, 1986

Compiled by

P. Jeffrey Burton and
Henry C. Berg

Front cover: *The Tuluksak Dredging Company's dredge near Nyac, Alaska. Built by Washington Iron Works, the dredge has a 5⅓-ft^3 bucket capacity. Photo courtesy of T.K. Bundtzen, 1987*

1986 PLACER-MINING CONFERENCE COMMITTEE

Rose Rybachek (Livengood-Tolovana mining district)	Conference Coordinator
Gail Ackels (Circle mining district)	Entertainment Chairperson
Mary Albanese (Alaska Division of Geological and Geophysical Surveys)	Poster and Session Chairperson
Lela Bouton (Koyukuk mining district)	Advertising Chairperson
Roger Burggraf (Fairbanks mining district)	Member
Jeffrey Burton (Alaska Miners Association)	Speaker Coordinator and Session Chairperson
Curt Freeman (Placer Miners of Alaska)	Session Chairperson
Kathy Goff (Alaska Division of Geological and Geophysical Surveys)	Assistant Conference Coordinator and Session Chairperson
William Lex (Tanana Valley Community College)	Member
Dr. David Maneval (School of Mineral Engineering, University of Alaska)	Member
Ron Rosander (Alaska Miners Association)	Member
Bruce Stillwell (Placer Miners of Alaska)	Member
Rosaline Stowell (Alaska Women in Mining)	Exhibit Chairperson
Helen Warner (Miners Advocacy Council)	Member
De Weber (Alaska Women in Mining)	Assistant Conference Coordinator

CONFERENCE SPONSORED BY

Alaska Miners Association
Alaska Women in Mining
Circle Mining District
Fairbanks Mining District
Livengood-Tolovana Mining District
Miners Advocacy Council
Placer Miners of Alaska
School of Mineral Engineering
Tanana Valley Community College

FOREWORD

In this volume, you will meet men and women who are truly Alaska's modern pioneers. They are placer prospectors and miners, following an ancient and romantic quest--gold! Some retrace the footsteps of Alaska's first placer miners, who sought paydirt 100 yr ago, but many are blazing new trails in virgin ground and using tools never dreamed of by those early sourdoughs.

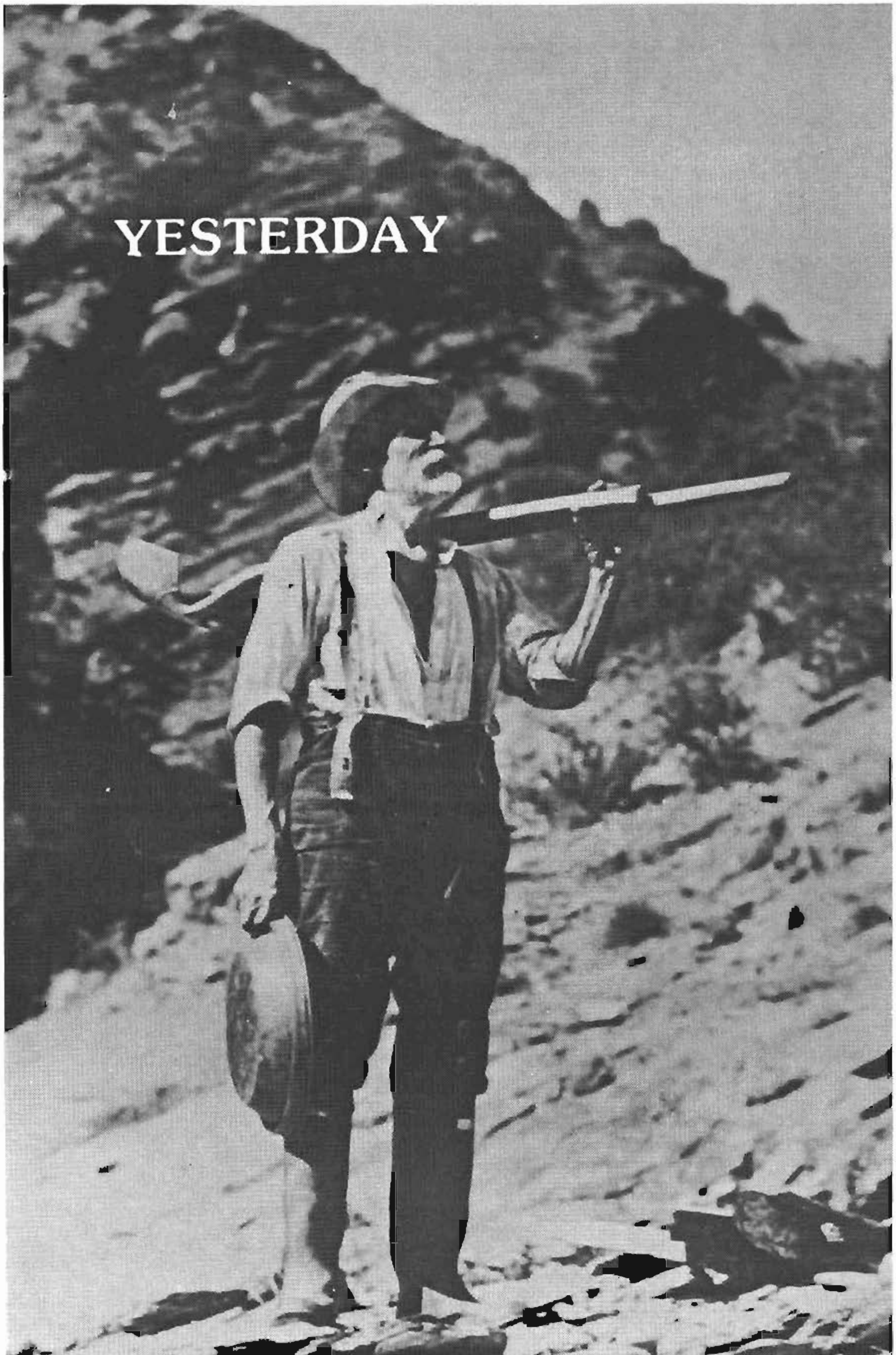
Here, these modern pioneers tell their stories. You will meet the man who invented and developed the mighty automatic hydraulic monitor, capable of moving mountains of material for pennies a yard. You will meet men who are testing machines and methods to coax the last particle of fine gold into their concentrates, while simultaneously striving to meet the world's toughest environmental standards. You will meet an Alaskan who has actually found the golden fleece. But, first, he had to deal with a foot-thick anaconda, foot-long scorpions, rabid vampire bats, and murderously poisonous no-see-ums.

Gold! The lure is strong. The rewards may be great, but the risks even greater. The work is hard and comfortless. Nature is not an easy partner. Yet these Alaskans persevere.

We wish to thank all the speakers who participated in the conference, particularly those who submitted manuscripts for this volume. Appreciation is also extended to members of the conference committee. This report was made possible by the support of the Division of Geological and Geophysical Surveys, Alaska Department of Natural Resources.

P. Jeffrey Burton and Henry C. Berg
Fairbanks, Alaska
January, 1988

CONTENTS


YESTERDAY

A HISTORICAL PERSPECTIVE

By
E.N. Wolff, Professor Emeritus
University of Alaska-Fairbanks
Fairbanks, Alaska 99775

My topic today is 'A Historical Perspective.' This is a rather broad mandate, but I will try to give you my concept of the history of our industry. The history of Alaska from 1867 to 1939 is the history of mining. This is not to downplay the importance of the fishing industry to Alaska, but, truly, the geographic exploration of Alaska and the establishment of towns, cities, and roads have, in large part, been the result of mining.

The gold rush stories are so well known and evoke such romantic images that the average person thinks of them immediately. They have the added attraction of having a nice, neat beginning---'the Klondike Stampede'--and a nice, neat end---the First World War. Certainly for me, they tell the tale of one of the most romantic episodes in our history and leave the observer with a sense of the importance of that era. In the space of scarcely 20 yr, the vast extent of Alaska was explored and settled, just by the efforts of gold seekers. This last great adventure, involving mass movement of great numbers, really opened up the northern part of the continent. It is simply called 'the gold rush.' I hope this perspective will give you a little more insight into how our state came into being.

I would like to give you a few facts and stories about the gold rush and trace the progress of men and events. For those who have heard the story before and who know it better than I do, I apologize.

Robert Marshall, in his book 'Arctic Village,' reported a statement made to him in 1930 by a Koyukuk miner: "And then I love the wild mountains and the unexplored country and the fighting through hardships. There's no people in the world have overcome as many hardships as the Alaska pioneers. Why, the dangers and discomforts which these Arctic and Antarctic explorers have stood is a joke in comparison with what thousands of Alaska men and women have gone through. If Stefansson could get to the North Pole and discover gold, there'd be 500 Alaska whores there inside three months. There's no hardship in the world that a real old time Alaskan couldn't bear, and I'm glad that I was one of them." Although the speaker attributed the hypothetical discovery of gold at the North Pole to an explorer, it would most certainly have been by a prospector. Certainly the presence of 500 Alaska 'sporting women' could only mean that the miners had established another outpost of civilization and a prosperous one, or 'good camp,' at that. For some reason, this criterion of prosperity seems to have taken hold among our prospectors. I remember well an exchange between two of them while we were rebuilding a churn drill in the Chandalar district, 100 mi from the nearest town:

Younger prospector: "There never was a good camp that didn't have its share of sporting girls."
Older prospector: "The Chandalar never had any."
Younger prospector: "I said 'good camp.'"
Older prospector: "By God, that's right."

With that we had better move on.

I would like to direct your attention to a remarkable document. Prior to World War II, the Alaska School of Mines required that each student write a senior thesis. Some students elected to write on historical or broad industrial topics. James C. Hildebrandt, who attended the University shortly before World War II, wrote 'History of Placer Mining in Alaska,' then disappeared. He never returned to Alaska. The library record shows that in the last few years 52 people checked out his thesis. Wherever you are, Jim, you should know that your work has finally received the recognition it deserves.

There does not seem to be any history of native Alaskans using placer gold to make artifacts; therefore, for the first documentations of mining in Alaska, we must go to the Russians.

I am almost ashamed to begin with Peter Doroshin. Poor old Peter has had his story of failure told so often that he must be tired of it. Very briefly it is this:

> The Russian American Company was interested in anything that would turn a dollar, and if anything happened as a result that advanced the cause of Russian dominion, that was all to the good---the higher-ups, 5,000 mi away, might hear of it. Because the fur trade was prospering (albeit with worse effects on the natives than on the fur seals), Peter Doroshin was sent to see what he could come up with in the way of minerals. We know today that he was on an impossible mission. He might have found a little coal, but he was after gold. In two years (1848 and 1849) of driving his natives, he found only a few ounces of gold. This was on the Kenai Peninsula.

Now we must jump ahead almost to the purchase of Alaska from Russia. The Russian American Company had a monopoly on Alaska commerce. After Alaska was transferred to the U.S. Government, however, some of the commercial functions of the Russian American Company were taken over by the Alaska Commercial Company, later to become the Northern Commercial Company.

In 1862, 'Buck' Choquette reported gold from the bars of the Stikine River in southeast Alaska. The Russians, cognizant of other gold rushes, tried to keep it and other discoveries quiet, but by 1871 there were many prospectors in the country. A few hard-rock deposits were found in southeast Alaska, including Windham Bay and Sumdum Bay, but placers were few because the area had been glaciated.

In 1879 John Muir, the naturalist, went through southeast Alaska by canoe. He reported favorable geology, and, in 1880, Richard Harris and Joseph Juneau set out to explore the area. At the present site of Juneau (Harrisburg mining district), they found abundant placer and lode gold. In short time, several mines were in operation, both on the mainland and on Douglas Island (Treadwell Mine) across the channel. Meanwhile, in the 1870s, at least two parties had descended the MacKenzie River and crossed to the Yukon basin. As word of these explorations spread, more men wanted to cross Chilkoot Trail or White Pass to see for themselves, and, during the 1880s, more and more men did. Most of them got their grubstakes working in the Juneau-Treadwell mines.

Before we leave Juneau-Treadwell, we should note a very important fact: Juneau was struck in 1880. This was a time of absolutely no civil law in Alaska, nor criminal law for that matter. What had happened? The mining law of 1872 allowed miners of various districts to organize and make certain rules to govern mining. In the absence of civil law, the miners made their own rules and conducted the broader functions of government. In general, they acquitted themselves well and jurisdiction was only gradually transferred to the civil authorities.

As noted, during the 1880s more and more men crossed to the Yukon basin. At first they worked the bars of tributaries of the Yukon River. Then they worked farther and farther afield. A fascinating account in 'Sourdough Sagas' tells of the men reaching the Kuskokwim and of working the bars of the Koyukuk, both a long way from the upper Yukon. In 1886, they found creek placers in the Fortymile; in 1893, they found placers at Rampart and on Birch Creek (lately discovered by our new pioneers as a wild and scenic river).

The next and crucial event has been described by so many that there is no need for me to do so here. In August of 1896, coarse gold was discovered on a tributary to the Klondike River, which is a tributary to the Yukon. The news travelled up and down the river, but such news had been heard before. This one, however, proved to be the big one that everyone had been waiting for. When in the following year the gold from the winter's mining reached the States, it precipitated a world-wide gold rush. People that arrived two years after the discovery had little chance to profit from the Klondike discoveries. However, since most of the stampeders were Americans anyway, they set out for Alaska. Fortunately, gold had just been discovered at Nome, which gave them at least one destination to head for, but it wouldn't have mattered anyway. The stampeders of 1898 didn't just head for the Klondike but for every conceivable locality rumor suggested. (By the way, the Nome story is the only black mark on the long history of Alaska mining and miners. Rex Beach and Judge Wickersham have told the story of how a political boss, Alexander McKenzie, and a dishonest judge, Arthur H. Noyes, conspired to cheat honest claim owners out of their property. Beach called them 'The Looters.')

There are nine separate recognized mining districts on the Seward Peninsula alone. All of them were discovered by 1902. Elsewhere, districts of more or less significance were struck all over Alaska, the greatest was the Tanana (Fairbanks) district. Through the years from 1898 more than 50 mining districts were struck. Some of the best known are the Koyukuk, Iditarod, Innoko and Kantishna districts. Suffice it to say that by 1914, all the districts had been found or at least noted. The last to be discovered was the Tolovana or Livengood mining district, struck in 1914.

I have not said much about the important lode deposits. As noted previously, the gold mines at Juneau and Treadwell were of extreme importance. In 1899, copper deposits were found at Kennicott. By 1911, a railroad had been built from Cordova to the mines, and Kennecott Copper Corporation became a household word. The mines closed in 1938.

I would like to bring home to you the flavor of this era by describing some of my contacts with people from those times.

Two thousand men in the spring of 1898 took ship to Kotzebue Sound. I was told that when one schooner dropped anchor, several men jumped into row boats and began heading for the Kobuk River mouth. By winter, several hundred were spread along the Kobuk for more than 100 mi. These men found nothing. In the spring of 1899, they either took ship outside or drifted down to Nome or other mining districts on the Seward Peninsula. Six of them formed together and went up the Kobuk. The next winter (1899-1900), the six were camped at Reed River Hot Springs between the Kobuk and Noatak Rivers. I later came to know one of these men and got their story.

They pitched a tent right over a hot spring and had hot baths all winter. A young native boy came their way almost starving. They fed him all winter and with the spring told him "get out and rustle." That summer they reached the Noatak and walked to near the head. They raised prospects on a creek they named 'Lucky Six,' after the six-man group that made up the party: A.W. Amero, Tom Pierson, Louis Lloyd, Manuel Mello, and Jimmy Ackema (I didn't learn the name of the sixth). They hiked back to timberline, then whipsawed lumber and built sluice boxes, which they carried back 30 mi.

Of course, Lucky Six Creek was a 'glorious failure' as the prospectors say. That spring they crossed the divide to the head of the Alatna, then rafted down to the Koyukuk. By spring they were in the Koyukuk district, mining and prospecting. By 1906 at least three of them were in the Chandalar district, 600 mi from where they started. Louis Lloyd discovered the Kobuk copper near Shungnak. Manuel Mello raised a family in the Chandalar district. Jimmy Ackema drowned in a lake between the Middle and East Forks of the Chandalar River in 1912. The Board of Geographic Names calls the lake he drowned in 'Ackerman Lake,' but it was named for Jimmy. Tom Pierson stayed for awhile in the Chandalar district, then ran a saw shop in Fairbanks. A.W. Amero stayed in the Chandalar. In his 90s he made several trips on foot, packing dogs, to the East Fork of the Chandalar 90 mi away. He died in his 90s at the Sitka Pioneers Home.

In 1891, a Japanese immigrant named Frank Yosuda left the Revenue Service ship 'Bear' in Barrow. There he married a chief's daughter. He also met Thomas G. Carter and formed a partnership. Carter was to prospect both sides of the Arctic Mountains, now called the Brooks Range. Yosuda furnished the grubstake and with his Eskimo followers brought the grub and supplies to Carter, as he said, "right to his door." In 1905, they found coarse gold on Little Squaw Creek in the Chandalar district (also an interesting story).

They went to the Yukon and established the town of Beaver, which became the only Eskimo community on the Yukon. They established a trading post there and were instrumental in establishing a trail to the Chandalar. I met Frank in the Chandalar district when he was 80. He had returned to check out a prospect that his partner had told him about.

Shortly after the discovery of the Chandalar, Samuel Marsh crossed the Arctic Mountains alone, in winter, from the Arctic Coast via the East Fork of the Chandalar to the Yukon. He was a mining engineer and in Ft. Yukon met Alfred H. Brooks. Yasuda said Marsh looked like a wild man, for he was dressed in skins and filthy. Brooks took him to Washington, where he furnished the first geologic information on the area (published in a U.S. Geological Survey Professional Paper). Marsh established a commissioner's office in the Chandalar. Wherever he went he established law and order for the protection of the public. I wish that I had time to read you a coroner's inquest report composed in the wilderness at 50 below, 115 mi from the commissioner's office and 75 mi from the nearest town.

Most of you here know what has happened since those early days. The Hammond Company at Nome and the U.S.S.R. and M. Company in Fairbanks saved the day after the rich deposits that could be worked by hand were exhausted. From 1924 to 1939, the Fairbanks dredges supported a thousand people in Fairbanks. You also know that gold mining died during the 1960s because of the fixed price of gold. Most of all, you know that since about 1974, the increased price of gold has brought work to the working people and businesses of Alaska. During the 1970s, many mining and exploration companies established offices and found billions of dollars worth of minerals to be developed in the future. Today more than 80 of these companies have left us. What's going on?

I think you will agree that I have been talking about heroic deeds and people. They established towns and roads, fed the population, and brought law and order to the wilderness. Let me relate one more story to you.

Sammy Pingelo, an Eskimo born in the Arctic mountains, was used to the rough good humor of the white miner as well as to the ways of his own people. In the 1920s, Sammy was driving a dog team up Little Squaw Creek and came to the Mello cabin (by this time a log bunkhouse). He stopped, then decided to try to reach Big Creek that night and pressed on. When Manuel heard of this, he lectured Sammy severely: "Don't you ever pass my house again without stopping to eat and rest." The code of the country had been affronted.

Now I would like to read an item from the Fairbanks Daily News-Miner of Tuesday, April 1, 1986[1]:

BLM Moves To Oust Squatter

> The Bureau of Land Management has cited a man for living in a mining-claim cabin on Nome Creek without submitting a plan for mining.

---

[1]Names in quotation marks were changed by the author.

BLM employees found Jim Tungate living in the cabin last summer. In response to questions, Tungate said he had no intention of mining, submitting a plan of operation or moving, according to Sharon Wilson, the BLM's public affairs representative. Tungate's cabin is within the White Mountains Recreation area.

'Jesse James,' the BLM's outdoor recreation planner, told Tungate that he would probably be evicted. By law people cannot live on mining claims unless they complete a certain amount of mining work annually.

About a month ago, the BLM's special enforcement agent, 'Al Capone,' and 'James' travelled out to Tungate's cabin to issue a citation. Tungate was not at home but 'James' and 'Capone' met him as he was returning by dog team on U.S. Creek Road, Wilson said. They issued the citation. Tungate reportedly has not moved out yet.

In the last 15 yr, the Congress, driven by people with God-knows-what motives or God-knows-with-whose money, has passed three laws handing over 72 percent of Alaska to special-vested interests. Twenty-seven percent has been given to a state controlled by the same and other special interests. We, the people, have 0.3 percent of our land.

Never in their wildest and fondest dreams did the looters of Nome visualize theft on this scale. God help us all.

52 YEARS PLACER MINING IN ALASKA

By
Bill Carlo, Alaska Placer Miner
2113 Southern Avenue
Fairbanks, Alaska 99701

I was born in Rampart on the Yukon River in 1918. At that time, the mining industry in Alaska was booming. Every creek that showed any color at all was being worked. A lot of the people who tried to make their fortune on the creeks eventually went broke from mining poor to marginal ground, but a lot of miners made good. Some of the families that started back then in the placer mining business are still mining today. It's a family tradition for as much as three generations.

I started out in the mining business in the fall of 1934, when I was 16 yr old. Mining was one of the few ways for us village people to really make some money. I went to work for a fellow by the name of Baldo Forno who had a big placer operation down at Poorman.

The town of Poorman was booming in those days. There were three big mining operations there at the time. There was Baldo Forno's operation, Denny Coil's, and another one owned by a man named Jensen. There were also operations on Flat Creek, Spruce Creek, and at Placerville. Poorman was the center for supplies. There must have been 200 people or more living in Poorman then. There was a Moose Hall for social events, a store, a dog barn for dog teams, a horse barn, warehouses, and barracks for the mine workers.

My first job at Forno's mine was working underground in the drift. We didn't strip the ground down to pay dirt. What we did was tunnel into the pay streak. We worked this ground only in the winter, when it was frozen solid. Since it was frozen, we had to steam-thaw as we went by driving in steam points. There were twelve of us 'in the hole,' as we called it, with six wheelbarrows, two men to each. The ore bucket came by once every 6 min and held three wheelbarrow loads, so two six-man teams alternated filling the bucket every 6 min. I made six bucks a day at $0.75/ hr. That was pretty good money then.

As I said, we had to thaw the ground with steam points as we worked the drift. The steam was generated by a big boiler. In the winter of 1936, we hauled in 1,100 cords of wood to feed that boiler.

At Poorman, we worked all winter and stockpiled the dirt in a dump. The pile would have grown to a regular mountain by spring. Since we had to steam-thaw, the dirt was wet when it hit the bucket. When it was dumped, it would freeze solid. The pile would build up to a sharp peak, and every so often we'd have to steam-thaw the point so that the bucket could pass.

We had to start working up that pile of dirt as soon as spring thaw started. There were five sluice boxes under the pile down in the ground. The water that we used was mostly runoff from the melting snow. We had to have the whole dump sluiced by June 1, because the creek would dry up by then. Because there was so little water to work with, we had some bad trouble between the different operations over water rights.

I worked for Baldo Forno for three winters. Then, in the spring of 1938, I went to work for Billy Vick at Flat Creek, 8 mi out of Poorman. Billy did his mining by stripping the overburden down to pay and sluicing all summer. He had built a very fancy trestlework flume to bring water from a ditch 1,000 ft away. John Miscovitch bought Billy Vick out in 1940. Hagland is mining that ground now.

In 1939, I went to work for three partners by the names of Eric Hart, Gus Utilla, and a guy by the name of Hansen, I think. They had an operation at Bear Creek, in the Cripple Mountains, about halfway between Ruby and McGrath. We used hydraulic giants to strip overburden. I pushed dirt with a D-6 cat for part of that summer (fig. 1). Then I went to work for an old fellow and his three sons as a cat-skinner and oiler for the old dragline operation on Cripple Creek in the Cripple Mountains. Then, after 6 yr of working in other people's mines, I decided to try my own venture.

In the winter of 1939-40, my nephew, Greg Parker, and I worked our own prospect out on Granite Creek, about 150 mi south of the Yukon River, toward Lake Minchumina, at the head of the Novy River. We drove a drift tunnel 30 ft into the side of a hill. The pay streak sloped uphill from the creek bottom and was 4 ft thick. Greg ran the boiler and I worked 'in the hole' all that winter. Unfortunately, it wasn't very good ground, so we just about broke even. It was what we used to call 'fifty-cent ground.' We recovered $0.50 worth of gold for every foot of dirt at the old price of gold.

In the summer of 1940, I worked as a cat-skinner at Greenstone, 40 mi out of Ruby. I ran an old Alice-Chalmers HD-10 for two partners named Iver Johnson and Ash Richardson. I worked at various mines and prospects up until World War II started up. Then I put my heavy equipment experience to good use for the U.S. Government. I worked through the war years on the construction of the defense installation at Galena.

After the war, in 1948, I started my first real, successful mining venture. This was at Ophir Creek, just out of Ruby. I mined there until 1955 and made good until the last two years. Nineteen fifty-four and 1955 were very dry years. There just wasn't enough water to sluice, and I went broke. I gave up on mining and worked construction until 1958. During that time, I saved up enough to get started back into mining again.

In 1958, I got hold of my mining claims on Hunter Creek, just out of Rampart, where I was born 50 yr before (fig. 2). That was 38 yr ago. The whole Carlo family has been involved in placer mining all these years. All eight kids have grown up in the business, and my wife, Poldine, has been right in there with me all the way. There are three generations of the Carlo family working out there. My 14-yr-old grandson, Andrew, runs the loader.

I have a very modern operation now on Hunter Creek. It's a far cry from the conditions back in the 30s. I designed my sluice box myself, and Hector

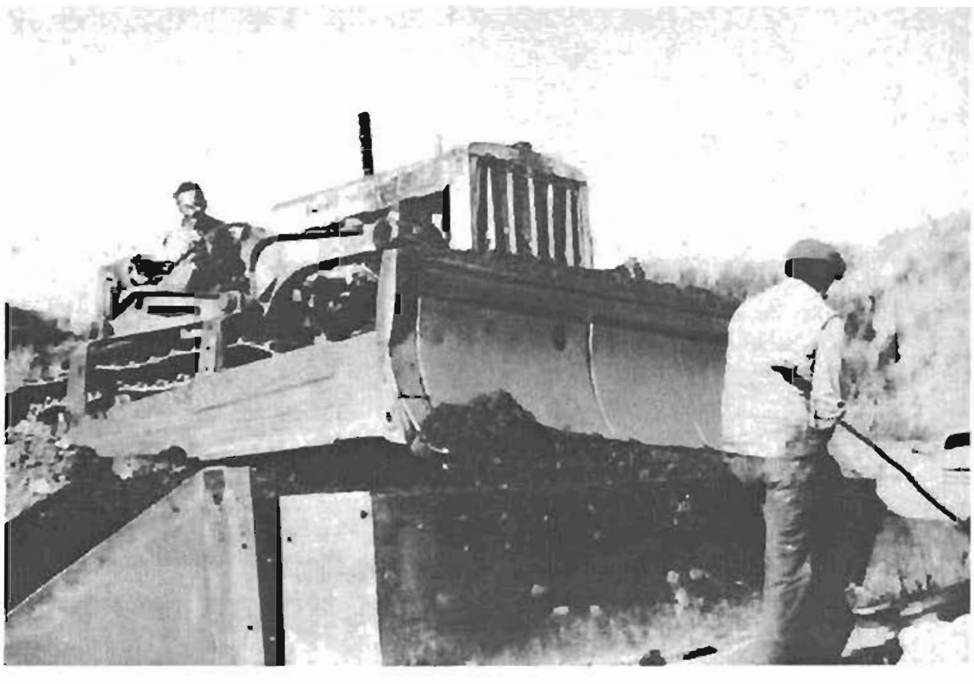


Figure 1. Bill Carlo mining at Bear Creek, Cripple Mountains, southwestern Alaska.

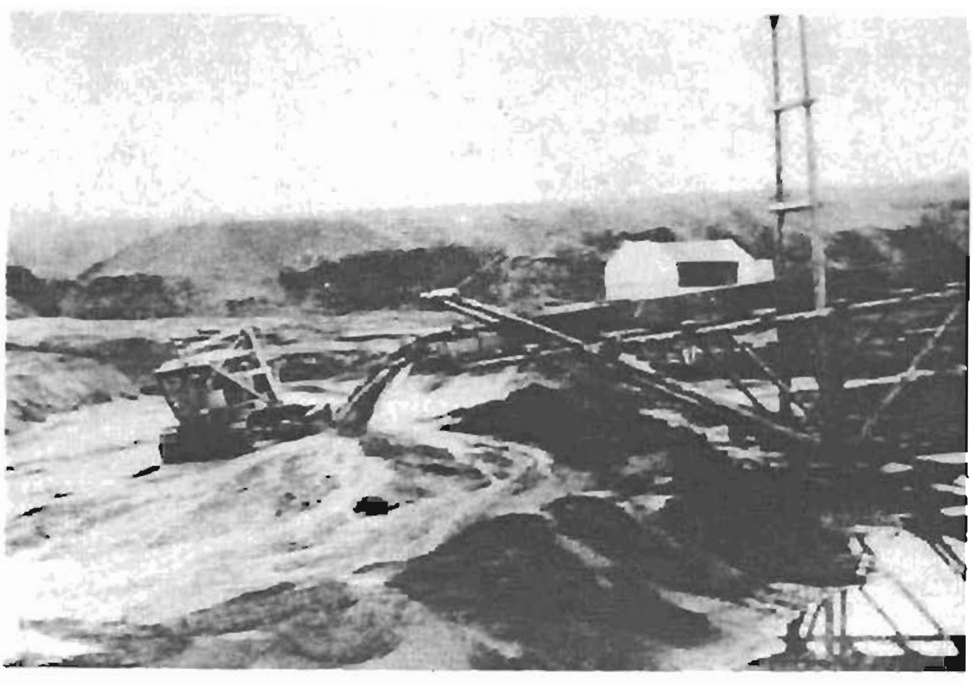


Figure 2. Bill Carlo's placer-mining operation, Hunter Creek, eastern interior Alaska.

of Hector's Welding built it for me. It works so well that Hector now builds them for sale to miners. It is now called 'Hector's Box.'

In contrast to all of my modern equipment out there, the cabin that we still use as the cookhouse is very old. It has to be at least 75 yr old or more. Our cookhouse is the cabin that was used in the movie, 'Spirit of the Wind,' about George Attla's life and career as a champion dog musher.

Mining has been good to the Carlos (fig. 3). My wife and I have traveled all over the world. I've put four of my kids through college and now have a nephew with a degree in petroleum geology and a grandson who is a student at the University of Alaska, Fairbanks. We are as Alaskan as you can get, since our people have lived on this land for at least 10,000 yr---maybe longer---and we're proud of that and of our tradition of mining in this country. Fifty-two years in mining is a long time, but I'll probably keep on mining until the end of my days. Since mining is part of our life---in our blood, you could say---I expect there will be Carlos mining long after I'm gone.

Figure 3. Mining has been good to the Carlo family.

DIAMONDS IN THE ICE

By
E.I. Erlich, Consulting Geologist
595 South Forest, Apartment 311
Denver, Colorado 80222
and
G.A. Slonimsky, Consulting Geologist
595 South Forest, Apartment 311
Denver, Colorado 80222

INTRODUCTION

Nineteen eighty-four saw the 30th anniversary of a momentous event in Soviet geology - the discovery of kimberlite in Siberia. About 100 kimberlite pipes were found, although less than 10 are considered mineable. Today the Union of Soviet Socialist Republics (U.S.S.R.) is the second largest producer of diamonds - both gems and industrial diamonds - in the world. Three diamond mines have been opened at Mirniy, Aikhal, and Udachnaya in the harsh environment of the Siberian Platform (fig. 1). Another mining community was built at Amakinsky, near a large diamond-placer deposit located 150 km from the Arctic Ocean.

There are many legends associated with this story: legends about those who predicted the discovery and those who achieved it in the field. The story told here unfolded during the field campaigns of the 1940s through the 1960s as witnessed by the authors and their companions of an earlier generation. This story illustrates science, chance, and human emotions at work, which resulted in a blend of conviction, courage, bureaucratic sniping, and a competition for glory. At the same time, it is an illustration of the working of the Soviet exploration system. In a broader sense, it is the realization of the Russian image of boundless natural wealth, indomitable ingenuity, and the rigid political-economic system that permeates the Russian landscape as well as the people.

Most accounts of the Siberian diamond saga begin with statements such as 'This discovery confirmed the predictions of...' or 'It was the result of predictions of one of the greatest Soviet geologists, academician V.S. Sobolev...' These are only part of the truth. It had become apparent, even in the 1930s, that the Siberian Platform contained one of the earth's greatest basaltic provinces, described in a monograph by Sobolev (1937). And at the end of the 1930s, G.G. Moor of the Institute of Arctic Geology found alnoites, which are similar in many ways to kimberlites. At the same time, the famous polar geologist, N.N. Urvantsev, found other kimberlite-like rocks in the southern Taymir Peninsula, adjacent to the Siberian Platform. Works of Moor were consulted by Sobolev, and he examined specimens collected by Moor and Urvantsev. Sobolev (1937) briefly described these alkaline-effusive rocks and concluded that they were not related to the doleritic basalts.

In 1938, on A.P. Burov's initiative, the National Geological Institute in Leningrad (VSEGEI) sponsored a comparative study of the geological fea-

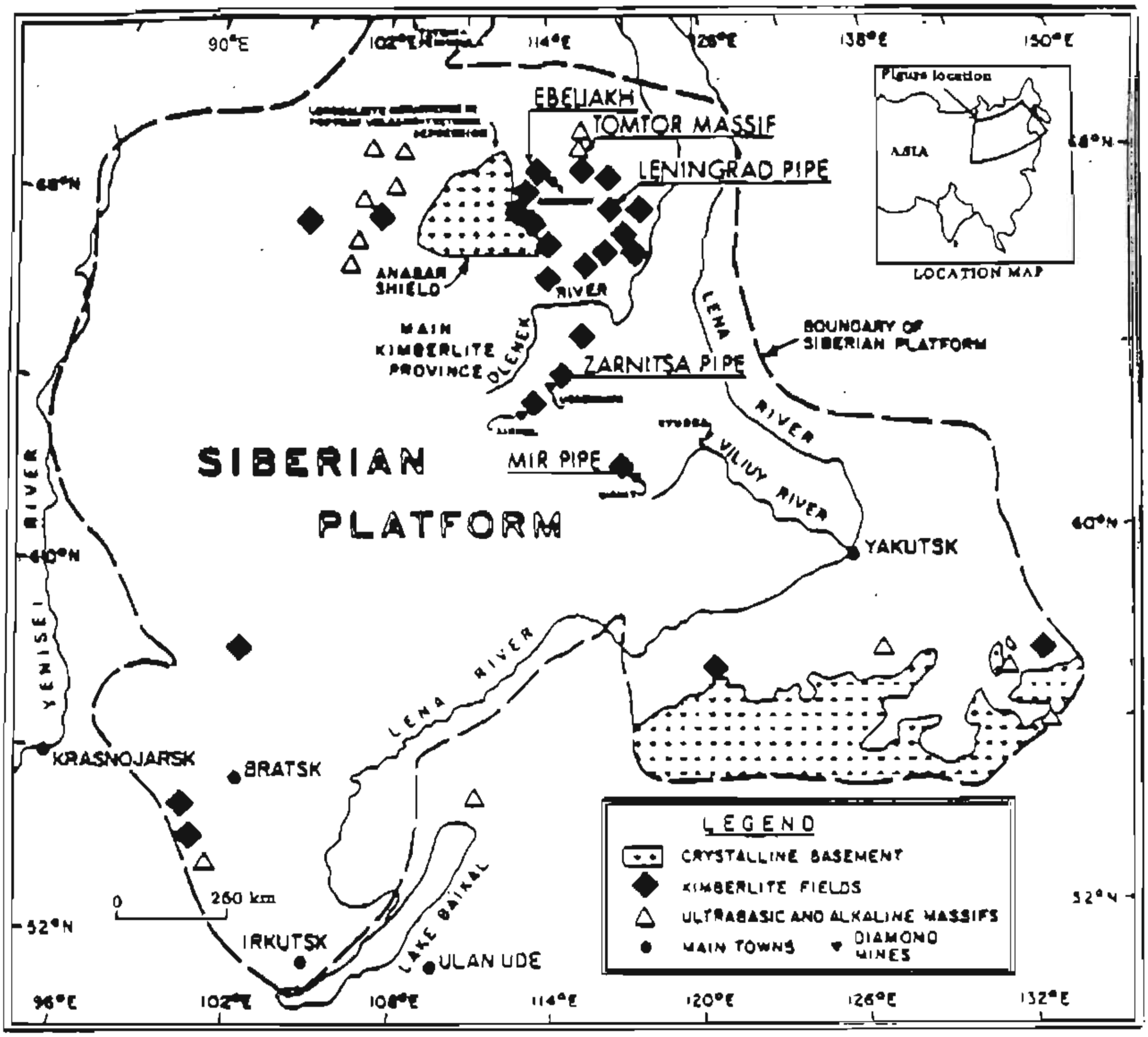


Figure 1. Siberian Platform and main kimberlite fields. Modified after Erlich (1985).

tures of foreign diamond fields with selected regions of the U.S.S.R. V.S. Sobolev participated in the study and, in 1941, presented a special report on the subject to the State's Planning Committee.

But it was Moor who used 'kimberlite' to describe rocks from southern Taymir and northern Siberia (Moor, 1940 and 1941), and he was the first to suggest the possibility of diamond occurrences in those regions. But Moor was considered by his superiors to be no more than a good descriptive petrographer, and Urvantsev spent little time in the field between terms in concentration camps.

Another early paper should also be mentioned: an unpublished paper found by M.I. Rabkin in the geological archives of the Norilsk Metallurgical Plant. It was written by Yu M. Scheinmann when he worked in the concentration camp at the plant in the late 1940s. This report placed the probable location of the yet-to-be-discovered kimberlites within less than 100 km of their correct location.

THE SEED TAKES ROOT

The real efforts to find diamonds began shortly after World War II, when the Ministry of Geology organized a special branch of exploration for optical materials and diamonds. This special branch, like others created from time to time during Stalin's era, received the highest priority for funds, equipment, and personnel (including substantial personnel from forced labor camps). Soviet practice has been to organize specially named expeditions, or working groups, to carry out specific geological missions over periods of several years. Each expedition has a relatively narrow goal and fixed time frame and usually operates over several summer field seasons and winter office campaigns. Several hundred people work for each expedition during the field seasons but only 100 to 200 people work in the winters. The special diamond branch established a summer base camp at Nyurba on the Viliuy River in central Siberia and operated as the Amakinskaya Expedition. It contracted specific field work from the Research Institute of Arctic Geology and the National Geological Institute to agencies of the Ministry of Geology.

The Institute of Arctic Geology at this time was undertaking geologic mapping at 1:1,000,000 scale east of the Anabar Shield in a Cambrian limestone terrane. One of the field parties was led by K.S. Zaburdin, a geologist of low academic stature, but great field experience. In 1951, Zaburdin found an outcrop of dark green tuff, totally unexpected in this terrane. At the end of the field season, he submitted samples for thin-section description to the Arctic Geology Institute at Leningrad. The routine thin-section work was carried out by hard-drinking, but talented petrographer, Vladmir Cherepanov. He was the first to apply the name 'kimberlite' to a specific rock description from the U.S.S.R. The term piqued Zaburdin's interest, and he reviewed the rocks with the head of the laboratory, Professor M.G. Ravich, who immediately realized not only the implications of finding kimberlite, but also the responsibility of those who reported it. He exclaimed, "Do you know what the term kimberlite means? It means diamonds! I think we should be very careful. Call these rocks eruptive tuffs related to basic volcanism. I think Cherepanov was all wet when he wrote his description." Consequently, sheets N-56 and N-57 of the National Geologic Map, published in 1954 at 1:1,000,000 scale, show to the east of the Anabar Shield a spot of dark green in a sea of Cambrian sedimentary units shown in pink. This spot, necessarily exaggerated on the map, depicts the now famous 'Leningrad' kimberlite pipe, one of two majorly exposed pipes in the Siberian Platform.

TRACKING DOWN A CLUE

The next act of the drama took place in 1952, when the Birectinskaya Expedition, at that time called Aganyliyskaya, started an exploration program for kimberlites in the Olenek-Viliuy drainage with geologic mapping at 1:200,000 scale. The Olenek-Viliuy drainage, at that time another nondescript tract in central Siberia, is now known to contain numerous kimberlites. One question to be investigated was the source of heavy-mineral sands in rivers that drain a mainly limestone terrane. The problem was attacked by mineralogists attached to the Birectinskaya Expedition, as well as some from the All-Union Geological Institute in Leningrad. One of the expedition mineralogists was Yagna Stakhevich, a bright and enthusiastic woman from whom ideas poured in profusion for all who would listen. While most of her flashes of inspiration did not light fires, one of her more perceptive ideas was the use of pyrope as a tracer for kimberlites.

The association of pyrope with diamonds had been known for a long time. Pyrope was already used by geologists in South Africa as a mineral indicator for kimberlites. The value of Stakhevich's contribution was to boldly recommend the use of this association for exploration purposes in a region where it had never been applied. This idea was taken up by Natalia Sarshadskikh, a mineralogist with the All-Union Geological Institute. Her husband, Professor Alexander Kukharenko, from Leningrad University, emphasized to the geological staff the importance of this subject. Consequently, the expedition leaders grudgingly began to pay attention to Yagna Stakhevich's work. She found abundant grains of pyrope in river sands and constantly insisted that these were derived from kimberlites. The chief of the research party attached to the expedition, Professor Rabkin, took her aside and counseled her to use caution: "Describe these grains as minerals of the almandine-pyrope series, which could be from the crystalline Anabar Shield." This clever suggestion put the party chiefs at ease.

In the autumn of 1953, a boat with two women mineralogists from the All-Union Geological Institute visited the expedition camp on the Olenek River. Stakhevich enthusiastically related the abundance of pyrope in the heavy-mineral suite and provided a map of the distribution of pyrope. Over the winter, the pyrope nature of the grains was confirmed in the laboratory.

During the 1954 field season, Natalia Sarshadskikh remained in Leningrad on maternity leave. A graduate-student mineralogist, Larisa Popugayeva, came to the Olenek area, armed with Stakhevich's map. From then on, the discovery of kimberlite was just a technical problem. Occurrences of pyrope in heavy-mineral samples were found in the creek bed and traced up to a point where the mineral was no longer present in black sands. There, black sands from soils on the banks of the creek were sampled.

Rumors about the possible finding of kimberlite traveled ahead of Popugayeva's team. Even the chiefs of the Amakinskaya Expedition helped her to obtain food and transported it to her work area. They reminded her of this when the time came to divide the glory of the discovery.

The sampling work continued under the usual Siberian difficulties: long and agonizing expeditions in temperature up to 35°C with clouds of mosquitos. At the end of the 1954 field season, the moment arrived for which hundreds of geologists had worked for more than a decade. In a prospect pit, at a depth of about 2 m, the first pieces of kimberlite were found under alluvium intruded by permafrost ice.

PETTY POLITICS COME TO THE FOREFRONT

The discovery of kimberlite to this point was the result of straightforward scientific technology. Immediately, however, the darker side of human nature surfaced in the form of petty jealousy and rivalry between organizations.

At the end of the 1954 field season, Popugayeva wrote up her report on the season's work. Upon reviewing the report, the chiefs of the Amakinskaya Expedition called in Popugayeva and gave her a choice: either sign an application to join the Expedition, which had been predated to the beginning of the 1954 field season, or never work in the Olenek area again. The student accepted the first choice, as she hoped to use the kimberlite occurrence in a thesis project. This woman, who had survived the rigors of field artillery service during World War II and had experienced several harsh field seasons in Siberia, was not able to overcome the wiles of desk-bound empire builders, who now had garnered for their organization the credit for discovery.

To the victor go the spoils. In this case, the victorious staff of bureaucrats of the Amakinskaya Expedition received the bonuses and the fame for finding the U.S.S.R.'s first kimberlite. Popugayeva was relegated into the background and ignored by the Expedition. Furthermore, she was fired from the All-Union Geological Institute, whose directors were robbed at the last minute of the discovery credit. Her only publication about the discovery was a brief paper authored with Sarshadskikh (1955). The names of others involved in the chain of events, including Stakhevich and Kukharenko, were never mentioned.

There is a belated happy ending to the story. A reporter heard about Popugayeva's role in the discovery, and she was awarded the Order of Lenin for her accomplishment.

THE RUSH BEGINS

Once the first kimberlite pipe, Zarnistsa (Summer Lightning), was discovered, diamond exploration began in earnest, guided by a clear idea about the source of diamonds in the region and by a simple method of exploration---the tracing of pyropes in black sands. Chiefs of the Amakinskaya Expedition previously had exhorted geologists (on the forthcoming anniversary of the Revolution) to redouble their efforts to discover and sample differentiated dolerite intrusions, which were considered as the potential source of diamonds. Now they directed their teams to seek only new kimberlite discoveries.

Members of the Birectinskaya Expedition remembered the ultramafic breccia found by Zaburdin in 1951 and reanalyzed heavy-mineral samples previously collected in the region. In 1955, the famous 'tuffs' were described in great detail as the 'Leningrad' kimberlite pipe. In subsequent years, the Birectinskaya Expedition found several scores of kimberlites.

The Ministry of Geology decided to map the entire area east of the Anabar Shield at 1:200,000 scale. This work was undertaken by two organizations, the Research Institute of Arctic Geology and the All-Union Trust of Aerial Geology, both based in Moscow. The 119th East Meridian would separate their respective areas. This boundary, however, bisected some sheets of the National Geologic Map. Consequently, geologists assigned to compile the mapping for these sheets had to commute frequently between Leningrad and Moscow to liaise with field personnel. The dividing line was named the 'meridian of Lobanov' after the official responsible for choosing it.

Field work was carried out in two stages. The first was a reconnaissance effort. Follow-up was carried out by mapping at 1:200,000 scale. Any kimberlites found were assigned to the Amakinskaya Expedition for assessment of their diamond potential. Each mapping party included a team of geophysicists to conduct magnetic surveys of each kimberlite and a mining team to carry out trenching and sampling. In this manner, exploration systematically proceeded northward.

During the period from 1957 to 1961, a veritable flood of data was generated about the petrology and diamond content of Siberian kimberlites. Eventually it was possible to analyze the spatial and temporal distribution of kimberlites in an attempt to predict the location of new diamond-bearing pipes.

DEMORALIZATION SETS IN

The reluctance of the Birectinskaya Expedition chiefs to accept the new findings led to a deterioration of morale and initiative among the expedition geologists. New observations were forced into preexisting pigeonholes for fear of upsetting the bureaucracy, instead of being used as building blocks for new models. Efficiency was an early victim of such repression. A team led by Vishnevsky found it very difficult to take stream-sediment samples during the high spring runoff. Rather than surmount such difficulties, the team members decided to take a fishing trip to a nearby major river while waiting out the high-water period. By doing so, they missed the largest diamond-placer deposit in the northeastern Siberian Platform - the Ebeliakh deposit.

When after several years of exploration, geologists of the Amakinskaya Expedition discovered the diamond-rich Aikhal kimberlite, the first thing they found at the discovery site was a set of playing cards left by workers of the Birectinskaya Expedition. Similar events affected scientific theory. Vladimir Milasev was the author of a series of interesting papers and books about the petrology and geology of kimberlites. Using the data available at the time he constructed a general model, which pointed to a decrease in diamond potential northward. When diamond-rich kimberlites were found in the northernmost region, he was the first to insist that exploration be stopped because the discovery contradicted his theory.

It seems, then, that the discovery of these rich, new deposits ultimately depended upon the application of scientific methods, rather than the subjective interpretations of men. East of the Anabar Shield, a series of massifs composed of alkaline rocks were found. Previously, massifs of this type had only been found in areas west of the shield. Radiometric dates of the previously known massifs averaged about 250 m.y. The age of the new massifs was a surprise with dates about 420 m.y. Reaction from the leading specialist in alkaline rocks for the Institute, Dr. L. Yegorov, was typical: "It is impossible! It is a mistake!" But inevitably, the application of pure science was converted to practice, and it has been proved that massifs located to the east of the shield are not the same as those previously known on its west side. With these new massifs are associated several great economic mineral deposits that are absolutely unusual for this region. The study of these massifs has been assigned to the same Yegerov - a good descriptor-geologist but an absolutely wretched scientist, whose scientific recommendations have been the same for decades: "We have to drill more and deeper!"

OLD PEOPLE IN THE NEW TIMES

Yagna Stakhevich was fired from the Institute of Arctic Geology. She grew old and died forgotten by everybody. Up to the last days of her life, she continued to fuss over a new geological idea: how to assess the new gold-bearing region of the Taymir Peninsula using the quantity of gold grains in heavy minerals sampled from the streams.

Vladimir Cherepanov was fired from the Institute of Arctic Geology and subsequently died from a heart attack. After being dismissed from the Institute, he was among those who created one of the best analytical complexes in the U.S.S.R. for the analysis of rocks and minerals at the National Geological Institute in Leningrad. He proposed a new method of evaluating the depth of formation of magmatic rocks using boron isotopes. The identification of a new boron-bearing province in the northwest Siberian Platform is also among his contributions.

Konstantin Zaburdin is also dead. In his last years, he became an administrative worker. For him, whose life as a geologist was based on his intimate knowledge of the geological facts of a single region, modern geology became too complicated. But up to his last days, he spent most of his time in the polar regions, organizing new expeditions.

Larisa Popugayeva also died. After her starry hour, she was plunged into obscurity to become one among thousands of geologists who gave all of themselves to their beloved work.

There is no more Ravich, Godfather of the discovery of Zaburdin, in the Arctic Geology Institute. His death was mourned by students and detractors alike. The Executive Committee of the Krasnoyarsk region immortalized his memory by giving his name to a small unnamed bay, unknown to everybody, in the Yenisey's polar region. In his place came a young follower named Vladimir Ivanov. As a scientist, he is known as the author of two books: a popular scientific brochure, 'The Mystery of Two Oceans,' and 'Going Along Lenin's Places in Leningrad.'

In 1983, Ivanov, following the tradition, participated in an expedition to Antarctica (Ravich, his mentor, had been chief of the geological part of the Soviet Antarctic Program). Venomous tongues were quoted as saying, "Now we have to expect the appearance of Ivanov's new book under the title, 'Going Along Lenin's Places in the Antarctic.'" The best recommendation for Ivanov, as a scientist, was given by the man who took him upward. To the Director of the Arctic Geology Institute, Ravich recommended Ivanov for membership in the Communist Party, "You know, there are a lot of narrow specialists, but Vladimir Ivanov is a specialist in all areas, something which is much needed by our Institute. Yesterday we needed a specialist in radioactive raw materials, and he became a geochemist in solid rocks. Now our main goal is oil, and he has become an oil geologist. He is the 'what will you be pleased with' type of specialist."

And in the uppermost echelons of power, there also appeared a new style, with a new Director - Igor Gramberg. A brilliant bureaucrat in geology, he led the Arctic Geology Institute to new heights. Under his leadership, the Institute changed its name and specialization twice in a decade and became a first-category (national) institute. For a decade the Institute worked on oil on the continental shelves, but now (probably because all problems of oil-bearing arctic shelves were solved by the Institute) it has become the National Institute of Oceanic Geology with a consequent increase in salary for the administration. Practically every year, to the great profit of the Institute's chiefs, it is a winner of the Socialistic Competition among the organizations within the Ministry of Geology. Regional descriptor-geologist Gramberg, as an administrator, was appointed to the list of distinguished oil geologists, being first among those who have never in this lifetime seen crude oil. He is, therefore, privileged to calculate billions of tons of hydrocarbon reserves for the Soviets, based upon potential hydrocarbons thought to be dispersed in hard rocks. A great master of political combinations, not in vain has he obtained a position as a Correspondent Member of the U.S.S.R. Academy of Science - the rarest of cases for a geologist who is with the Ministry of Geology (not an academic-type organization) and who lives in Leningrad, not in Moscow.

But whatever the Institute's specialization at any point in time, it traditionally preserves a section devoted to geologic study in the area between the Lena and Yenisey Rivers, because "The Institute is dedicated to preserving a school of great specialists in the geology of this particular region." These are the same people, of course, who missed diamonds, and due to whose activity, all geological exploration in the territory has been postponed for decades.

And to the power in high places, there has come a new generation -- one of a kind people -- students, and the best polar geologists, such as Milashev, Vishnevsky, Yegorov, and others of the same breed. They are of very different styles, these people. Some of them stay in positions of pure theoretical workers; others, in contrast, state that they are interested exclusively in pragmatic practice, "We are not theoreticians. We are looking for ore and oil only."

EXPLORATION CONTINUES

But so rich is this country, so great the creative drive among the explorers, that a continuous string of advances in exploration techniques was followed by an array of mineral discoveries. Several seasons after the pyrope-tracing technique was adopted, Galina Anastasenko proposed a new indicator mineral, chrome-diopside, which can survive transport only over short distances. At the same time geophysicist Vadin Litinsky worked out newer and more direct methods of exploration: geochemical sampling and magnetic susceptibility in soils.

In succession, geologists Boris Lopatin, Sergey Tabunov, and Andrey Ukhanov described new suites of kimberlites along the eastern slope of the Anabar Shield. Other teams investigated the northern margin of the stable craton, and instead of kimberlites, they found a new province of ultramafic-alkaline rocks with rich rare-earth deposits in the Tomtor massif, very similar to deposits of Mountain Pass, California. Several years later a prominent petrologist Victor Masaytis predicted, and afterward found, in the northern apex of the Anabar Shield, a large deposit of lonsdaleite, a hexagonal modification of diamond, in the Popygay volcano-tectonic depression.

All work in the Anabar region is logistically difficult, especially drilling. But within 20 yr after the epochal diamond discovery, new types of exploration had been developed and were in use. And within 4 yr after the diamond discovery, the world's largest carbonatite complex, and additional diamond-rich kimberlite pipes, were found.

CONCLUSION

What can be said in conclusion? It is obvious that geology today is not a true science but akin to both science and art. Geological success depends on knowledge of empirical facts and application of a variety of ideas bearing on the subject under consideration. Therefore, today it is impossible to successfully explore for diamonds anywhere in the world without considering facts and ideas developed by Russian geologists, especially those developed during the great Siberian diamond saga.

This story, in the authors' opinion, embodies the U.S.S.R.'s image -- its inexhaustible resource wealth, the boldness of its explorationists, the ineradicable creativity of its people, and the courage of ordinary Russian geologists.

ACKNOWLEDGEMENT

The authors are very grateful to Fred Barnard, a geologist formerly with Anaconda Minerals Company, and Paul A. Bailey, President of Fulcrum Management, for help in reviewing and editing this paper.

REFERENCES CITED

Erlich, E.I., 1985, The geodynamics of the northeastern Siberian Platform and the regularities of kimberlite distribution in space and time: Transactions of the Geological Society of South Africa, v. 88, pt. 2, p. 395-401.

Moor, G.G., 1940, Potential diamond occurrences in northern central Siberia: Problemy Arctici, no. 3, p. 124-135.

Moor, G.G., 1941, Micaceous kimberlites in the north of central Siberia: U.S.S.R., Doklady Academy of Science, New Series, v. 31, no. 4, p. 363-365.
Sarshadskikh, N.N., and Popugayeva, L.A., 1955, New data about manifestation of ultramafic magmatism within Siberian Platform: Razvedka Nedr., no. 5, p. 11-20.
Sobolev, V.S., 1937, Magmatic epochs of Yenisey - Lena part of the Soviet Arctica: Transactions of XVII session of International Geological Congress, v. 5., p. unknown.

In Siberia during the early 1960s, geologic expeditions used caribou for transportation. Each mapping party had a herd of about 100 to 150 caribou that were cared for by three to five herdsmen.

GOLD FARMS OF PERU

By
Paolo Greer, Mining Historian
P.O. Box 543
Fairbanks, Alaska 99707

TO DISCOVER A PROSPECT

In October 1973, I quit the Merchant Marine in southeast Alaska to visit South America for the first time. I arrived unfamiliar with the Spanish language or local custom. I traveled overland south along the Pacific Coast through Colombia, Ecuador, Peru, and Chile and then by ship down the stormy Chilean archipelago to Tierra del Fuego. I came north from the Straits of Magellan across Argentina to Uruguay, Brazil, Paraguay, and Bolivia to descend, by dugout and barge, the rivers Purus and Amazon to the city of Manaus. Ten months later and 50 lb lighter, I returned to 'los Estados Unidos' ('the Lower 48') via Central America.

In 1978, I returned to the southern hemisphere for 5 mo with a notion to research and explore. In many dusty archives of the Republic of Peru, I encountered the oft repeated legend of a remote portion of the Upper Amazon on the frontier of Peru and Bolivia known as the Carabaya (or since 1875, as the two provinces of Carabaya and Sandia). For a couple of months, in Lima, Cusco, and La Paz, I gleaned what I could of any pertinent history of the Carabaya. Afterward, I made arrangements to accompany government gold buyers for a personal reconnaissance of the area.

When I returned home to Alaska from my second odyssey, I wrote to every major Latin American library in the States and, once I had another grubstake, visited several of the best of them. Most noted were the Benson Latin American Collection in Austin, Texas (where I spent 10-hr days, 6 days/wk for 1½ mo), the Library of Congress, including the National Archives, and the U.S. Geological Survey national headquarters in Washington, D.C. (in all, I have made three separate trips to Washington for research).

In 1981, I returned a third time to Peru and devoted several months more to intense study at any accessible source in the Republic. This time, when I quit the 'archivos' (archives), I bought a couple of hundred pounds of provisions at the native market, trucked the lot through the passes of the Continental Divide, and packed my supplies by mule down to the Inambari River. I came out, after 3½ mo, on an old Inca trail, 50 lb thinner as usual. My third adventure to South America lasted 324 days.

I returned to Peru once more for a fourth research and prospecting trip from April 1984 until January 1985.

EL PERU AND THE FRONTERA

Peru is located on the Pacific coast of South America. The Republic is 84 percent the size of Alaska. It is bordered on the north by Ecuador; on the northeast by Colombia; on the east by Brazil; on the southeast by Bolivia; and on the south by Chile.

The population is 20 million, mostly native campesinos and mestizos, ruled historically by an oligarchy of Spanish caucasions. The country is, nominally, 99-percent Roman Catholic. The language on the desert coast is Castellano (Spanish) and in the mountains mainly Quechua (Inca). Like Alaska and much of the Third World, the economy is based on exportation of natural resources.

There are 24 'departmentos' (states) in the Republic. In the southeast, on the Bolivian frontier, is the department of Puno. Here lies the 3,000 m 'altiplano,' possibly the most extensive high plains in the world. To the northeast of Puno, beyond the altiplano and the Andes Mountain Range, is the Amazon Basin.

The 'Rivers Amazon' is a moving sea. In flood, it contains one-quarter of all the river water in the world. Where its drainage touches the northern section of Puno, more than 300 in. of annual rain slam into the eastern slope of the divide. There, the bedrock is soft slate. In this land, where monsoons beat for eons on the tender rock, a precipitous topography of steep, narrow canyons are created, blanketed with a seemingly impenetrable forest. It is not like the 'triple canopy' of jungle, far down river, that one may walk beneath. This is 'Ceja de la Selva' (Eyebrow of the Jungle), where every step forward is cut out of dense vegetation that seems to close in behind. It is a dark forest of sharp vines, lightning-quick vipers, legions of biting insects, rabid vampires...and gold. These are the provinces of Carabaya and Sandia.

HISTORIA

Conquistadores

When Pizarro plundered the Inca, the conquistador introduced to Europe the greatest accumulation of gold ever encountered by western civilization, enough to put Europe on the gold standard. 'Atahualpa's Ransom' was won from many regions about the Native Empire but none so rich in precious 'oro' (gold) as north of Lake Titicaca, across the high altiplano and Andean passes, down in the fabled Carabaya.

In the early years of the conquest, Spanish fugitives from wars of Pizarro and Amalgro fled from the conflict to the rich Carabaya jungles. The placers had only recently been vacated by the Inca's own miners. In 1556, in a 'resquicio' (bedrock crevice), near a place called Callahuaya, a 'pepita' (nugget), the shape of a horse's head, was found---a pepita that weighed '4 arrobas' or '100 lb.' The nugget was sent to Spain for Charles V by the Viceroy Antonio de Mendoza, and the mining village of San Juan de Oro was founded above the discovery site (I have seen the ruins).

By 1605, slaves, who had escaped from 'golden' San Juan, discovered another deposit to the west of the village at 7,500 ft above sea level. The site came to be known as Aporoma. Only 15 yr later, in 1620, Quinones

Frisancho worked the mountain-top placer with 6,000 additional slaves, building great aqueducts to wash down the earth.

Beyond Aporoma, on the western edge of the Carabaya, flourished yet another wealthy Spanish settlement, San Gavan, on a river of the same name. There were others, in a desperate frontier, inhabited by Old and New World greed and the subjugated survivors of a wasted empire.

In the early 1700s, San Juan de Oro crumbled under civil wars between roving bands of miners, who had become warriors vying for territory. By 1767, the Jesuits were evicted from the Western Hemisphere and with them their benign influence on an unstable continent. Before long, the overseers of San Gavan and Aporoma, like San Juan de Oro before them, were sniping 'molten bedrock in hell.'

In 1808, the 'Caciques' (religious leaders) from Bolivia invaded the region and found little resistance. Soon after began the long war for 'independence' that liberated South America from Spain.

El Sabio

In 1850, a 24 yr old Italian, Antonio Raimondi, arrived in Peru. A year later, he began his 'viajes' (travels), which lasted until 1869. For nearly two decades, he surveyed the Republic on foot, studying, among other things, the botany, zoology, geology and geography of the area. From 1874 to 1880, this rustic renaissance man and great observer, known even today as 'El Sabio' (the Wise One), published, in three volumes, his life's work, 'El Peru.'

From 1880 to 1902, 37 plates of Raimondi's maps of Peru, drawn from his copious notes, were published in Paris at a scale of 1:500,000. One of his maps, published at a scale of 1:200,000, covered an area 'of all the auriferous regions of Peru, the most celebrated, without doubt, . . . known as the provinces Sandia and Carabaya.' Yet, by the time El Sabio had arrived in the early 1860s to explore the Alto Inambari, the once great mining communities of the Carabaya were merely lost legends in the Ceja de la Selva.

Post Massacre

In 1890, a Quechua Indian, Mariano Quispe, searching for 'cascarilla' or 'cinchona' (quinine) bark and gold in the drainage of the Alto Inambari, discovered a large nugget weighing 43 troy oz on a tributary called the Huaynatacuma. It came to be known as the 'espejo de oro' (golden mirror). The place where it was found, a 7-ft-wide gorge, was named El Suche for the abundant fragrant yellow flowers that grew there. Quispe returned to his village, Macusani, and showed the pepita to a local merchant, Francisco Velasco, who sent for a wealthy acquaintance, Manuel Estrada. Velasco and Estrada offered the native four head of cattle to show them the discovery site. The return to El Suche took 10 days. Quispe was never seen alive again nor did his family receive the livestock.

During the following dry season, Velasco and Estrada constructed a small stamp mill on an artificial flat at El Suche. The mill had four 450-lb stamps that dropped 16 times a minute and was powered by an overshot water wheel. At first, they recovered about 14 oz of gold daily. Eventually, they crushed 2 m^3 of select ore a day, sluicing the fines without mercury (losing perhaps 40 to 60 percent of the gold), and panning the concentrates in wooden 'bateas' (goldpans). In 1896, during one 3-mo period, Velasco and Estrada took from two of their four tunnels more than 12,000 oz of gold. They named their mine the Santo Domingo after Sunday, the day Quispe had first seen his reflection in a 'golden mirror.'

In 1896, Wallace L. Hardison of Union Oil and his nephew, Chester E. Brown, came to Peru to scout out a petroleum reserve. A well was drilled and it was determined that the oil sand extended under the sea, out of grasp. Afterward, the oil men met a member of President Castro's cabinet, a Senor Pando, who told a tale of his fabulous gold prospect in the North of the Puno department.

To examine the claim, Hardison and Brown undertook the journey by steamer to the port of Mollendo and by train to Arequipa, Juliaca, and Pucara. They continued on horseback across the altiplano and, by foot, down into the Amazon drainage, beyond the headwaters of the Inambari River. Pando's prospect, on a tributary called the Machotacuma, turned out to be underdeveloped with little evident value. However, en route, the men had passed a small working gold mine called the Santo Domingo.

They retraced their steps and soon convinced themselves of the richness of the Santo Domingo vein. The owners were offered \$360,000 for the mine (\$10,000 down, the remainder within 90 days), which was accepted. Hardison and Brown then hastily returned to the States to secure the necessary money.

The oil men arrived back at the Santo Domingo on the 87th day of the 90-day option. The deal was struck, and the new venture was called the Inca Mining Company. (Incidentally, the main shareholder of the Inca was the president of Union Oil Company, C.P. Collins, from Bradford, Pennsylvania. Collins' grandson, Dick Collins, lives with his wife, Florence, at Lake Minchumina, Alaska. Dick and Florence's daughters, Miki and Julie, frequently contribute tales of their own adventures in the bush to the Fairbanks Daily News-Miner.)

The owners of the Inca spent a fortune accessing and developing the hard-rock prospect (fig. 1) and its sister enterprise, the Inca Rubber Company. A railroad station, Tirapata, was built at 12,780 ft above sea level on the Cusco-Puno line. A 100-mi hard-surface wagon road was constructed from the depot to Huancarani, near the village of Phara and the smaller community of Limbani on the eastern slope of the divide. From Huancarani, an incredible 46-mile mule trail, often a half tunnel, was blasted out of the rock precipice to the mine (keep to the inside of the trail when passing 'mulas' [mules]---they might bump you over the edge, intentionally). The path continued beyond the Santo Domingo to Astillero on the Upper Tambopata River in the heart of the rubber country. At one time, 500 mules were used to haul out the bundles of 'gome' (elastic sap) from the Tambopata area. The

Figure 1. Hanging bridge probably first constructed late last century by the Inca Mining Company to access the Santo Domingo gold mine.

trail from Huancarani to the Santo Domingo alone was said to have cost more than a million dollars.

In July of 1898, the old Velasco-Estroda stamp mill was modified to a 10-stamp battery with a chlorination plant, 25-mesh screen, canvas apron, and hydraulic classifier. A 17.5-A, 6,600-V AC generator, fed by a double-pelton wheel, was installed nearby at Tunquipata on another tributary of the Huaynacacuma. By 1901, a telephone line was installed that reached Tirapato. (When I was down on my last trip, a 'coaseno' [a farmer-miner from the village of Coasa], panning near Tunquipata on a creek that has never known hard-rock development, found a 750-gm [24-oz] nugget that contained 10-percent quartz.)

In 1912, the price of rubber fell to less than the cost of transportation. By 1914, C.P. Collins sold the major portion of shares for the Inca Mining Company to another partner from Bradford, Senator Lewis Emery.

When Emery took over the Inca, he was 75 yr old. He lived and worked at the Santo Domingo, often attending to the duties of a common miner or pushing the laden ore cars. The native workers called him 'Tata' (Father).

Emery had installed, for $200,000 in a man-made cavern at 'Bella Pampa' (beautiful field), a 540 hp hydroelectric turbine to replace the old Tunquipata generator. For another $200,000, he added an 'all-slime' cyanide plant that extracted only 10 percent of the gold (the Senator was an 'empire builder' not a mining engineer). In 1924, before the turbine was finished or the cyanide problems resolved, Tata died.

In 1927, Clarence Woods, a U.S. mining engineer and manager of the Chojnacota tin mine at 15,623 ft in Bolivia, was returning by train with his wife from a vacation in Cusco, Peru. En route, Woods conversed with a Peruvian military officer who was familiar with the Santo Domingo. (The gold mine was well known, even in Bolivia, for its richness and high grade ore.) As the train approached Tirapata Station, the soldier suggested that they step out to see if anyone from the mine was about. By chance, the men encountered an exshift boss from the Santo Domingo. The old employee said that the mine itself was right and good, but the management was crazy. The comment impressed Woods. Once back at Chojnacota, he wrote to Emery's heirs and received permission to inspect the mine, which he was able to do six mo later.

He found that the mine was indeed 'right and good' but thought the operation inefficient. He prepared a report for his Bolivian employers and tendered an offer to Emery's family. The company for which he worked was not interested, but Emery's family accepted his terms, so Woods took the option himself.

In July, 1928, Clarence Woods arrived at the Santo Domingo with several 'gringos' (slang for foreigner) who had joined him from Bolivia. The 'optionistas' (opportunists) found 13 workers already at the site repairing the timbers and keeping the mine free of water. On August 1, 1928, 10 days later, they began to grind ore. Their first cleanup, on August 31, 1928, was 132.7 troy oz, worth $2,641.40. On September 30, 1928, the second cleanup netted 256.08 troy oz; the third, 607 troy oz. After 22 mo, the mine was paid for, and Woods had $60,000 in the bank.

Woods added a MacDougal-type roasting furnace to correct for losses in Emery's cyanide process. When he attempted to bring the new turbine on line, he was rewarded with only smoke and flame (the cockroaches had eaten the insulation). It cost $10,000 to get the facility going and it worked fine after that.

GEOLOGY OF THE SANTO DOMINGO TERRAIN

Depending on your altimeter, the Santo Domingo is between 6,000 and 7,000 ft above sea level and a mite shy of 1,000 mi south of the equator. In its day, the mine was the leading gold producer of Peru. The Inca Mining Company, which developed the prospect in the late 1890s, opened the roads that even today access the Upper Inambari.

The country rock contains gold-quartz veins, stibnite, sulfides, and tellurides. Ore from the veins was often free milling and many bonanza shoots assayed up to 500 oz/ton gold. Strata are highly folded, black carbonaceous slate with a great quantity of fossils that date from the Silurian period.

The rock drilled and broke easily, but the mine made much water. The wood (camphor, laurel, mahogany, and rosewood) for the timbers lasted only 2 to 3 mo in the Ceja humidity. During Woods' era, 150 of the 350 to 400 employees did nothing but fetch lumber from the jungle slopes. Woods even brought in a truck (in pieces, on the backs of mules and men) to haul the logs, but he could not keep a trail open for it to follow (how much wood would a Woods' truck truck if...if it were in the Ceja de la Selva?).

SOURDOUGH CHUNOS

William Charles Fredrich Julius Anlauf Gates was born in Redwing, Minnesota, July 1, 1869, and was murdered February 21, 1937, by Eduardo Gonzalez in a hut on the Tunquimayo Tributary of the Nusiniscato River in the Quispicanchi district of southeastern Peru.

When Gates was 13 yr of age, he moved with his family to the Washington Territory. At 21, he was prospecting in Idaho. Later, while working for another grubstake at a copper mine in Michigan, he heard of a small gold strike in the Fortymile district of Alaska and bought passage for Juneau.

To reach the Interior, Gates traveled with three companions over the then virtually unknown Chilkoot Pass (this was several years before the rush of 1898). While rafting down Miles Canyon, he lost courage at the sound of the Whitehorse Rapids and deserted his fellow argonauts. Gates shouldered his pack and began to portage. By the time his friends had passed the torrent, he was waiting below, ashore, puffing on a cigar. He reboarded and was known for the rest of his life as 'Swift Water Bill.'

After his grubstake ran out, Swift Water worked as a bull cook in the roadhouses near Circle. When word of the Klondike reached him, he was near enough to get a lease with six other men on unlucky number 13, Eldorado Creek. By the sixth shaft and no gold, Gates was down to one partner. The seventh attempt struck paydirt. They kept quiet, played the discovery down, bought the claim,...and took out a fortune. In November of 1896, Swift Water sold out.

It was never Swift Water's nature to invest wisely. A fellow named Jack Smith brought his vaudeville group from Fortymile and set up a tent saloon near Dawson. Smith wanted to build a new dance hall and casino in town. Gates put up the money. It was called the Monte Carlo (that is, 'Monte Carlo Night').

While it lasted, Bill Gates did not accept his newfound affluence well. Instead, he lusted for notoriety. What he did not gamble away in the limelight, he good-naturedly squandered 'standing the camp for drinks.'

Soon, Swift Water left the Klondike and followed the rush to Nome. There, in character, he won (and lost) another bonanza before quitting the Seward Peninsula for the newer 'Tanana' diggings of the Interior. Somehow, Gates got a line on what became the most productive creek in the district. Cleary, the camp that grew around his claims, was the second most populous in the area after Fairbanks. Originally, it was known as Gates City.

Cleary Town burned down in 1907, but Swift Water had already gone south a couple of years before. He became involved in several promotional schemes in California but not for long. By 1910, Gates was in Peru. Oddly, it was there that he found his niche. Between 1916 and his death, 21 yr later, he never left the Republic.

When Clarence Woods came into the Carabaya country in 1928, Swift Water had already been prospecting and mining in the region for 18 yr (Woods, too, had worked in Alaska). They became friends and were involved in several enterprises together. For three months in 1931, the two men descended the Inambari from Oroya down into the Quispicanchi-Marcapata district. They tried the gravels as they went and encountered many workable prospects. Woods, however, already had a mine and returned to the Santo Domingo. Swift Water stayed below to explore the unknown streams. He is buried there now.

I first saw a late 1920s photo of those men (fig. 2) in the Library of Congress on poor copy microfilm of a 1935 edition of the newspaper 'New West Coast Leader,' now called the 'Lima Times.' The editor's father had melted down the old, lead picture plates years ago. I eventually got a print from the original, which belonged to Clarence Woods, grandson of Swift Water's friend.

Figure 2. The man on the left is Clarence Woods, owner of the Santo Domingo gold mine. The other, with the pan, is Swift Water Bill Gates.

FOLKS - KNOW THE PEOPLE AND YOU'LL KNOW THE MINES

Lucho Kiser

I am told Lucho Kiser was relieved by Uncle Sam of his position in Luftwaffe and given free room and board stateside for the duration. In 1946, he appeared as a merchant on the Alto Inambari River in the Peruvian Amazon. His customers at the remote gold placer were hardy but simple miners from the village of Coasa far up the eastern slope. Each 'minero' (miner) had a claim, one meter wide, on the river bank. They cleaned up several ounces apiece with wooden bateas before leaving for their village's festival day near the end of the dry season. Lucho stayed on a bit and panned another $1\frac{1}{2}$ kilos.

The next season he was back, this time armed with a tire pump and a set of goggles that he had picked up in Costa Rica. He tied himself to some boulders in the river, so that he could reach six meters farther out and two meters deeper than the Coasenos could. Lucho searched the murky waters, while an Argentine companion manned the pumps (Lucho told the story well, throwing himself on the thick rug of his comfortable home and tossing each time he spoke of the current). He kept only those nuggets that he could pick up with his fingers. Even so, that particular dry season he found 11½ kilos of gold.

The Padron of Quiton Quiton

Several decades ago, before the 'arbitrarily' fixed price of the precious metal legislated most gold mines out of existence, Quiton Quiton was still a small community beside the Santo Domingo trail. The pueblo was founded on a 'cafetal' (coffee plantation) of about 150 acres, very narrow and strung out a kilometer or so along the valley. Now, the farm is derelict and virtually abandoned.

Maybe it was a combination of fatigue, the particular Quiton water, and the best coffee beans ever grown, but I never did taste a better drink. With a hot cupful before my lips, I would hardly notice the 'murcielagos' (bats) attacking the small candle flame on the table, which burned and sputtered brightly, like a jewel in the jungle's throbbing darkness.

On my last trip through, the Senora served 'Suave' (powdered coffee). Her language is Quechua and my poor Spanish could not ask why. The next morning, heading down the trail, I passed a recent grave at the edge of the compound, her husband . . . snake bite. I remember him talking of a new hard-rock gold prospect he had discovered. Perhaps he was a good man. I don't know.

I will miss the coffee of Quiton Quiton.

Don Lucho and the Ghost

Lee Woods, son of Clarence, was not so trusting as Swift Water. When out of camp, he usually carried a sawed-off Winchester. The rifle had a gold nugget for a sight. The yellow metal did not rust and showed up well in the gloom of the forest.

Raul Vill-lloch met Woods for the first time coming the other way on the Oroya bridge near Pacu. They almost punched it out but instead became friends. Years later, Vill-lloch oversaw the crew that kept the water out of the mine until shortly before Woods decided to give the Santo Domingo back to the jungle. Before that, Vill-lloch took out 200 kilos of gold, which he later invested wisely in Lima, down on the coast. Today, he owns most of the placer claims on this stretch of the Inambari.

Lucho Barreda is Vill-lloch's man on the spot, keeping track of his friend's mining interests (although nobody is mining much these days).

On my last trip, I met Lucho, who arrived down the trail from Limbani to Pacu, limping pretty hard. It seems that he left Quiton Quiton in good spirits (with several spinnakers taut in the wind, actually) and encountered, walking on the waters below Monkey Bridge, the 'ghost of a gringa.' (At least I hope it was a ghost.) In the ensuing shoot-out, Don Lucho somehow fell over a cliff but managed to escape with only a few mean bruises (and a nasty hangover).

Raul's Cholo

During and after World War II, the Santo Domingo enjoyed a brief resurgence, for tungsten. The operator was Dasso, an in-law of Lee Woods. When the new, tough man arrived from Lima to take over the Woods residence near the mine, he found a native, Raul Vill-lloch's watchman, guarding the property.

At first, Dasso could not evict this man who defended Woods' home so fiercely. One night, caught between duty to his Padron and the terrible authority of the new boss, the 'cholo' (watchman) drank himself into a rage. He torched the structure to ashes and drowned himself in the Inambari.

Francisco Portugal

Francisco Portugal lives with his wife and two sons on a farm near Pacu on the Alto Inambari. Francisco is 74 yr old and a noble fellow. He works industriously and patiently at his gold 'chacra' (farm) as well as at his normal farm (I saw only one other serious attempt at agriculture within a long day's walk in either direction along the river). He grows mandarin oranges, pineapple, bananas, 'chima' fruit (the white, sweet pulp of a tree whose trunk looks like a porcupine), sugar cane, tomatoes, beans, avocado, yucca, 'matico' and 'achihua-chihua' (the leaves make a tea to cure colds), 'ocona,' 'ichate,' 'achote,' (local herbs and vegetables) and more.

Tambo La Mina (population 40 to 50) is the largest village in the area (there are only three). It is nine hours by foot up the Inambari. Don Francisco arrived in the region in 1935. He has never been to La Mina.

Don Carlos

Originally, Charley ('Don Carlos') Patra came from Wisconsin, but he worked with Clarence Woods in the mines of Idaho around 1920. Patra preceded Woods by several years to Bolivia, where he was employed as a mechanic. Later, in July of 1928, when Woods quit as manager of the Bolivian Chojnacota mine to opt for the Santo Domingo across the frontier in Peru, Patra, who had been running the electric plant at Chojnacota, was one of the few to go with him.

When the small party of optionistas arrived at the Santo Domingo there were less than 20 hands, total, to run the mine. Everyone did whatever needed to be done at the time. Don Carlos ran the old double-pelton power plant at Tunquipata, nearby. In the early 1940s, Clarence Woods turned over the Santo Domingo to Lee (Woods) and returned to the States. Don Carlos stayed.

In September, 1984, Felipe Yapa of Calapampa Village told me that he had worked with Don Carlos at the Montebello prospect of the Inca Mining Company in the 1940s. By that time, Patra spoke only broken English, favoring Quechua (the language of his fellow workers) over Spanish. Returning to Montebello, late one evening, after a bit of drunken camaraderie, he missed his step a kilometer below the mine and fell to his death.

Lee Woods sent over an American flag from the Santo Domingo to drape the body. Don Carlos was buried in the jungle above the mill site.

TOOLS AND METHODS

Quimbalates of Rinconada Pueblo

In the native hard-rock mines, the 'barreteros' (underground miner) gut the vein. The 'quepires' (Quechua for 'ant') or 'porters' haul the ore to the surface. The rock is rough sorted by the women in an area called the 'cancha' and crushed by hand to pebble size inside a thick crown of rags.

The 'quimbalate' (a frontier or poor man's 'arrastre') is a rectangularly shaped boulder with a rounded bottom. It is set snugly in a water trough carved in bedrock and rocked by an individual standing atop. If a larger boulder is used, two or four men lean, teeter-totter, against a timber that is pinned across. Every few moments, pieces of ore are tossed into the tight gap beneath the crushing stone. When the smaller rocks are pulverized, the low-specific-gravity waste washes away. Later, the water is diverted and the golden residue scraped up. Occasionally, women will pan downstream from the operations to catch the fines that get away.

The stone huts of Rinconada (the remote mining village is at 17,200 ft above sea level), like the 'cemeterio' (cemetery) nearby, are almost indistinguishable from the mountain. Most of the dwellings have television antennas, purchased with gold extracted with the quimbalate, that crop from the thatch roofs.

Batea

The 'batea' (goldpan) is made of wood. I have never seen one available anywhere except near the mines, and usually there are few to spare. The batea lacks the pie-pan bottom of a sourdough's pan, resembling instead a wooden bowl in the shape of a flattened Chinaman's hat. It is fabricated from the materials at hand, although certain trees produce a better quality for wear and insect resistance. Ideally, a section is cut from the tall, curtain-like root of a particular jungle cedar and cured for a year. A block, 12- to 30-in. diam, is roughed out by machete and set to with a small adze. It takes about a day to chop out a 'pan' to 3/4-in. thickness.

The mineros claim nothing better exists for washing the muck. I suppose any type of pan is adequate in the hands of someone with experience and the attitude to work at it. Perhaps, though, practice makes more perfect, especially if a crude sluice box is considered a new-fangled contraption and pay gravel is habitually 'run through the pan.'

There are advantages, of course, of certain bateas over another. Larger pans of any kind work more dirt. Wood floats and does not rust, as metal would in the jungle (still, improper care, such as wetting and drying, will crack it badly, even if exposure to the sun seems unlikely in the Ceja). Too, bateas are not so fragile as even the better plastic pans attached to the back of a cantankerous mule. More pertinent, I think, is that wooden pans are familiar to indigenous folks and one could rig up a lathe to turn them out in quantity.

Ingenio

Originally, the 'ingenio' (sluice box) was only the pelt of a sheep, a 'golden fleece,' lining a ditch. Now, of course, it is more sophisticated.

The Rio Inambari ingenio of today is usually two stout limbs, crossed by spaced bamboo strips. The frame is covered with plastic and moss (before plastic was available folks would use large, durable leaves). A transverse strip of sticks, cut wide as the sluice and tied tightly together, is placed lengthwise over the vegetation (for fine gold, the moss and 'riffles' are often omitted in favor of simple, coarse burlap). Pay gravel is carried to the ingenio in bateas and washed, a handful at a time.

The minero also carries in his outfit a 'rascador' (iron hook) to crevice bedrock. If he is prosperous and industrious enough, he may pack a long bar to lever the boulders aside. Add a batea, a machete, a 'plastico' (to keep the inevitable rain off his shoulders), a charred aluminum pot, a cup, some 'chunos' (a primitive sort of food involving potatoes), 'challona' (dried mutton), toasted corn, a few coca leaves (to help dull the taste buds for eating chunos) and he is ready to wade up the river toward his bonanza.

Chacras

The Incas won their gold mostly through hard work, patience and 'chacras de oro' (farms of gold). Once, money did 'grow on trees.'

The rivers were prospected at low water by panning the beaches with bateas until a rich inside bend, as near the water as possible, was discovered. The pay was defined and a chacra created, tear shaped, conforming to the whim of topography, by firmly setting large cobbles in roughly two meter squares. The spaces between were 'empedrado' (paved) with flat pieces of slate that were upended, laid perpendicular to the water course, and tilted downstream (fig. 3). After the chacra was 'sown,' the natives would quit the site until another dry season.

In the Ceja, the 'Rivers Amazon' plunge from the skies on soft eastern slopes, creating incomparable torrents of great volume and irresistible

Figure 3. Modern gold farm, Peru.

force. Unusual amounts of muck and gold are carried over the natural and man-made 'riffles,' allowing them to become enriched (an experienced miner will build his 'pavement' outside the avenue of principal violence, lest nature tear out his hard labor or bury it under five meters of fresh alluvium).

The slate does not simply abrade to sand. The soft rock uproots along cleavages and is eroded into irregular boulders and cobbles. The sharp edges 'lock in,' forming a matrix of integrated overburden, relatively impervious to displacement, until the particular storm hits that bursts the river bottom asunder and scours the bedrock.

Today, the chacras de oro are still farmed, inherited over the generations by indigenous folk who come down from the altiplano when the rains briefly subside. The miners harvest only as many squares as can be worked in a day (the river could flood at any moment during the year, without warning, and the empty cells are a weak link where the enraged waters may gain purchase on a gold farm and dispose of it down the Amazon). A diversion ditch is dug as near the workings as possible and lined with fleece or ingenios. The pay is carried from the 'empedrados' (sluice) and washed. The better chacras produce 0.3 gm (about 1/5 dwt) of gold/m^2.

Incidentally, this method will not work so well up 'Grizzly Bar Crik' off the Tanana-Yukon Rivers, even if the Alaska stream is very rich (Swift Water Bill or some other sourdough probably scraped bedrock thereabouts in 1905, anyway). It takes not only the unusual gold content of the Carabaya to make a 'farm' so rich, but the specific interaction among soft, irregular bedrock, mighty rains, and steep, narrow canyons.

LOGISTICS

Viveres

I have heard that garlic keeps the vampires away (fig. 4). Once, I took 5 k. of this tasty remedy down to the Inambari but much of it went sour in the humidity of the rain forest. The next trip I brought plenty of powdered 'ajo' (garlic) from the States. The rest of my provisions I buy at the Campesino market in Juliaca near Lake Titicaca. Dried goods are easily available since refrigeration is not.

My typical fare includes a variety of beans, rice, mushrooms, peas, 'chunos' (dried potatoes; that is, a small variety of tubers, grown high in the Andes, that are scattered on a dirty tarp and trampled underfoot to let any moisture escape, which leaves a weightless relic of a vegetable with all the flavor and texture of moldy cork), seaweed (beaucoup), olives, garbanzos, rolled oats, rolled wheat, soy and corn flour, 'chaquepa' (coarse-ground barley) ground 'canihuaca' and 'quinua' (good-tasting grains with much protein, not available homeside), raisins, prunes, figs, almonds, brazil nuts, peanuts, dried cherries and coconut, jellies, powdered milk, cocoa, coffee, 'api' (a Bolivian drink made of maize), powdered fruit drinks, cinnamon, oregano, anise, sugar, honey, salt, numerous prepared soups and cereals and spaghettis, olive oil, mustard and other sauces, margarine, crackers, cheese, sausage, matches, carbolic soap, and candles.

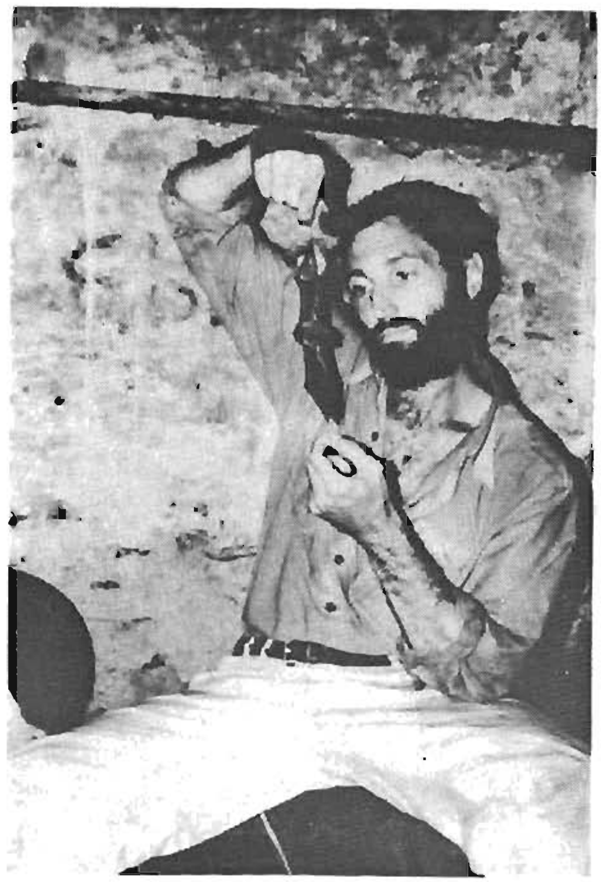


Figure 4. The author inspecting an unwanted but benign guest.

I triple wrap the food with clear plastic bags, in batches of several pounds, and sew the lot into comfortable, mule-sized loads under three thick, synthetic fiber bags for protection against 'arrieros' (muleskinners) or an epidemic of jungle cockroaches.

Critters

The jungle may keep your health or wrack you. There are piranha, jaguars, vipers, and vampires (in 1974, in a dugout between Bolivia and Brasil, I met a foot-thick anaconda that was quicker than any other living thing). You awaken at night, thinking the tin roof of the 'tambo' (roadhouse) is leaking, and find your face or toe awash in blood. You may enter a 'vace' (cave), inhale, and die insane (the bats groom themselves fastidiously, and their spittle becomes volatilized). A breath in the dark could fill your lungs with rabies---insidious, mortal, and far from the nearest dirt road.

Forget the vampires. The armor-plated scorpions, larger than the big tarantulas, with legs and antennae over a foot long and mismatched, lobster-like claws, make rabid, flying rats seem benign.

Forget the scorpions. The insects are everywhere. They arrive from all directions with a chitin biomass of many tons to one remote, gringo soul. The quantity of 'bugs' is stupendous and the variety...fascinating, but do not touch and try to avoid being 'touched.' I am told the 'tabanas' (horseflies) have their own saints day. I know they have swords for tongues. The 'mantas blancas' or 'white shawls' (no-see-ums) dive through your mosquito netting and hair with the effect of a cluster bomb. The 'zaccapalas' (cockroaches in Quechua) swarm in tides during the full moon and eat anything that cannot move away (best to sprinkle your buddy with a little sugar water when he knocks off for the night). They will riddle even new, unused plastic containers.

Set an empty food can, unwashed, with hinged lid bent down, on the floor for the night. Come morning, take it, nigh full, outside the hut and gather a quorum of 'pollos' (chickens). The birds become feathered, squawking piranha, and their eggs the following day, with blood-red yolks, are the most satisfying that I have known.

Also too common is the 'sututo' or 'tornillo' (bot or tropical warble fly). The parent insect may lay its eggs on your laundry when it's spread out on the rocks to dry. Larvae enter through pores or hair follicles and gestate for 2½ to 3 mo, growing from 18 to 24 mm (an inch) before hatching. They squirm in your flesh but are rooted with too many hooks to pluck. A 'gringo medico' (city doctor) would cut the parasite out, but secondary infections in the jungle are trouble. A better way, also painful, might be to suffocate the worm with tobacco juice under tape. Best, would be to strap some pig fat over its hole. Within a day or two, the 'tornillo' should swap places.

The fireflies have two beacons on their heads with flames that seem never to expire. The caterpillars display colors unimagined, laced with toxic spines. The butterflies are bits of rainbows, come unstuck and burst into movement. Move down river or up a mountain slope and a whole new multitude of species appears. Take your 'goretex' tent into the Selva and a billion leaf-cutter ants could mistake it (or your occupied sleeping bag) for food supplement and notch the lot with bite-size skylights to let in the ambient monsoon. Once, I awoke, writhing in pain, to find myself covered with several hundred ants chewing the sweat and salt out of my pores. A day later, I swam the Inambari, between a whirlpool and the rapids below (my fingernail and tooth marks are probably still visible on the moss-covered opposite ledge, where I finally ground to a halt a short ways before the maelstrom). I sat to rest in a bamboo thicket and thought I had skewered my arm. Several 1½-in.-long ants had welcomed my company with open mandibles.

Et cetera.

Forget the ants. Too many mites suck blood. Often, the host before you was infected. A common malady (wanna see my scars?) is leishmaniasis, a kind of paraleprosy. The textbooks say (if you survive) leishmaniasis will cure itself...spontaneously. I heard that a German kid, down river, lay in the jungle hospital until the disease ate his throat away. Go down to the river to wash and get bit by a sand flea. The infection begins as a pimple but contains protozoa that eat flesh. Shortly, a lesion is growing, wide, deep and surprisingly painful. I brought what fungacides I could purchase in Lima, both grease and powder, to no avail. One day in Pacu, Juan Cordova asked why I did not put some 'cuti-cuti' leaves on my ulcers. I did. It worked, again and again. Good old cuti-cuti.

THE INCA

Figure 5 shows a turn-of-the-century photo (from Dick Collins' collection) of the mill at the Santo Domingo mine. It was erected in 1890-91 by Velasco and Estrada and improved in 1898 to what is shown here by the Peruvian mining engineer, Fuchs. Later, Chester Brown, manager of the Inca Mining Company, constructed an entirely new mill that has long since been 'remontado' (overgrown with jungle).

In 1981, I spent two weeks at the site of the Santo Domingo. I sought the location of the mill. It was a ways off from the vestiges of the mine. I waded up a creek that my research had indicated and returned to a place, the only one, where the slope was not quite so steep. I panned the dirt banks and discovered anomalous colors, mostly scheelite and a bit of gold. I climbed above, cleared an area with my machete, and detected rusted fragments of forgotten scrap---tools. Nearby, I dug ore tailings, at first invisible, from below the muck, below the roots of the forest. I discerned an old ditch line that had led to a water wheel. The wheel, now so long silent, had powered the stamps where men once labored to break golden rock.

Today, the sounds are different. The 'Tunqui' (a flame-red cockatoo) mutely guards its cliffside chambers. The large, yellow and green 'Cuchos' build meter-long, teardrop-shaped nests that are suspended from the limbs above the river and, like the 'Stellar' jays of home, enforce their presence with raucous cries overhead. The incredible chime of the 'bell bird' re-

Figure 5. Mill at Santo Domingo gold mine, Peru.

sounds through the green Ceja. The primordial forest lives on...oblivious to the brief interruptions of mankind?

GOING BACK TO SACRAMAYO

There is a particular river that I know, not 20 km in length. It drains a steep box canyon in the Ceja. In 1849, this jungle stream produced more gold than California streams produced in the same year. The Peruvian military built a road from the Inambari to the stream's headwaters. Another well-used path already approached the diggings from the opposite side, up another ravine. Today, there is no evidence of either trail on blow-ups of aerial photographs.

The small river has three tributaries that form it: the Pucamayo (Red River), where much of the gold was mined; the Quimsamayo (Third River), also very rich; and the Chaupimayo (Middle River), untouched according to history and local account. The drainage of the Chaupimayo is several times that of the other two combined. It carves sheer canyon walls far up the mouth---dangerous access. Above, plentiful 'tigres' (jaguars) and serpents are well noted by the few old timers aware of the river's existence. Now, they call it the 'Sacramayo' (Devil River).

The aerials do show a trail, though, a bold strip, high up on the mountain ridge. It is an Inca road. History does not mention it. The locals did not know it was there. It leads toward Devil River.

FORMATION AND CHARACTERISTICS OF PLACER DEPOSITS

By
D.W. Parkhurst, Mining Consultant
P.O. Box 4179
Carson City, Nevada 89702

INTRODUCTION

The subject of this chapter involves a broad range of material, and due to limited space and time, I have condensed the information under several headings: placer derivation, alluvial placers, beach and marine placers, residual and eluvial placers, desert placers, ancient placers, glacial-stream placers, eolian placers, and reconcentrated placers.

For a basic definition, a placer deposit can be described as any localized concentration of heavier and more durable minerals that has resulted from surface erosion. The most notable characteristic of placer deposits is that the majority of them are not typical. Each individual deposit tends to be slightly unique, and, in many placers, the type of materials found in the deposit varies considerably over a relatively short distance. The more typical placer is usually one that has undergone extensive alteration subsequent to its original formation.

PLACER DERIVATION

Mechanical concentration of placer minerals is accomplished by gravity separation of the heavier and more durable minerals from the lighter and more soluble minerals by the combined action of wind, water, and gravity. In most instances, the materials, disintegrated by mechanical weathering and chemical action, move slowly down slope to the nearest stream or shoreline, where the moving water sweeps away the lighter minerals, while the heavier, more resistant minerals sink toward the bottom. In residual deposits, wind, rainfall and chemical action gradually separate the minerals. Eventually, the valuable minerals contained in many thousands of tons of matrix rock are concentrated into a relatively small volume of sediment.

Although gold is often considered the most important mineral in placer deposits, there are many other placer minerals which may occur in sufficient quantities to warrant an economically viable mining operation. These include cassiterite, chromite, columbite, ilmenite, magnetite, monazite, rutile, scheelite, tantalite, zircon, and the platinum-group metals. These minerals are concentrated in placer deposits under the same conditions that regulate deposition of gold particles. Many of these minerals are being mined as primary economic commodities in some deposits, and they are also being recovered as significant byproduct or coproduct minerals in other operations. Other mineral deposits that result from residual placer concentrations add to the list of placer minerals mentioned, including iron ore, manganese, cobalt, bauxite, nickel, phosphate, kyanite, barite, ocher, and tin.

ALLUVIAL PLACERS

Alluvial deposits are formed where placer minerals have been transported by running water to a new location, as in rivers and streams. Most of these placers are classified by their location or type of formation, such as bar, alluvial fan, gulch or ravine, creek, stream or river, bench or terrace, and delta or flood-plain deposit. Some of these placers have little or no relationship with the present topography, depending upon the geologic conditions when and since they were formed. Some deposits are found on the tops or sides of hills, where they were left by streams that changed direction or that disappeared entirely as the surface of the earth changed due to crustal uplift or subsidence. Also, as streams or rivers cut deeper into rock strata or the direction of flow changes, gravel bars are often left behind at higher elevations.

The process of mechanical concentration in water involves the size, shape, and specific gravity of the mineral particles and the velocity and movement of the water body. Heavier particles sink more rapidly in water than lighter ones of the same size, and the ratio between their respective specific gravities is higher in water than it is in air. The settling rate of the particles is also affected by their surface area: when two particles have the same weight but differ in size, the smaller will sink more rapidly. In addition, the shape of the particles affects their settling rate, as a rounded particle will settle much more rapidly than a flat or coarse particle.

The ability of flowing water to move or transport a solid object depends on the velocity of the water and varies according to the square of its velocity. If the velocity doubles, the transporting power increases to about four times the original force. Conversely, if the velocity is halved, most of the load is dropped. Most placer deposits are formed wherever the current flow decreases.

Accumulation of placer minerals in gravel deposits requires a long-continued adjustment between water velocity and the amount of material transported, as well as a fairly continuous supply of placer minerals. Lindgren (1933) noted that the largest and richest placer deposits are formed in streams and rivers that have a gradient of about 30 ft/mi. The gravels cannot be too thick, have to be moving slowly downstream, and have to be water-soaked for a 'jigging' action to occur. The presence or absence of these conditions largely determines whether the placer minerals accumulate or are dispersed throughout the gravel mass.

A constant flow of water will produce well-rounded and smoother gravels, because of the continual abrasion between individual rocks in the mass. Abrasion also releases more placer minerals by wearing away the host rock. As the placer minerals are released, they are either pounded and flattened, or fractured and ground into fine granules, depending upon their malleability or brittleness.

Water turbulence in stream currents is similar to the upward pulsations and vibrations of jigs and tables used for concentrating ores. The shaking action tends to move the lighter particles upward, enabling the heavier minerals to move downward and gradually concentrate toward the bottom of the channel. A change in gradient, an obstruction, or a meandering or widening of a watercourse all produce conditions that allow heavier placer minerals to drop and accumulate.

The most favorable regions for placer deposition are usually those that have appreciable topographic relief and contain highly mineralized rock formations that have undergone extensive decay and erosion. Sometimes even more favorable localities are those that have undergone recent geologic uplift. The newly incised ravines and canyons expose older gravel structures, which results in rewashing and reconcentration of preuplift gravels. Generally, the more times such reconcentration of older gravels takes place, the higher the degree of concentration obtained in the newer placer deposits. In some cases, however, the placer values contained in the older deposits are dispersed throughout larger volumes of younger gravels.

Most river and stream gravel deposits will have barren stretches among the richer concentrations in pockets, bars, bends, and 'streaks.' The richer placer deposits are usually found on the inside curve of meandering streams, in pockets behind boulders, in gravel bars that accumulate directly below major obstructions to water flow, and in areas where the channel widens or levels off immediately below a narrow or steep section of the watercourse. In areas with a high stream gradient, very little placer material will accumulate in the gorges because heavy runoff usually scours the stream bed.

Older, intact alluvial gravel deposits, such as bench or terrace deposits or stream channels where flow was cut off, usually exhibit a pronounced 'layering' effect. Separate and distinct layers of gravels that have an obvious separation, both by type of material and mineral content in each layer, reflect the conditions existing at the time each individual layer was deposited. Some of the layers may carry significant placer values, whereas others are relatively barren. The richer gravel areas may or may not bear a relation to bedrock. Deposits have been found where the bulk of the placer minerals occur at or near the surface, whereas the deeper gravels are relatively barren. In other deposits, the mineral occurrence is reversed.

Alluvial-fan gravel deposits are normally a combination of multiple channel systems, deposited during periods of heavy runoff or flash flooding, that interlace bench and terrace gravels. In wetter climates, channel gravels and mineral particles strongly resemble those found in most stream and river placers. Bench and terrace gravels can be similar to channel deposits but are more often combinations of well-rounded and smooth gravel with angular and subangular material. Mineral particles in bench and terrace deposits also vary considerably. They can be smooth and rounded, flat or flaky, or extremely coarse and jagged. Fan gravels are normally deposited at the mouth of canyons or rivers in sloping terrain. Fan gravels in dry climates are entirely different, as will be described later.

River-delta and flood-plain deposits are more difficult to classify because they normally grade into river-channel and bench placers. These deposits are found where rivers or streams enter standing bodies of water, low-lying valleys, or wide, flat canyons. The delta and flood-plain placers are similar to river placers except the delta and flood-plain placers are considerably more layered, much larger in size, and contain mineral particles that are usually smaller and distributed throughout a greater volume of gravel. Because most flood-plain and delta deposits are formed by a shifting in the river channels and flash flooding, the placer minerals are more likely to be distributed over a much greater lateral and vertical extent. Also, since the waters flowing over flood plains normally have a low velocity, except in times of flooding, the placer minerals are usually composed of small to extremely fine particles. Larger mineral particles will normally be dropped at the upper edge of the delta, where water velocity is drastically reduced. Most of the delta and flood-plain deposits are fairly permanent, although surface materials are subjected to occasional flood wash and erosion.

Because of their greater volume and more general distribution of placer values, flood-plain, alluvial-fan, and river-delta deposits have been mined extensively by means of low-cost bucket-line dredges. Most placer production in the past was obtained from deposits of this type, and several dredging operations today are still operating profitably in them.

BEACH AND MARINE PLACERS

Many economical beach and marine placer deposits have been discovered and mined throughout the world and have produced significant quantities of gold, monazite, platinum-group metals, precious stones, tin, titanium, zircon, and several other mineral commodities. Beach sands are currently the major source of monazite, a rare-earth phosphate, and zircon, a zirconium silicate. Large quantities of placer gold have been extracted from beach placers in the past, and large-scale bucket-line dredging operations are presently being conducted in several localities.

Mechanical concentration of minerals in beach and marine placers is similar, but not identical, to concentration of minerals in stream placers. Only those minerals possessing a high specific gravity, hardness, or malleability and a resistance to chemical decomposition will be concentrated into profitable placers. Waves and tidal action on beaches causes continuous movement of surface sand and gravel. The resulting friction between particles gradually grinds the materials into a fine sand. Lighter and less resistant material is washed away, whereas heavier and more durable minerals settle slowly toward the bottom. Less chemically resistant minerals are dissolved and removed.

Wave action produces a pounding and vibrating effect that tends to shake the mass of material and causes the heavier mineral particles to settle downward. This jigging action slowly concentrates the heavy minerals on the first impervious layer of clay or on bedrock beneath the sediment surface. Over long periods of time, this sorting and concentrating process can reduce a tremendous volume of sand and gravel into a relatively thin layer that contains almost all of the heavier and more resistant minerals originally distributed throughout the larger mass. Tidal action and shoreline currents aid in the process by continually removing the lighter materials and by bringing in additional heavy material for sorting.

In some areas, lode deposits or massive rock formations that contain widely disseminated placer minerals are eroded by these same processes, which produces a combination of an eluvial or residual placer in a beach placer. A similar concentration of placer minerals occurs where river deltas and alluvial fans are reworked by constant wave and tidal action.

Beach placers are found not only along areas bordering the oceans but also on fresh-water-lake shorelines, on ancient ocean beaches now located far inland, and on shorelines of ancient lakes that have long since disappeared.

Geologic uplift and subsidence of coastal regions often displaces beach deposits from their original position by several feet to several thousand feet in elevation. As a result, economic beach placers can now be located in underwater areas or several miles inland from the present shoreline. Lindgren (1933) makes reference to two older beach lines that were uplifted in Alaska, both of which contained significant placer gold.

More recently, a major effort has been organized to dredge submerged gravels offshore of Nome, Alaska. This deposit has been reported as a river-delta formation that has been reworked by wave and tidal action. Another beach placer has been worked for platinum values at Goodnews Bay for a number of years.

Most beach placers are characterized by thin-bedded and lenticular placer concentrations that contain fine to extremely fine, smooth particles of placer minerals; the bulk of the material is very fine sand.

RESIDUAL AND ELUVIAL PLACERS

Concentration of placer minerals in both residual and eluvial placers is accomplished almost entirely by disintegration of mineralized rock in a relatively confined area. Residual placers are formed by in-place decomposition of mineral-bearing formations or ore deposits on relatively flat terrain, whereas eluvial placers result from similar conditions acting upon rocks that are located in sloping terrain.

The basic difference between the two types of placers is that eluvial deposits are subjected to greater gravity movement and transitional creep and to more movement by rainfall and water runoff because of the sloping terrain. As a consequence, the highest concentration of placer minerals in an eluvial deposit is usually near the bottom of a slope or in a thin layer near bedrock that extends a short distance down slope from the source.

Since residual-placer minerals are eroded in place, they are normally subject only to vertical movement as the rock disintegrates and the heavier and more durable mineral particles migrate downward. A certain amount of sorting and concentration of minerals is effected by the combined action of wind and precipitation, but the concentration of valuable minerals is largely dependent upon the minerals' relative abundance in each unit volume of the decomposing host rock. Richer placers form by disintegration of higher grade ores or deep, extensive decay of massive mineralized formations, whereas lower grade deposits form by decomposition of low-grade ores, the superficial decay of mineralized formations, or partial disintegration of higher grade ore minerals.

Another form of residual and eluvial placer is found in regions where massive rock formations have undergone deep weathering, which produces large placer concentrations that resemble primary ore deposits. Most of these deposits are found in tropical and subtropical regions, where the conditions are conducive to extensive rock decay, or in areas where tropical conditions were present in the past. These deposits are formed over extremely long periods of time; thus constituent minerals have often undergone considerable alteration in both physical character and chemical composition. Many of these deposits are mined by open-pit methods for iron, alumina, or clay. They often contain significant quantities of other interbedded placer minerals, which can be extracted as byproducts.

Residual placers are sometimes called 'seam diggins,' and they have been found on top of several notable lode deposits. The 'roots' of vein structures or ore bodies are normally found directly beneath residual concentrations, except where deeper weathering has completely disintegrated the original host rock. Eluvial placers are commonly found directly below ore deposits from which they originated and are often called 'hillside' placers. Various combinations of the two types of placers are commonly found together near the original source of mineralization.

Residual reconcentrations of placer values also occur on top of older alluvial placers, particularly those which have been either left as terrace gravels or have undergone uplift subsequent to their deposition. This sometimes produces richer surface concentrations on the top of lower grade stream gravels. In sloping terrain, this reconcentration of surface gravel is further enhanced by wind, rainfall, and runoff. In some areas, this form of residual reconcentration produces higher grade surface gravels than those found toward the bottom of the deposit.

The physical characteristics of residual and eluvial placer materials are quite distinct. The bulk of the gravels are very rough, jagged, and angular, with little evidence of the rounding and polishing typical of stream-placer gravels. Metallic particles and mineral compounds, unless chemically altered, will retain most of the same physical characteristics that were present originally. Intact crystals are common, as well as rough and coarse metallic particles. Soluble salts commonly form encrustations on the rocks or, if the soil and gravel is permeable, form layers of caliche, clays, or encrustations near bedrock or the prevailing water table.

An important consideration should be taken into account when dealing with these types of placers. Since the mineralized rocks have been subjected to minimal decomposition, and abrasive action between the rocks is practically nonexistent, a significant percentage of the mineral values will probably be locked in pieces of the original host rock. This is particularly so if the matrix material is quartz, or some other hard and durable rock, or if the valuable mineral particles are predominantly very small. Because there is normally only a superficial concentration of placer minerals in these deposits, the higher grade concentrations will tend to be highly erratic and localized.

Conditions conducive to the formation of eluvial and residual placers exist wherever older mineralized formations have been subjected to decomposition and erosion over extremely long periods of time or where softer ore minerals have disintegrated over shorter periods of time.

DESERT PLACERS

Depending on the geologic time period and the conditions under which they were formed, desert placers form by any of seven basic processes of placer formation, namely: alluvial, eluvial, residual, glacial stream, eolian, beach, or marine. Many of the original forms of placer deposition have been partially or wholly altered by changing conditions throughout geologic time.

Desert gravels are generally composed of angular and subangular materials, the bulk of which shows very little, if any, rounding or smoothing as a result of stream action. Due to the absence of constant water flow and the roughness of gravel, the heavy placer minerals usually can not work their way down to clay or bedrock as rapidly as they would in a steadily flowing stream. Being suspended at higher levels in the gravel, the placer minerals are more accessible to secondary movement by subsequent surges of high runoff. Because of the tremendous cutting power and turbulence associated with sudden surges of heavy water flow, the mineral particles are redistributed throughout large volumes of gravel. Unless there are subsequent extended periods of fairly constant stream flow, the heavy metallic particles have very little chance of being sorted and concentrated into confined placer deposits.

For these reasons, most desert placers tend to have a fairly even distribution of small-to medium-sized mineral particles mixed in the gravel mass. It is also not uncommon to find larger and heavier particles near the surface with smaller particles located at various levels beneath. This results because smaller particles can work their way downward more easily than larger and coarser particles, due to the angular nature of the gravels and the lack of movement of the mass. The reverse is encountered during periods of maximum water flow or when larger particles are eroded first.

Intermittent surges of water over dry gravels results in multiple layers within the gravel mass. These layers are generally more spread out and composed of a greater mixture of material than the stratified gravels normally found in stream channels. In most desert regions, these layers are separated by narrow seams of clay or caliche that formed during the extended dry periods.

Depending on the geologic time period in which the heavier placer minerals were eroded and the cutting power of each surge of water flow, some of the gravel layers may carry significant placer minerals, whereas others do not. In some instances, mineral-bearing gravel layers are separated by layers of barren gravel or only carry values in the upper layers. In others, the layering is reversed or is in an entirely different configuration. This makes these deposits very difficult to evaluate accurately.

The intermittent surges of water flow in arid climates are usually present for only short periods of time, such as immediately after a thunderstorm, cloudburst, or rapid spring snowmelt. The flow of water is usually of such a brief duration that the underlying gravels are not softened or saturated with moisture. Consequently, the vibrating or jigging action of the gravel by water movement is limited to the top of the gravel mass. Under these conditions, larger mineral particles are deposited in narrow streaks of limited length and depth, whereas smaller particles are distributed throughout the surface gravels. Concentrations of placer minerals tend to be very irregular in form and are erratically distributed throughout the deposit.

Because of limited water transport of most desert gravels, very little abrasion occurs between pieces of the eroded material. As a consequence, most of the placer particles are very coarse and jagged, and a significant amount of the values remain encased in the matrix rocks. As a result, the heavy-mineral concentrates obtained from certain desert placers must be ground before final processing to liberate the minerals from the host materials. In some deposits, larger chunks and boulders of ore minerals are extracted from the placer material and processed in the same manner as ores from lode deposits.

Alluvial-fan (bajada) gravels deposited in desert regions are also formed during periods of heavy runoff or flash flooding by multiple channel systems. In desert placers, however, there are usually a much higher number of individual channels contained in the fan. This results from the lack of relatively constant stream flow, which is necessary to form deeper and well-defined channels. Most of the channels are fairly shallow and narrow, and they exist for only short periods of time, because they are usually cut off and buried by the next surge of material that accompanies a flash flood. This produces a series of multiple channels in the mass of gravel, which are usually spread over a wide area--both laterally and vertically. This in turn produces a very irregular pattern in the terrace gravels, which normally do not exhibit the uniform stratification typical of gravel deposits. All of these factors contribute to the more erratic distribution of mineral particles throughout the mass, as there is little chance to concentrate values by the transporting and sorting processes prevalent in wetter climates.

The nature of mineral deposition in desert placers creates ideal conditions for the formation of large, medium- to low-grade placer deposits. It is also conducive to the formation of hidden placer deposits, buried under a few feet to several hundred feet of detrital material, as well as fairly rich, localized concentrations of placer minerals.

Certain older placer deposits now found in desert areas were formed during periods where conditions were conducive for the formation of more typical stream and river deposits. These older placers exhibit many of the same characteristics as those found in wet climates today.

ANCIENT PLACERS

Placer deposits formed in earlier geologic time are commonly referred to as ancient placers. This category would, of course, include all types of placers formed prior to more recent times, including the more famous Tertiary and Quaternary river-channel deposits found in California, Alaska, Australia, Canada, and South Africa. These older gravel deposits, and those that were subsequently eroded and reconcentrated into newer placers, were largely responsible for many of the major gold rushes of the past.

In some regions, entire mountain ranges were eroded away and the resulting alluvium gradually disintegrated over millions of years by constant movement and abrasion of gravels in ancient river systems. Some of the rivers existed so long that 'white channels' were formed, where only the hardest or most chemically resistant constituents, such as quartz and gold, remained. The less resistant materials were either deposited as sand, silt, or clay or were dissolved. The transporting, sorting, and wearing away of such a tremendous amount of rock resulted in some of the richest placer concentrations known.

Many of these older river systems were very extensive and deposited considerable volumes of placer minerals in multiple channels and in bench or terrace gravels. Subsequent to their deposition, lava flows, stream overloading, and regional subsidence eventually buried some of the placer gravels to depths of tens to hundreds of feet. In California, the ancient river systems were first buried, then uplifted several thousand feet, and later exposed in newly eroded canyons and ravines. In other areas, such as Alaska and Australia, placers remained buried, and their gravels were extracted by drift mining to depths of several hundred feet. Bateman (1950) refers to several rich gold-bearing channels at Ballarat, Australia, that were drift mined to depths of about 300 ft and similar gravels at Fairbanks, Alaska, that were mined to the same depth.

Older gravel deposits have been compressed into conglomerates or 'cemented' gravels that range in age from Cambrian to Recent. These lithified placer deposits have been mined for gold in South Dakota, New Zealand, South Africa, and California.

Many modern river and stream placer deposits are reconcentrations of placer minerals contained in ancient placer deposits. Modern deposits were further enriched by the influx of more recently liberated placer minerals.

GLACIAL-STREAM PLACERS

Glacial-stream placers, or gravel deposits that are transitional from glacial moraines, are mentioned here for reference, as they are not normally a significant source of placer minerals.

The tremendous erosive power of glaciers can be seen by observing the large, dish-shaped valleys that were carved out of solid rock by glaciers in the past. In some areas, glaciers deposited huge mounds of detrital material, called moraines. Where the moraines contained significant quantities of placer minerals, meltwater streams transported and sorted part of the material as the glaciers retreated. In this manner, localized glacial-stream placers were sometimes formed that contain sufficient values to warrant placer mining.

The material in glacial placers is normally characterized by angular to subangular gravel that contains very finely ground placer minerals and large quantities of glacial silt or 'flour.' Depending on the length of time these streams flowed, some rich localized placers were formed. In certain localities, glacial till in the moraines was mined for its mineral content.

EOLIAN PLACERS

Eolian placers are also mentioned here for reference, as only a few have been of economic significance. The process of concentration involved, however, is sometimes significant in the formation of several types of placers.

The action of wind in blowing away lighter material from the surface of placer deposits is a significant factor in placer concentration where high winds prevail for much of the time. In more stable areas of the Earth's crust, wind action over a period of several million years can move a tremendous amount of material. In addition, the erosive force represented by the abrasion of wind-borne particles of sand should not be underestimated. In some dry areas noted for high-velocity dust storms, the greater part of mountains has been worn away by this force acting over long periods of time.

RECONCENTRATED PLACERS

A number of the placer-mining areas that have been worked extensively in the past have potential for formation of newer placer deposits by the natural reconcentration of values. Usually, the longer the period of time that has elapsed since the area was mined, and the greater the extent of subsequent erosion, the better the chances are for formation of new placer deposits.

Lindgren (1933) stated that many placer areas would be reworked in the future as a result of 1) natural reconcentration of values lost in original mining operations; 2) accelerated erosion that results when tremendous amounts of material are disturbed and redistributed by mining activity; and 3) natural erosion, which provides an influx of new placer minerals.

By the time mining activity ceased in most placer districts, a tremendous volume of sand and gravel, containing widely disseminated particles of placer minerals, had been dumped into streams and rivers. As time went on, the huge amount of loosened sand and gravel was washed, resorted, and removed by spring runoff, flash flooding, and normal water flow. The process of erosion was greatly accelerated by the tremendous volume of loose material.

In addition, there were large amounts of unworked gravel left exposed by past mining activity. Marginal and low-grade gravels were stripped of vegetation and exposed to the effects of increased erosion. The river and stream beds had been cleared and disturbed to such an extent that the loosened material could move downstream at a much faster rate than normal. All of these conditions contributed to the amount of placer minerals available for reconcentration in newer placer deposits.

Another important consideration is the value of other metals and minerals in placer gravels that were largely ignored by previous miners in their search for gold. The heavy placer minerals will be reconcentrated along with precious metals but to a much greater extent because none of the heavy placer minerals were removed during original mining operations. The price for a number of placer minerals has seen a significant increase over the years, and it is possible that the value of these minerals may exceed the value of precious metals contained in placer reconcentrations from these areas.

All things considered, economically feasible placer deposits formed by reconcentration of values are very likely, especially if all the valuable minerals are utilized.

CONCLUSION

This presentation was designed to give an overview of the various types of placer deposits and the conditions that regulate their formation. It was also intended to illustrate the more important characteristics of the minerals and materials contained in the deposits. Because so complex a subject could not be fully covered in the limited time and space available, the material presented is by no means complete.

REFERENCES CITED

Bateman, A.M., 1950, Economic mineral deposits: New York, John Wiley, 916 p.
Lindgren, Waldemar, 1933, Mineral deposits: New York, McGraw-Hill, 930 p.


TODAY

HYDRAULIC MINING METHODS

by
John Miscovich, Alaska Placer Miner
1093 North Green Grove
Orange, California 92667

We are going through critical times in our Alaska mining history and all of us are going to have to work together to find a solution to the problems that lie ahead. Environmental regulations are not going to melt away with the spring thaw nor will the standards for settleable solids or turbidity be modified. I am afraid we placer miners are in for some tough times ahead.

When I came to Anchorage and said that I was coming to Fairbanks to talk about hydrosluicing, some miners looked at me as if I had 'AIDS.' For the past 50 yr, I have worked on improved designs for hydraulic mining, using water delivered through the 'Miscovich Monitor' (fig. 1). This interesting world-wide adventure has involved me in mining, fire fighting, the space program, the military, and other industrial activities. I have enhanced my knowledge through engineering-design work.

Much of what I learned can be applied to Alaska placer mining. I am well aware of the water problem and know that it will require the cooperation of all miners involved to engineer a solution. I might point out that 'engineer' is the word that must apply. Miners are not going to go their own way and disregard their neighbor below, beside, or above them. There is no way the settleable solids problem will be easily resolved. Turbidity level and materials in solution are the main issue. If present standards are going to be enforced, we might as well shut our operations down now, rather than take the risk of heavy fines that can destroy the small miner. And while it looks as if Union Carbide had a wonderful setup up here with their coagulant system, let's wait and see.

I would like to add just one comment on the individual demonstration programs. The only one that is going to survive is the one presented at this meeting by Karl Haneman, mark my word on that. He has developed and is going to further develop a system that you can all use.

This paper briefly summarizes my experiences with hydrosluicing during 40 yr of placer mining in Alaska and overseas. I suggest some systems that may meet the environmental requirements for taking care of your water. I also suggest that if the State of Alaska is going to expand in the mining business, it has to use water and lots of it. If anyone tells me they can run a system with 300, 400, 500, 600 or 1,000 gal/min and call it production, they are just snipers.

My story began 40 yr ago when I developed the first automatic water cannon. That was in Flat Creek, where Pete Haggland is now mining. He took the property over from Billy Bick. Back then, George Miscovich operated the first automatic monitor ('Inteligine') in Alaska. The unit, which could also be operated manually, used two cylinders, and water from the two cylinders

Figure 1. Sluicing with the 'Miscovich Monitor,' 1975.

allowed vertical and horizontal movement. It didn't take a very astute student to know what was wrong with the traditional monitor. It didn't move. That's why I redesigned the old hydraulic monitor to move on its own water power, 24 hr a day, in a preset pattern. A vertical cylinder propelled the vertical motion and a horizontal cylinder propelled the horizontal motion. By interrelating the two valves on the two cylinders, we preset a pattern of continuous automatic motion.

I also mined on Fairbanks Creek, where I started the first shift on the '4th of July' mine, and for the first time, had an opportunity to put 60 lb pressure through a 2½-in. nozzle feeding the sluice box. Bear in mind that this is 40 yr ago. I showed some of the men exactly how to control the unit. On Flat Creek, we had the same problem of water just shooting in one place when we were not working the monitor. At today's labor rates, it would be impossible to have the number of monitors and monitor operators that the FE Company had.

The automatic monitor moves only overburden in front of it down into the bedrock 'drain.' The mine cut remains open while the liquified bedrock is in motion. The monitor works in the dark, it doesn't mind the mosquitoes, it doesn't require any lunch, and above all it never complains. The design has changed drastically because there are sediments that have to be removed to get at other valuable metals, not only gold.

Forty years ago, permafrost could be washed away for about 10 cents/yd or less. Today's cost, I would estimate, including pumping water, recycling, and settling ponds, would probably run about 30 to 40 cents/yd. The idea is to keep the warm water on the frost 24 hr a day. And that is the secret to permafrost removal, using the ice in the frost and the water to help transport it downstream to a settling pond.

It has been an interesting experience. There have been a lot of heartaches during those 40 yr but also a lot of successes, which is why I like to come back to Alaska. On Flat Creek, for example, our monitors worked against the face of the mine cut, automatically stripping overburden. We had a 17-mi-long gravity water ditch with pump boosters and a very large pit that we used as a settling pond. We were recycling water with a 6-cylinder Murphy (Gold Stream Creek has a lot of water when it rains, but when it's dry, it's dry). The overburden was about 60 to 80 ft (when miners talk about 10-ft ground, how I wished I could have been mining 10-ft ground). Our later model monitor worked over the permafrost face, wearing off the ice and mud. A little water was diverted through the cylinders to operate the horizontal and vertical motion.

Permafrost conditions are also common in parts of Canada and Russia. In those countries, three guns were often working in a pit. The monitors can be operated from the pit car with all the controls inside. There were big pumps that pumped up to 2,200 tons of slurry/hr, 24 hr/day. These pumps would be applicable to recycling systems found in permafrost areas of Alaska.

I also used my system in the Phillipines, where a river had filled up with sand from a copper mine. The river had threatened to flood the Consolidated Fertilizer Plant, where there were docks, and a village upstream of about 10,000 people. I set up the system and hydrosluiced the river channel, which saved the village and an airport. In South Africa, we used a similar system for a similar problem. That system is still being used today. There, the monitors broke down tailings from earlier mining days, then moved the tailings through channels into the processing plant. That's recycling!

In the Phillipines, we installed a caisson for tailings removal. The system didn't work with reverse air jets, so I concluded that it would never work in an Alaska placer. The 10-ft caisson had a vertical range of 60 ft, in 10-ft increments, to provide a reserve area for tailings from the mine. We cut down an embankment for a road and pumped water up to 2,200 ft from the river through three pumping stations. We were using 225-lb pressure through a 2½-in. nozzle. We installed eight of our monitors around a tailings pond, 800-ft diam, that we would repeatedly fill and empty of tailings. We put a monitor on a 300-ft catwalk that also carried our power and led the pipeline away from the caisson.

At Bouganville, we worked at the very crest of a copper deposit. It is the largest copper and gold mine in the world, and in 18 mo, we moved about 30 million tons of material at a cost of about 10 cents/ton. Boulders had to be deslimed, then picked up by mechanical equipment and moved away.

In view of today's environmental regulations, we should focus on obtaining the legal right to have our first settling stage right at the end of the sluice box, so we can wash the heavy clays common to Alaska placer mines. We will need to use about 40-lb pressure through a 5-in. nozzle. My first grizzly was designed some 15 yr ago for very high clay content in the pit.

Material was pushed up with a tractor and washed over the grizzly, which contained riffles. The material then fell through the grizzly and down into the sluice box. Since then I have made some modifications, but it is basically the same, except it has no moving parts. Today I have a front-end loader feeding the grizzly. We can move about 250 to 300 yd/hr, depending on how far we have to haul with the loader. The coarse stuff comes off the grizzly and the fine stuff goes down the sluice box. It is important to watch the washing carefully to recover all the gold attached to the rock surface.

In Kotzebue, our airport project moved millions of yards. The Russians let us use their settling pond (in the Bering Sea). In Sitka, we had to extend the airport 1,700 ft. We designed the whole plan in California and shipped it to Sitka. We used monitors to deslime gravel where we couldn't get in with a cat. What was most interesting about the Sitka project, however, was the high bank of dry muck that we stripped and drained along its edge, using the hydraulic giant. The pit where we were sluicing was 12-ft deep and we picked up the tailings with a dragline. Around the corner we had another lift that took the tailings up about 30 ft and then dumped them into the drain below.

Hydrosluicing in the 1900s commonly used six or eight cutters and three monitors to stack tailings. A dam was built to feed a large monitor that took all the soft material off the rock using 300 lb of pressure with a 2-in. nozzle. If this were properly done today, the water would be recycled. If you had bad water conditions, you could use a closed circuit system and a pump. You could work it manually or work it automatically by remote control or from a television screen.

QUESTIONS AND ANSWERS

(Q): Did you ever use a water lift for moving rock into your sluice-box operation?

(A): Yes, as long as the rock is not too heavy and the boulders not too large, you can water lift them right into your sluice box. The advantage is that you won't wear out all the parts in the sluice box with a lot of heavy rock.

(Q): Have you done some experimenting with a water lift for lifting large material?

(A): Yes, we had formerly used strong backs and wheelbarrows but we eventually developed a hydraulic lift. Any number of scalpers can be used ahead of your hydraulic lift. The best approach is to put a grizzly in front of the pick-up point of the lift and then anything that is too big will not go into the pick-up point and cause the water to reverse. You can use a hydraulic lift without any other equipment to feed right into a loader.

A MEDLEY OF AUSTRALIAN PLACER-MINING OPERATIONS

By
J.A. Madonna, Assistant Professor
School of Mineral Engineering
University of Alaska-Fairbanks
Fairbanks, Alaska 99775

INTRODUCTION

As the title indicates, this presentation is a medley of Australian placer-mining operations.[1] The three operations that will be covered include 1) a sapphire fossicking operation; 2) a small two-person sapphire and tin washing plant and jig operation; and 3) a large cassiterite (tin) placer-mining operation. Perhaps after viewing these Australian placer-mining operations we can use some of the concepts to enhance our fine-gold recovery and possibly solve some of our water-quality problems.

FOSSICKING

The first of our medley of three mining operations is fossicking for sapphire. Sapphire is one of the gem varieties of the mineral corundum, which has a hardness of nine on the Mohs scale and a specific gravity of about four. Corundum is an oxide of aluminum (Al_2O_3), which when in pure crystalline form is clear and glassy and commonly known as white sapphire. Trace elements in the crystal lattice impart colors that are important to the value of the gemstone. Iron and titanium are responsible for the blue color in sapphire, whereas iron alone usually causes yellow, green, and brown colors. The greater the percentage of trace elements present, the darker the color of the gemstone.

Although recent years have seen spectacular growth in large-scale sapphire mining, certain areas of New South Wales have been set aside for fossicking. The areas are clearly marked. Even more interesting than the fossicking sign is the rather flat topography and moderate vegetation that is typical of the stream valleys in the Glen Innes, Emmaville, and Inverell areas.

A willing group of Mining Extension students, including Kay Kletka, Allan Coty, Ted McHenry, and Leah Madonna, systematically followed through the steps of proper sapphire fossicking. First, it was necessary to select an area where sapphires were most likely to be concentrated, generally a seam of gravel sandwiched between layers of clay-rich silt. Material was dug from the gravel using a rock pick. The gravel was tightly bonded together with clay and broke loose from the exposure in resistant clumps. The material was shovelled into a 12-mesh screen nestled in a 6-mesh screen positioned over

---

[1] Information provided here was gathered during the University of Alaska, Alaska-to-Australia Mining and Milling Tour, held in January 1986.

a gold pan. Then the load was carried to one of the ponds, which generally had to be deepened for the washing process.

Clay bonding the gravels was thorougbly washed, thereby permitting the larger particles, including the sapphires, to be freed. Once the material was broken down, the clay washed away, and the fines filtered through the coarse sieve, the residue on top was examined for large, 'world-beater' sapphires. This was achieved by a screen-flipping technique most clearly described by our Australian guide, Gordon Bouveret. It is quite similar to flipping over sourdough pancakes, where the bottom is then the top. Because sapphires have a medium specific gravity of four, they are heavier than an equal volume of country rock and rock-forming minerals. Therefore, when the screen was vigorously rotated from side to side, any sapphires contained in the bed of material migrated to the bottom. Consequently, when the screen was properly flipped over, the sapphires were exposed on top and were easily spotted and plucked from the pile. We first followed this procedure with material in the coarse screen, which we had hoped would contain the large sapphires. Then, after vigorously washing the material in the finer screen, we carefully examined it using the same screen-flipping technique. Although several sapphires were found trapped on the fine screen, no large 'world beaters' were found on the coarser screen.

Because over 50 percent of a sapphire can be lost during faceting, sapphires smaller than 12 mesh (the size of a paper-match head) are not considered worthwhile. Therefore, the fine material that passes through the 12-mesh screen is generally discarded. For demonstration purposes, however, Ted McHenry panned the very fine material to show black sand that is predominantly cassiterite and is mined economically in the area. In addition to sapphires and cassiterite, zircon, spinel, and hematite were found locked away in the gravels.

Fossicking for sapphires was a fascinating and exciting experience that will long be remembered. Perhaps someone on our next Alaska-to-Australia Mining and Milling Tour will discover one of those large 'world beater' sapphires.

GORDON BOUVERET'S SAPPHIRE AND CASSITERITE WASHING PLANT AND JIG

The second in our medley of Australian placer mining operations is in Glen Innes, New South Wales, where we visited Gordon Bouveret's sapphire and cassiterite washing plant and jig.

As shown in figure 1, the washing plant consists of a hopper complete with spray bar. A gas engine on the left drives a dual shaker screen, which separates the gravel into coarse, medium, and fine fractions. A sluice box had been constructed to accept riffles in separate channels that are spaced approximately 1½ ft apart. The washing plant is mobile, so that it can be moved to the mining site or used to wash test material in Gordon's backyard.

Water for the washing plant is pumped from a holding tank, located at the end of the sluice box, up to a spray bar located on the dump hopper (fig. 1). The hopper leads to the live screen system, which is driven back

Figure 1. Gordon Bouveret and his washing plant, Glen Innes, New South Wales.

and forth by the engine. The top screen is 4 mesh and the lower is 12 mesh. Product off the screens is directed by channels that lead to independent catch buckets.

To begin processing the material, the water tank is filled and the water pump started, thereby producing a full water cycle through the machine. The material is dumped from buckets into the feed hopper, where it is reduced to a slurry by the water from the spray bar. The coarse material is washed and processed across the coarse ½-in. screen.

Finer material is separated from minus 12-mesh material by the second screen and collected in a separate bucket. The finer material is then channelled into the sluice where the larger particles settle out and the finest material is directed to the water tank.

It required about one hour to process eight buckets of material through the washing plant. The result was two buckets of coarse material and one bucket of finer minus 4-mesh plus 12-mesh material.

The final step was to run the buckets of material through a small compact 2-cell jig. Jigs come in a variety of sizes and forms that range from 1 yd/hr, two-cell units all the way up to large six-celled doublets capable of processing 50 to 75 yd/hr. Components include a water tank with water inlets appropriately positioned and outlet spigots on the bottom. Each cell has a corresponding pulsator that consists of a rubber diaphragm clamped securely to the framework. Attached to the diaphragm is a bar driven by an electric motor that moves the flexible rubber back and forth, piston style. This action forces the water filling the jig tank (hutch) to move up and down, thereby creating a pulsating effect.

Now imagine a screen mounted on a slant within the top of the water-tank framework. Bedding material is placed on the screen to trap the ore minerals. Ore-bearing gravel is fed into the receiving hopper, then it passes onto the screen. The pulsation created by the diaphragm, as it moves forward and backward, first forces the bedload of material to move upward, thereby momentarily suspending the material. The higher specific gravity material displaces the lower specific gravity material at the bottom of the gravel column. As the lighter material moves up, it is swept into the cross flow of water that passes across the jig bed and is ultimately carried off as tailings. During the backward movement, the bed of material is drawn to the screen. Coarser ore minerals that have worked down through the bedding settle at the interface of the bedding material and the screen. The finer ore minerals pass through the screen into the hutch and subsequently fall to the bottom, where they pass through a spigot and into a catchment tank.

Another important feature of a jig is the hutch water. Because a certain amount of water is lost to the cross flow of the jig cycle and also through the spigot at the bottom, it is necessary to add hutch water. Not only does the addition of hutch water replace that lost, but it also provides a constant upward force that keeps the bedding material in suspension at all times, except during the final stages of the suction stroke, which pull the bedding, for a brief moment, compactly against the screen.

In most cases, the bedding (ragging) can be any material with a specific gravity of five or more, such as steel shot, taconite pellets, or hematite. The bedding material, however, must be of a lower specific gravity than the ore mineral. Consequently, because sapphires have a specific gravity of four, the ideal bedding is natural gravel, which has a specific gravity of three and one-half or less. Gordon uses the coarser material from his concentrator to make up the bed for his sapphire jigging operation.

Figure 2 shows that Gordon's small jig is composed of two cells and a dual-purpose engine, which drives the pulsator and the water pump. Interestingly, the diaphragm is mounted on the steel framework partition separating the two cells; therefore, the one diaphragm serves both cells. Clearly, one cell will experience the suction stroke, while simultaneously the other will undergo the compression stroke. Also shown in figure 2 is the water pump, which supplies water to the hose that directs water to the feed hopper. It also supplies water to the two hoses that lead to the independent water valves on each hutch.

The two cells have a flange, welded 4 in. below the top, that supports the screens. Once the screens have been put into place and coarse gravels deposited on them, the jig is ready to start. Once started, the jig pump fills the cells with water and the diaphragm moves back and forth creating the compression and suction strokes that raise and lower the bedding material.

Once the coarse material from the washing plant is run, the finer fraction is processed. When the lighter material has been discharged out to tail, the screens are pulled, taken to a tray and flipped over. Because the sapphires are heavy they will have concentrated at the interface between the gravels and the screen. As a consequence, when the screens are flipped over and the contents fall to the tray, the sapphires are exposed on top where they can be easily picked from the pile.

Figure 2. Gordon Bouveret jigging sapphires, Glen Innes, New South Wales.

THE WIRY CREEK TIN MINE

The final step in our medley of Australian mining operations is the Wiry Creek Tin Mine near Emmaville, New South Wales.

In capsule form, the processing plant consists of a wash hopper, where ore is dumped and washed. Slurry passes into a trommel-scrubber. The coarse material falls from the trommel onto a tailings conveyor, while simultaneously the fine, minus 1-in. material collects in a tank below the trommel from which it is pumped to a three-stage jigging plant. By examining individual components of the mine in more detail, certain aspects of the operation may provide information which might prove useful to the Alaska placer miner.

Water recycle

The water from the plant is directed to a series of settling ponds that meander for about a mile before entering a small estuary. In the estuary are two 3-in. pumps used to transport water back into the washing plant. As shown in figure 3, one line from the bottom settling pond is used to supply water to a hydraulic monitor, which provides initial washing of gravels in the dump hopper, and a spray bar, which washes material as it passes over the trommel screen. The second line supplies water to the three-stage jig plant.

Ore

Cassiterite mined from the Wiry Creek drainage is disseminated as fine grains, generally 10 mesh or smaller, within the gravels. The actual mine is about a half mile from the washing plant. The ore-bearing gravels are loaded into large trucks with a back hoe, which digs to a depth of 18 to 20 ft. The material is transported to the processing plant, where it is dumped into a hopper. High pressure water issuing from a hydraulic monitor washes the material and turns it into a slurry. Once in a fluid form, the material moves down the dump hopper and through a chute to the mouth of a 30-ft-long, 6-ft-diam trommel that consists of a 24-ft blinded scrubber and a 6-ft 1-in. screen (fig. 3). After passing through the scrubber part of the trommel, the larger material that does not pass through the screen is washed once again by a spray bar to free any remaining particles of cassiterite. The material then falls to a conveyor belt that carries it to the tailings piles. Periodically, the tailings at the end of the conveyor are removed by a back hoe and stacked into tailings piles.

The fine material, along with water from the wash hopper, passes through the trommel screen where it collects in a hopper, from which it is pumped by a dual pumping system to the jig plant. Each system pumps the slurry through a 4-in. pipe to independent distributors. Each distributor splits the material between two independent dewatering cyclones, which withdraw most of the water and the minus 100-mesh, low specific gravity material, allowing the remaining cassiterite-bearing material to enter the jigging system for concentration. Fifty percent of the water is recirculated back to the trommel tank, where it is mixed with fresh water and reused in the mineral transport cycle. Overflow water from the tanks that supply the primary and tertiary jig hutches, is also returned to the trommel tank. The remaining water from the dewatering cyclones is sent to the tailings pond. Figure 4 is a view of the entire jig plant and components.

Each of the four dewatering cyclones deposits dewatered feed into a splitter box. The feed is divided and directed to two opposing rectangular jigs (one processes to the left and the other processes to the right). The primary jig system consists of two banks of four three-cell primary jigs. Each of the 24 cells is driven by an independent pulsator.

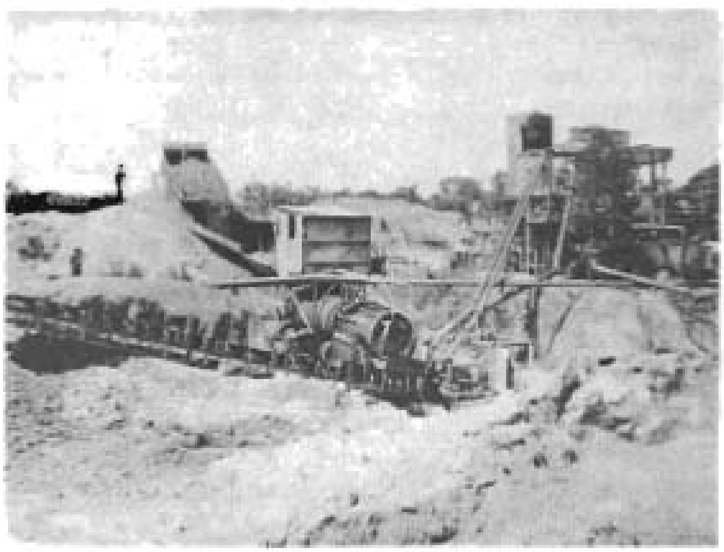


Figure 3. Wiry Creek Tin Mine, Emmaville, New South Wales.

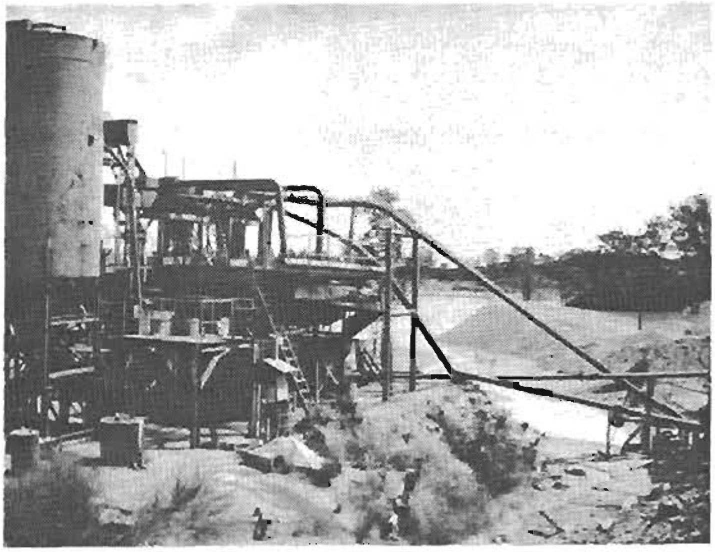


Figure 4. Jig plant at Wiry Creek Tin Mine, Emmaville, New South Wales.

The low specific gravity fraction of the minus 1-in. feed, along with a limited amount of water, works its way down the jig bed and out the tailings trough. Additional water supplied by the hutch watering system is forced up through the 3/16-in. jig screen. Independent tailings troughs from each bank of jigs direct the tailings and excess water to a tailings tank. The tailings are pumped from the tailings tank to a tailings dam. Simultaneously, high specific gravity material, including the cassiterite (specific gravity = 7), works its way down through hematite (specific gravity = 5) bedding material. The coarser ore minerals concentrate at the interface of the bedding and the 3/16-in. jig screen, while the finer material passes through the screen into the hutch and subsequently falls to the bottom, where it passes through a spigot and into a middlings tank. An important feature is the presence of a restrictor, which regulates the amount of water passing through the spigot at the bottom of the jig.

The product from the middlings tank is pumped to the secondary jig dewatering cyclone where the water is drawn off and returned to the middlings tank for reuse. The feed is deposited by the cyclone on the feed hopper where it is split across two jig beds, each consisting of two cells and driven by independent pulsators. The tailings are directed back to the primary jig; therefore, any cassiterite lost in the tails of the secondary jig is recycled through the system. As in the previous jig cycle, the high specific gravity material, including the cassiterite, works its way down through hematite bedding material. The coarser fraction settles at the interface of the bedding material and an 1/8-inch jig screen, while the finer material passes through the screen into the hutch and subsequently falls to the bottom, where it passes through a spigot and into a pipe that leads to the tertiary jig.

The tertiary jig is a two-celled jig. Each cell is driven by independent pulsators. The tailings from the tertiary jig are directed to the primary jig middlings tank, and, as a result, any values are locked into the system. Once again, the high specific gravity material, including the cassiterite, works its way down through hematite bedding material. The coarser concentrate settles at the interface of the bedding material and a 1/16-in. jig screen, while the finer material passes through the screen, into the hutch, and subsequently through a spigot and into a catch hopper (fig. 4). From the hopper it is pumped up into a holding tank, where it awaits processing in a gravity separator, and, finally, across a concentrating table.

SUMMARY

Information gathered from the three Australian placer-mining operations may serve well as guides to more efficient water use and fine-gold recovery.

In each of the operations, a limited water supply required that water be recycled. At the Wiry Creek Tin Mine, water was used efficiently by recirculating it throughout the plant and limiting discharge to tailings ponds.

Clearly, fossicking for sapphires is an exciting recreational activity similar to panning for gold. The process shows how two different minerals with a rather wide difference in specific gravity can be captured in the same mining operation but in two different areas, one on top of the screen and the other beneath the screen.

Gordon Bouveret's washing plant and jig operation demonstrates even more clearly how a machine, such as a jig, can successfully capture a mineral of specific gravity 4, such as sapphire, at the interface of natural gravels and the jig screen. The Wiry Creek tin operation demonstrates how efficiently a large-scale jig plant can capture very fine cassiterite grains with a specific gravity of only 7.

From this information it is possible to conclude that if a jig operation can capture 12-mesh and larger sapphires with a specific gravity of 4, then it would be even more efficient capturing 12-mesh or larger gold nuggets with a specific gravity of 16 to 19. Similarly, if a jigging system can efficiently capture very fine minus 12-mesh cassiterite with a specific gravity of 7, then it follows that it would be even more efficient in fine-gold recovery.

THE GYROSEPARATOR AND ITS APPLICATION ON DEADWOOD CREEK

By
Red Olson, Alaska Placer Miner
P.O. Box 199
Central, Alaska 99730

You all know about sluice boxes and the problems you have with them. They are especially troublesome at freezing temperatures. In order to reduce such problems, we bought a gyroseparator, which are circular, rotating recovery systems that use centrifugal force to concentrate gold.

To summarize our operation, the mine feed runs over a grizzly. Oversize gold is directed to a nugget trap, while coarse rock goes to waste. Material that moves through the grizzly is then passed through a trommel and is washed at high pressures to break up clumps. Oversize from the trommel goes to waste; undersize from the trommel passes to a dewatering cone. Solids from the cone are then fed to the gyroseparator.

The gyroseparator takes only about 10 min to clean a sluice box. The gold concentrate is then pumped out through a spigot. The gyroseparator seems to run better with smaller size feed, particularly in the range of < 3/8 in. It uses about 200 to 300 gal/min (at low rpm settings) of just about any kind of water. One of its greatest advantages is its ability to operate below zero, although by the first of October, when it was very cold, we put a box around the equipment.

We began testing the gyroseparator about the end of September, 1985, with the support of a grant from the State of Alaska's Placer Demonstration Project (Peterson and others, 1987). We ran four tests and processed 1,000 yd^3 of overburden and old tailings during each test. We used an existing 4- by 10-ft shaker screen and leased test units.

For the initial test, we set the speed of the gyroseparator at 18 rpm and were able to handle up to 16 yd/hr of material. Our water flow was about 800 gal/min, and our gold recovery was about 10 percent. In the mineral concentrate, there were traces of 200-mesh and a large amount of small-sized gold. After this first test, we built another variation into the system and did away with the hydroclone (a dewatering unit), which had worked fairly well, except that material didn't settle properly.

During the second test, with the speed set at 37 rpm, the feed was 34 yd/hr, water flow was 900 gal/min, and gold recovery was 15 percent. As you can see, there was a problem with gold recovery during the first two tests.

The third test was at 30 rpm, feed was 26 yd/hr, and water flow was 1,200 gal/min. We had water spillage but that was what the pipe was set for. Gold recovery was 26 percent, although that percentage was from tailings that had gone through the sluice box.

During the fourth test, the rpm was 50, material flow was 28 yd/hr, and water flow was 1,300 gal/min; gold recovery went up 50.2 percent. These latter test results indicated that we were approaching the right mixture of material and water at the correct rpm. Eventually, we may be able to raise recovery rates to that of a standard sluice box (about 80 percent).

The tests showed us what we were doing wrong. For example, we had an irregular feed from the pump to the gyroseparator of up to 1,300 gal/min. We tried a variety of arrangements to solve the 'pulsing' problem because the largest variance in performance of the gyroseparator and in overall gold recovery was caused by the rpm variations. The solid pump that was used to feed the hydroclone worked pretty well, except that the discharge from the hydroclone created so much turbulence that we could not keep water from going over the top of the gyroseparator. We also learned that we needed a large screen with a capacity of at least 100 yd/hr to handle the large volume of material. Occasionally, we had insufficient material to feed the gyroseparator.

This winter we are working outside on a laboratory grizzly and nugget-trap combination. With this equipment, we will be able to catch all the coarse gold so that we won't have to worry about any getting away. We will use a high-pressure nozzle to break down clay balls. We won't use a funnel at the gyroseparator unit, it will be just a screen plant. To feed the gyroseparator, we will have to dewater the paydirt from the trommel and then add the right amount of mix-up water.

REFERENCE CITED

Peterson, L.A., Hanneman, K.L., and Tsigonis, R.C., 1987, Placer Mining Demonstration Grant Project final report: Juneau, unpublished report for State of Alaska Department of Environmental Conservation, 113 p.

FINE-GOLD RECOVERY USING THE VENTURI EFFECT

By
Karl Hanneman, Alaska Placer-mining Engineer
P.O. Box 81416
Fairbanks, Alaska 99708

While mining on the Livengood bench during 1984-85, we modified our plant and used a hydraulic lift. Quite unexpectedly we demonstrated that this lift was responsible for reducing our gold losses in the sluice box from about 25 percent to < 1 percent. The hydraulic lift is not new to us, it has been used since the mid-1800s in California and since the early 1900s in interior Alaska. John Miscovich still uses a hydraulic lift in the Iditarod district. In this report, I'd like to show some data that will illustrate what a remarkable effect the hydraulic lift can have on sluice-box recovery.

Our operation is on the Livengood bench (fig. 1) in the Livengood-Tolovana mining district. We moved onto this property in the fall of 1984 and set up operations in mid-September. In our 1984 plant, we used a grizzly with a rod hopper on the top and a 1½-in.-mesh screen attached to the vibrating unit underneath. The minus 1½-in. residue fell through the screen and came down the box. During a couple weeks of operation, we had the chance to systematically sample our tailings and we were quite dismayed at the losses we noted. We took our samples from a loader right at the end of the sluice box. We also checked the whole sluicing system on both inside channels of the box by shoveling samples through a smaller more efficient gold saver. We kept getting consistent losses (under 30-percent recovery value), sometimes in excess of 0.005 oz/yd.

We collected samples of minus 3/8-in. material from the gold saver in tubs and carried them back to town, where we ran a pilot test on the Mark 7 spirals. The spiral test showed that 98 percent of the gold was contained in the tailings, instead of in the sluice box channels, so the problem for us was one of substantial gold losses. The pilot test indicated that the spirals would be effective in recovering our gold.

The shape factor for our gold (tested by Dan Walsh at the Mineral Industry Research Laboratory, UAF) ranged from a Corey 0.12 to 0.4 and averaged a Corey 0.26. Other Interior properties showed the following Corey shape-factor values: Gold Dust Creek, 0.39; Tobin Creek in Chandalar, 0.39; Chicken Creek, 0.68; and Yakutat Beach, 0.09. The higher the shape factor, the easier it is to catch the gold in a standard sluice box.

The reason we chose a spiral pump is because it cost only about $8,000, less than half of any other alternative, and because the pilot test demonstrated that we could use it effectively to recover the gold. We were committed to the process after ordering all the equipment and accepting an Alaska Placer Demonstration Project grant to finance it.

Our main objective in revising the pump was to substantially improve the wash. When we went to the end of the sluice box and took a sample by hand or

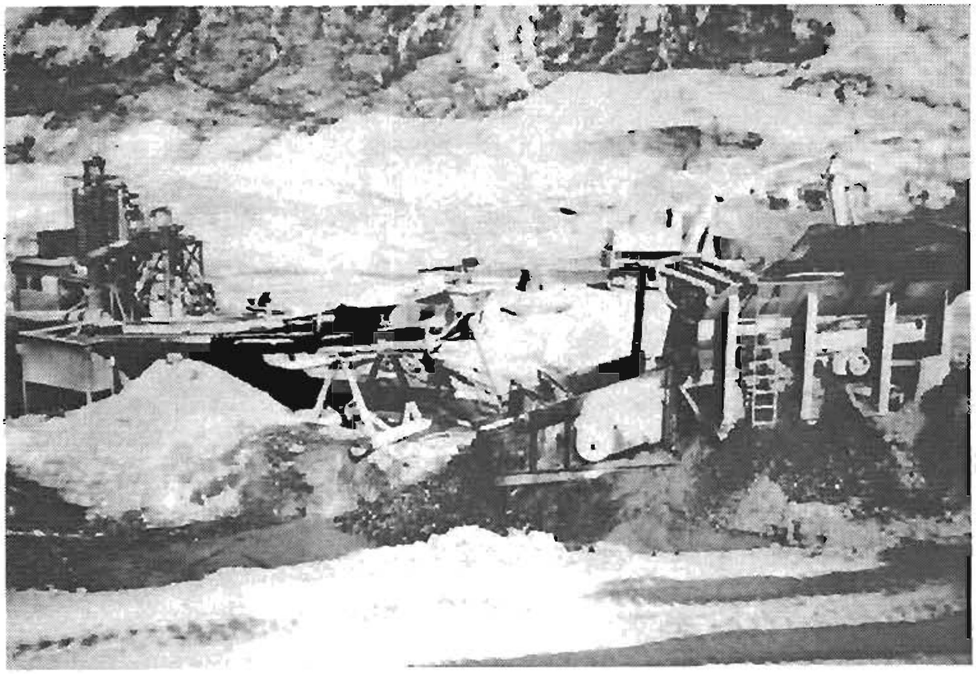


Figure 1. Washing plant, Livengood bench.

shovel, we saw that some of the clay was still stuck to the rocks and other coarse debris in the pay dirt. It is pretty tough washing 'dirt,' so we knew we had to improve the wash. At that time, we thought the spiral plant would recover the gold. I will run through the flowsheet of the spiral plant for you because we operated it for 207 hr and got a good handle on its capabilities.

There is a stationary screen at the end of the sluice box, which is 8 ft^2. The minus 10 mesh fell through the screen, where it was pumped by an 8 by 6 'Pecos' rubber-lined slurry pump. The total flow in the system to the sluice box was about 2,200 gal/min. About 200 gal fell through the screen and the remaining 2,000 gal met with the slurry going to the pump. The 'Pecos' pump moved water through a 9-inch slide to a distributor pot. The distributor pot led into two 20-in. cyclones. The cyclone underflow flowed down into a 4-in. solid pump that pushed the slurry at 26-percent solids up to the top of the spirals. The slurry ran down the spirals and the concentrate moved to the inside and was pulled off. Concentrate from the 'Triple Start' spiral flowed into the 4-in. pump. Water was added and that concentrate was pumped back up to the single spiral for upgrading. This concentrate, in turn, formed a slight bank of heavy minerals on the inside track of the single spiral. It came off the spiral onto a 'Gemini' shaking table. When this was run continuously, we ended up with about a cup or two per day of fine-gold concentrate.

On June 15, we fired the plant up and had a couple of problems with the shaft sizes on our pumps. We had to order parts from Seattle and wait a couple of days for their arrival. After that, it was full steam ahead, except that I was tearing my hair out because there wasn't any gold. I started at the bottom of the flowsheet and kept working my way back up. We would take the loader and get tailings out of the overflow. Then I would take some gold, run it over and dump it in the system. In 2½ min, it would come out on the table. So I would do it again, and it would do it again. Then we started sampling material at the end of the sluice box and there wasn't any gold.

The solution to our problem was a device that we put under our two-deck screen. This was a hydraulic lift, a cast-iron unit that weighs about 600 lbs, manufactured by the Pecos Pump Company in Georgia. The hydraulic lift injects water through a high pressure nozzle (fig. 2). This high-velocity jet creates a suction that pulls the gravel from the slurry that is underneath the screen. The suction combines all fractions and increases the slurry velocity by creating a venturi effect (fig. 3).

Ore falls into the hopper and water carries it to the lift. That creates a suction that pulls in the solids and pumps them up to the head of the sluice. There are no moving parts in this whole unit. All you need is hard facing on the pipes from your sump. The injection water flows at 750 gal/min through a 2-in. pipe at 0.67 psi. The oversize comes across the screen. The undersize falls across the hopper. This is where you apply all the high pressure water to the lift and it just beats the living daylights out of the sediment. It breaks up all the clay and brings the gold in for recovery.

When we used the hydraulic lift, our sluice box demonstrated > 99-percent recovery. The box has three channel blocks: the side channels are each 3-ft wide, and the center channel is 2-ft wide. John Burgeland, who built the box, installed baffles on the sides and down the center. The head end of the sluice has a 3/4-in. punch plate with 3/4-in. holes and a 51-percent open area. The step punch plate in the center covers 23 ft^2 at a 3 in./ft grade. All the underflow from all the punch plates reports to the side channels. The underflow doesn't go up the center because the baffles hold the water back, flooding the punch plate, and allow only fines and gold and limited amounts of water to move to the sides of the box. We get < 2-percent recovery in the center channel.

In summary, we recovered 2.78 oz of gold during the 207 hr that the spiral plant was in operation. From a salt test, we determined that the plant was recovering 98 percent of the gold; therefore, only a small amount of gold was getting to the spiral. During another test, we took the lower riffles out of the sluice box and ended up recovering 8 percent of the gold through the spiral plant, which is about the same amount as we normally recovered in the lower 16 ft of the sluice box. This indicates that the spiral plant and the screening were a valid test of the gold content of the tailings.

The effectiveness of the lift was shown in the difference in size classification of gold going out the end of the sluice box in 1985 vs. 1984. In 1984, the tailings that were going out of the end of the box were very similar to what we were recovering in the box. The material had a distribution of about 5-percent minus 100 mesh. But in 1985, with the hydraulic lift in place, the size distribution of the gold going out the box was much finer, approximately 30-percent minus 100 mesh. Our tests show that the gold that was lost in 1984 was going out the end on sediment stuck to the rock. In

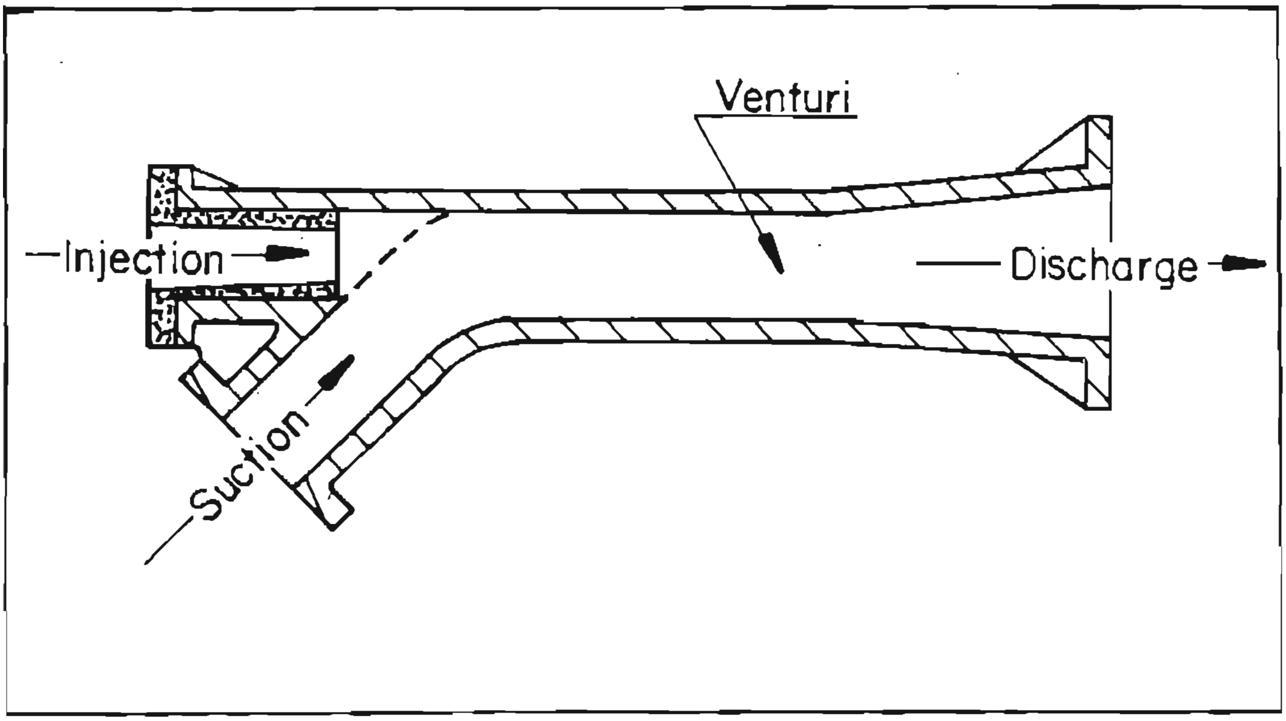


Figure 2. Principle of the hydraulic lift.

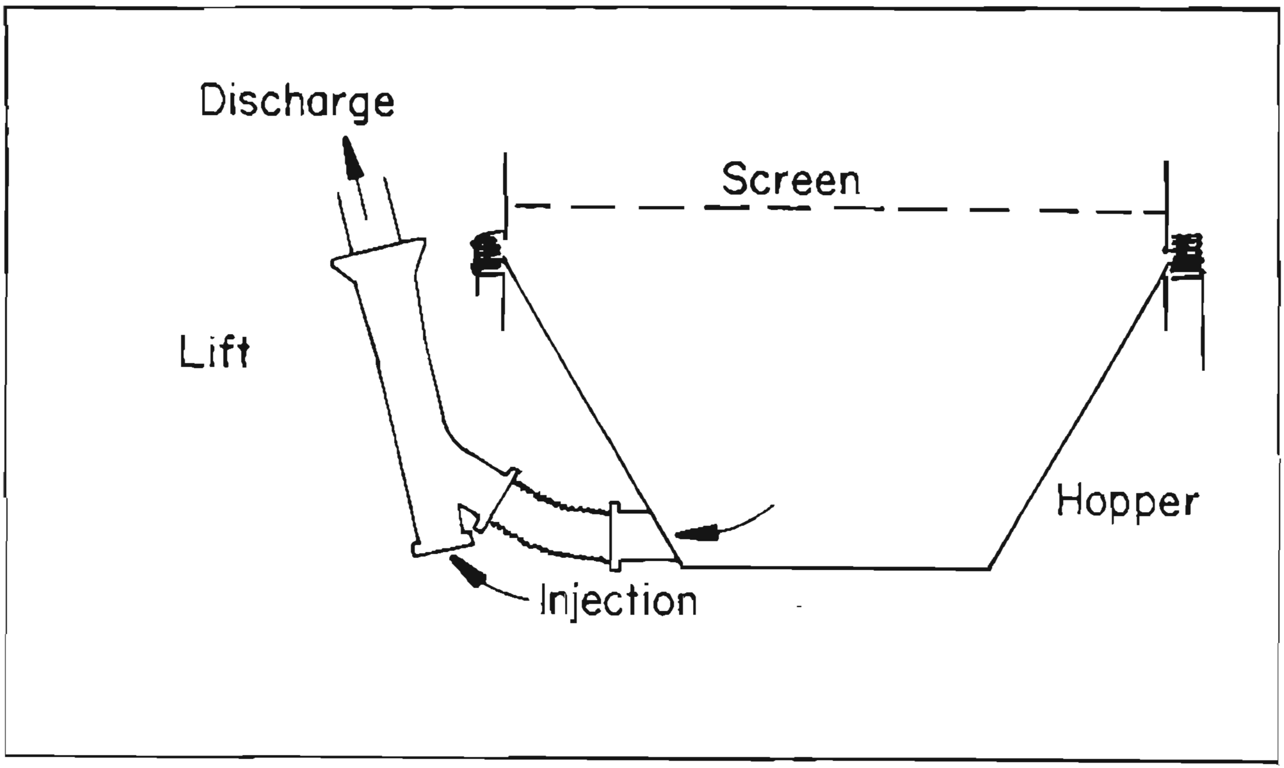


Figure 3. Hydraulic lift, Livengood bench.

1985, with the lift, the amount of gold that was going out the end of the sluice box was probably limited by the efficiency of the sluice box.

How might this apply to other operations? It worked well for us. The first step is to measure the losses in your existing system. You need to take large bulk samples from your existing system and run them through some process and see if you get gold. You can't catch it, if it's not there. Second, look at your recovery system. If you have poor classification problems and know that you are losing gold, there is a high probability that your problem is incomplete washing and the hydraulic lift may help. The capital cost of our hydraulic lift was about $2,200 FOB to Fairbanks.

QUESTIONS AND ANSWERS

(Q): Is there a riffle-placement or riffle-packing difference between the year you used the lift and the year you didn't?

(A): Yes, there was a perceptible difference in the riffle packing in the center channel. Adjusting the hole size in the trommel solved this problem. We still had packing, but it was different.

(Q): How often did you clean your riffles?

(A): Every 5,000 yd^3.

(Q): When you had 700 to 800 gal/min going in, what was coming out?

(A): Screen-wash water totalled 1,000 gal/min, injection water going to the lift totalled 700 gal/min, and water flowing to the manifold on the sluice totalled 450 gal/min. In all, 2,150 gal/min was coming out of the system.

NINE MILE PLACER, MONTANA

By
J.C. Lawrence, President
U.S. Antimony Corporation
P.O. Box 643
Thompson Falls, Montana 59873
and
P.R. Taylor, Professor
School of Metallurgical Engineering
University of Idaho
Moscow, Idaho 83843

INTRODUCTION

The Nine Mile Placer, operated by the United States Antimony Corporation, is located in the Nine Mile drainage, 25 mi west of Missoula, Montana. This paper gives an overview of the mining operation, including geology, stripping, mining, scrubbing, washing, recovery, cleanup, reclamation, and plant operation. The recovery flow sheet for the mine is shown in figure 1 and the cleanup sheet is shown in figure 2.

GEOLOGY

Local bedrock is Precambrian argillite with gold-bearing quartz veins. It occurs frequently as cobbles in the placer and is probably the source of the gold. The grade of the deposit has varied, but it has averaged more than $4/yd. Rocks in the deposit range up to 2.5-ft diam, but most of the material is much finer grained. Poor sorting, high clay content, striations, and local moraines and eskers indicate that the host sediment is glacial.

STRIPPING

Topsoil and barren material are stripped by dozers and scrapers to depths of up to 25 ft and are then piled separately to one side or the other of the active mining trench. Stripping costs vary from $0.07 to $0.03/yd, depending on the depth. After mining, the stripped waste is fed back into the trench along with plus 7/8-in. material from the trommel stacker and the minus 1/8-in. material from the settling pond; topsoil is saved for contouring.

MINING

First, a test is conducted on the upper section of the proposed mining site. Depending on the results, a mining trench is prepared, which is about 50- to 100-ft wide by 300-ft long. Dozers and scrapers---cable operated to minimize oil spills---are used to strip and mine. Ore is fed by dozer into an apron feeder and conveyor system, which supplies the plant. An operator watches the system and can stop it to remove large rocks. Water is used to remove material from the bottom of the apron feeder and is then sent back to a pond by a trough system. In the past, a dragline was used to feed the

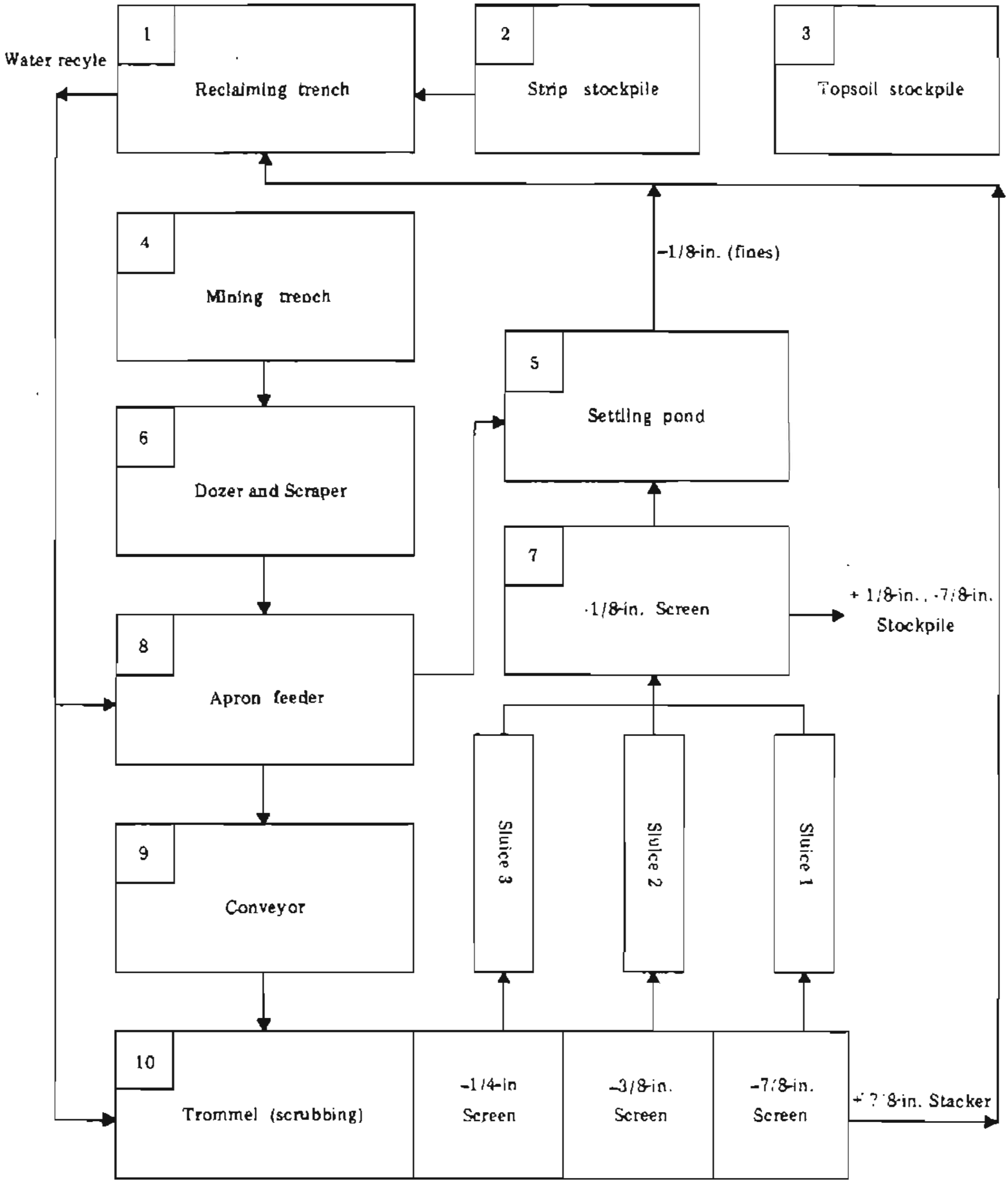


Figure 1. Mining and recovery flow sheet, Nine Mile Placer Mine, Missoula, Montana.

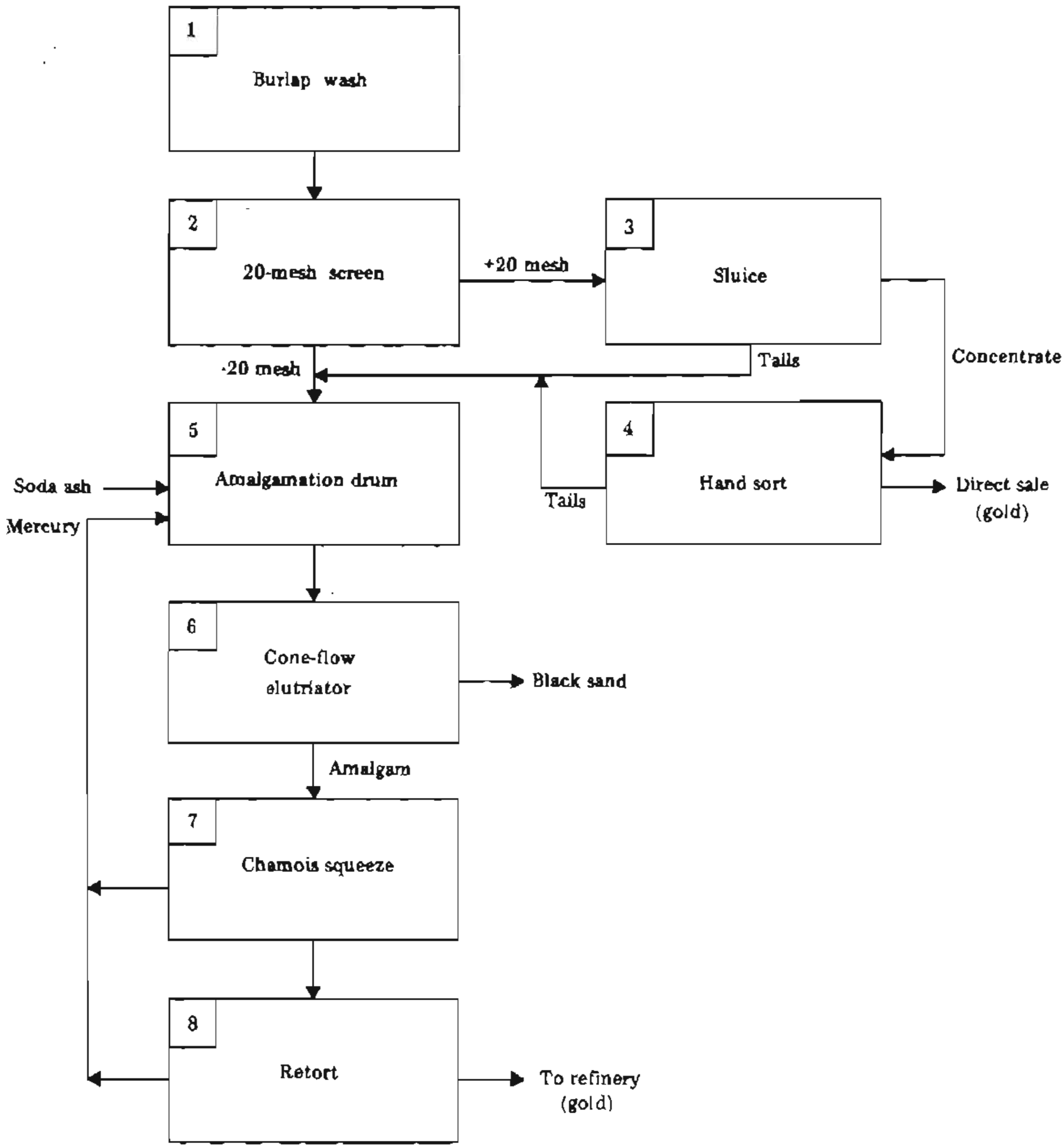


Figure 2. Cleanup flow sheet, Nine Mile Placer Mine, Missoula, Montana.

plant, but it was replaced because it did not provide a steady feed and it had a tendency to surge the plant. A grizzly was also used for awhile, but it often became blocked.

SCRUBBING

The first 15 ft of the trommel, which is 6-ft diam and 27-ft long, frees the gold from clay agglomerates by scrubbing. The scrubbing section is partitioned off by an 8-in. ring and contains sections of rail that provide lifters. Feed rate to the trommel is 40 yd/hr of ore and 800 to 1,100 gal/min of water (miners have been known to use too much water, which washes the fine gold from the riffles). The trommel rotates at 11 rpm and is driven by a 40-hp gear-head motor.

WASHING

The trommel has three separate screening sections. The first is 4-ft long with ½-in. holes punched on 3/8-in. centers. This leaves about 50-percent open screen area. Material that passes through the screen is fed directly into a sluice that is 22-ft long, 4-ft wide, and 6-in. high. The second screening section has 3/8-in. holes across a 4-ft length. Material that passes through this screen goes directly into a sluice with the same dimensions as the one fed by the first screen. The last screening section is 2-ft long with 7/8-in. holes. Discharge from this screen goes to a sluice that is 22-ft long, 2-ft wide, and 6-in. high. A common mistake is to use oversized holes (1 to 2 in.), which allow coarse rock to blind the riffles. To remove the oversized rock, water volume is increased, and the fine gold is washed out of the riffles. Water to each screening section is adjusted through a spray bar. Oversize material (plus 7/8 in.) is directed to a pile where it is removed by dozer and scraper and replaced in the reclamation area.

RECOVERY

The sluice boxes are lined with one layer of open-weave burlap and covered with expanded metal riffles, which are diamond-shaped and about ½-in. high by 6-in. diam; the box grades about 1.75 in./ft. Every 8 hr, the system is stopped for cleanup, which takes ½ hr. The riffles are removed, the burlap collected, and the heavy minerals washed into a tray. Fine material that passes over the riffles is sized to 1/8 in. The plus 1/8-in. material is stockpiled for sale as roadbed aggregate; the minus 1/8-in. material is pumped back to the reclamation trench; and the water is recycled back to the trommel. 'American Colloid Percol 65' has been used as a flocculant.

Jigs were tried for a while, but they recovered so much black sand that it was difficult to amalgamate the gold grains, and the black sand could not be reconcentrated on a table. In addition, there was a problem of environmental compliance relative to the use of mercury on large amounts of heavy materials.

CLEANUP

The burlap is washed free of minerals, and the collected concentrate is screened at 20 mesh. The plus 20-mesh material is reconcentrated in a sluice, and the gold is sorted by hand for direct sale. The residue is combined with the minus 20-mesh material and is amalgamated by batch in a drum. Soda ash is used to prevent flouring. Direct operating costs for the ashing are about \$1.35/yd. Amalgam is separated using a cone-flow elutriator, then squeezed through a chamois and retorted. The recovered gold is then sent to a refinery.

RECLAMATION

The mined trenches are reclaimed during operation by pumping back fines from the trommel, dewatering that material (the water is recycled), and then blending in both the coarse material rejected from the trommel and the waste material from stripping. After the mining season is over, the ground is contoured to its original slope, the topsoil is replaced, and duck ponds are constructed. The area is then planted with a seed mixture (20 lb/acre), designated by the U.S. Forest Service, and fertilized (50 lb/acre of PKZ).

PLANT OPERATION

A moving plant was used originally, but it has been replaced by a fixed plant because this eliminates much of the mud and the need to continually build ponds. From an environmental standpoint, a fixed plant is better because it eliminates drainage from the ponds to the creek. The plant is in operation from February (or April) through the end of November, 6 days/wk. It is run by a crew of five men that cover two 8-hr shifts. Two men are employed year-round and spend the winter reclaiming the land, maintaining the equipment, and guarding the property.

To summarize, the following is a list of equipment used at the Nine Mile Placer Mine:

3 'DW21 Caterpillar' 15-yd^3 scrapers
1 'Manitowac 3500' 2-yd^3 dragline
1 'D9 Caterpillar' tractor
1 'D8 Caterpillar' tractor
1 '8240 Terex' dozer
1 8-V '92-series GMC' diesel generator (300 kw)
1 8-in. by 10-in. 'Galigher' pump (100 hp-electric)
1 8-in. by 6-in. 'Galigher' pump (100 hp-electric)
1 6-ft-diam by 27-ft-long trommel
1 Apron feeder
Conveyors
3 Sluice boxes

UNDERGROUND PLACER MINING IN SIBERIA

By
F.J. Skudrzyk, Professor
School of Mineral Engineering
University of Alaska-Fairbanks
Fairbanks, Alaska 99775
and
J.C. Barker, Supervisor
U.S. Bureau of Mines
206 O'Neil Building
University of Alaska-Fairbanks
Fairbanks, Alaska 99775

Mention underground placer mining, drift mining as it is more commonly known, to most active placer miners and an image comes to mind of one or two die-hard gold miners tending their leaky steam lines at the bottom of a rickety log-cribbed shaft about 50 yr ago. Sophistication then may have been a wheelbarrow and a cap lamp. Until the 1930s, this was a common winter occupation for miners who hydraulically worked or sluiced shallower creek gravels in the summer. The purpose of our presentation today is to describe the underground placer-mining technology that has developed since the days of those old-time gold miners. Most of the discussion is derived from translations of work presented in recent Soviet technical journals and does not reflect any investigation performed by the Bureau of Mines or the University of Alaska.

Drift mining probably got its start around 1870 in the semiconsolidated Tertiary gravels in California. Then, in 1898, the Klondike was discovered. In the far north, placer deposits usually occur in areas that have escaped Quaternary glaciation. Typically, however, these areas are mantled by thick accumulations of wind-blown loess that must be removed to reach the auriferous gravels. Commonly, this overburden is frozen, and therefore, must be thawed before stripping. Shortly after the gold strikes in the Yukon, in Alaska, and later in Siberia, miners realized that an alternative existed for mining permanently frozen ground. Rather than thaw and remove massive amounts of valueless overburden, it was easier to mine only the rich pay gravels at or near bedrock, usually no more than 4- to 8-ft thick, and leave the overburden in place. It was indeed an attractive alternative in those days before the advent of the bulldozer and front-end loader.

Subsequently, many of the creeks in the Fairbanks area, even those which were later dredged, were mined by drift miners who worked with pick and shovel in narrow, dangerous, poorly ventilated tunnels. In Siberia, the death camp organization 'Dal strio' evolved, and in the 1930s, millions of political prisoners were sentenced to the gold-placer mines. In a recent account by Cieslewicz (1987), the average prisoner extracted 1 to 2 kg of gold before his death.

Following World War II, underground placer mining in Alaska became somewhat of a lost art. Meanwhile, in Siberia, fragmentation drilling and blasting came into use (Lubey, 1978; Sherstov and others, 1981; and Emelanov and others, 1982). Rudimentary drill jumbos, coupled with scrapers and vibrating conveyors, increased productivity by as much as 4½ times. By the 1960s, higher performance jumbo drills with hydraulic booms were installed in some mines, as well as belt conveyors and larger capacity scrapers (up to 2 yd^3). The death-camp labor force was abandoned and workers were hired for the increasingly mechanized mines. The room-and-pillar system was further modified for frozen placers with a reported improvement in efficiency and safety.

Since the mid-1970s, with the increasing cost of labor in remote areas of Siberia, further technology has attempted to meet the demand for greater productivity while decreasing labor intensity. Rubber-tired diesel equipment---for example, front-end loaders---has been introduced and electric rotary drills are replacing the older pneumatic type. A system of artificially frozen pillars has been developed, and research is now concentrating on controlling ground movement, improved mine design, and an optimum lay-out for the mine openings. In a mining journal that detailed underground placer mining, Lubey (1978) described longwall techniques that can extract material from unsupported roof areas of 20,000 to 35,000 ft^2 or roughly an area 100-ft wide by 200- to 350-ft long. For relatively shallow deposits not exceeding 100 ft, the ore extraction efficiency generally averages 88 to 92 percent; for deep deposits, defined as those exceeding 300 ft, extraction efficiency ranges from 75 to 80 percent. Under optimum conditions, daily mine output from underground operations is 100 to 1,000 yd^3 or more. Production rates have been maximized with labor organized into 3 shifts/day with a 1- to 2-hr break between shifts for blasting and ventilation.

What many people don't realize is the importance the Soviets have placed on gold mining in eastern and northeastern Siberia. Presently, the Soviets have organized 10 mining districts that together produce 110 to 135 tons of gold annually (Cieslewicz, 1987). A recent account by V.V. Strishkov (1981) lists 35 placer-gold mines, 23 dredges, and 500 gold-washing plants in the Magadan area alone. In addition, a 12-megawatt nuclear power plant has been installed to support the mines. Alluvial tin and tungsten mining also are developed in the area.

Gold mining is not only a major economic factor in the remote territories, but it is also an important source of foreign exchange for the Soviet Union. In 1982, the U.S.S.R. produced 8.5 million troy oz of the metal, second only in production to South Africa. That is roughly equivalent to the entire historic production from the Fairbanks district. Much, perhaps two-thirds, of Soviet annual gold production comes from the eastern and the far northeastern parts of Siberia (fig. 1), and much of that production is recovered from placers.

The gold producing regions of Siberia experience a cold, semiarid, continental climate with a temperature range even more severe than what we experience in interior Alaska. Below freezing temperatures persist for 8 to 10 mo of the year as they do here, but average daily temperatures in midwinter are much lower. Permafrost in Siberia can extend to 600 ft or more, depending on the latitude. Ground temperatures of shallow placers in

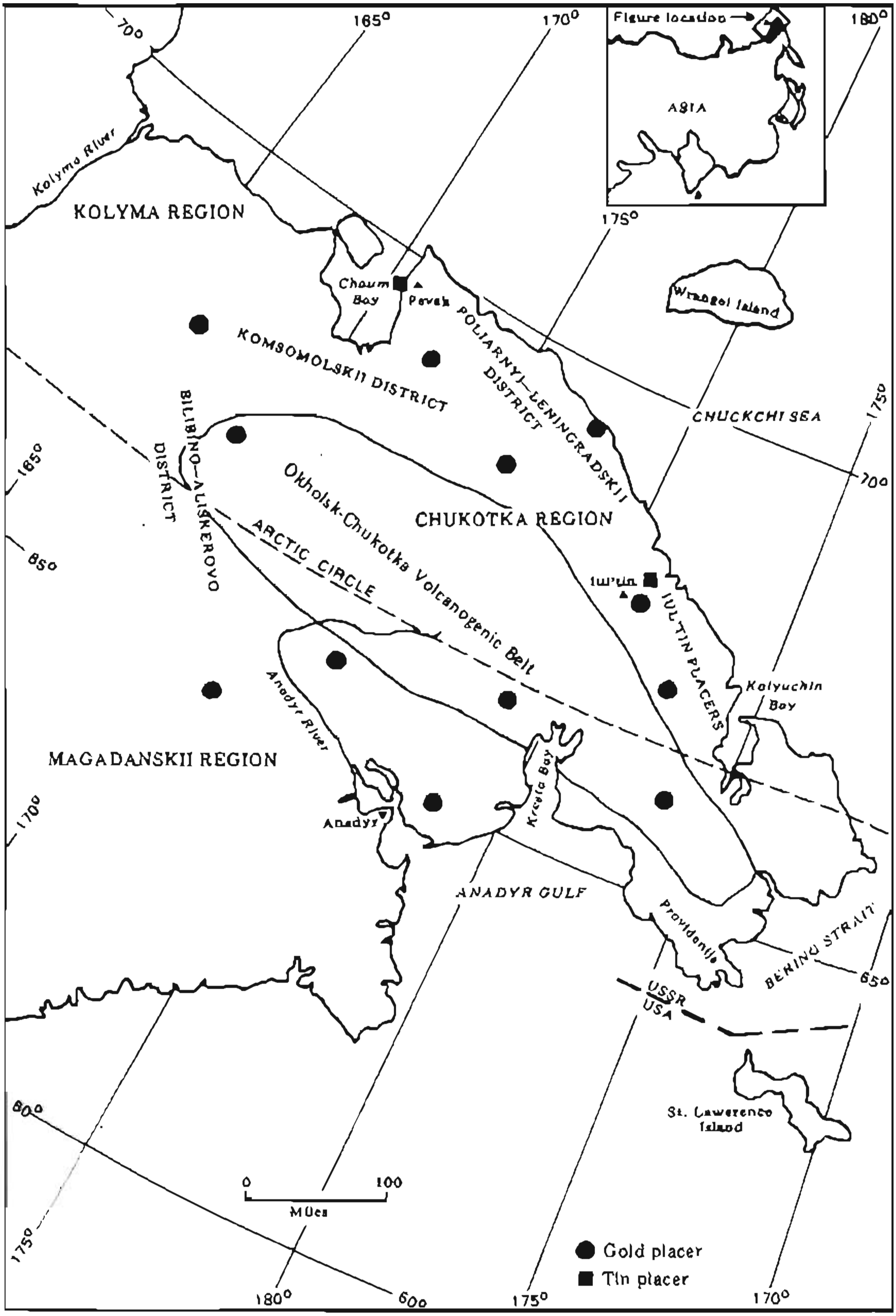


Figure 1. Principle gold- and tin-placer regions of far east Siberia.

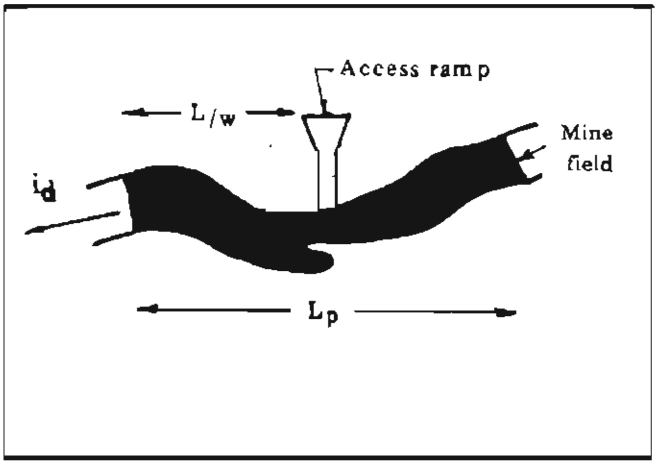
the Soviet Union, which contain most of their mineral reserves, range from 20 to 27°F; temperatures are warmer at greater depths. The best conditions for underground mining in Siberia are from October to May when freezing weather sustains the physiomechanical properties of the placers. Owing to the slightly colder temperatures of permafrost in Siberia as compared to Alaska, there is a corresponding increase in ground stability in Siberia. Temperatures of permafrost in the Fairbanks area vary from 28 to 31°F. In the Wiseman area, we have heard of 25 to 28°F reported for drift mines.

To examine the present engineering practices in the underground Siberian mines, we'll first review development access, then development of the mine in general. Together, these steps account for about 30 percent of the overall operation.

Placer deposits are notoriously discontinuous and sinuous, which necessitates a mine and access design tailored for each site. The favored mode of access to underground workings is by inclined ramp. Besides being easier for excavation than vertical shafts, ramps allow the use of continuous conveyors or rail-mounted skips. Ramps are used even for deep placers of 300 ft or more. Wherever possible, they are located in frozen ground just outside the paystreak, eliminating the need to leave a substantial foot pillar of pay gravel, but positioned to avoid undue haulage distance to some part of the proposed mine. Location of the ramp relative to the dip length of the deposit is determined by methods of Sherstov and others (1981) (fig. 2). Sometimes, particularly in lower grade deposits, ramps can be located within the placer to decrease the depth to the deposit, such as in a V-shaped valley, or to shorten haulage distance. Location of the ramp relative to processing plants and surface access may also be considerations.

The volume of a foot pillar needed to sustain an inclined ramp is computed in figure 3. Experience in Siberia has shown that the maximum angle of inclination when belt conveyors are used cannot exceed 16°; however, for rail-mounted skips this angle may be as much as 27°. Our experience at the CRREL tunnel in Fox, Alaska, has shown that a belt conveyor can move fragmented gravel at an angle of 20°. As expected, the economics of installation favor skips over belt conveyors for smaller, shallower operations; however, as depths increase so do operating costs (fig. 4).

The generally preferred method of mining frozen placers is by retreat longwall techniques. Initially, a loading station and storage chamber are excavated at the base of the ramp (fig. 5). The conveyor must have a loading station of at least 300 m^3 and storage capacity for 120 m^3 of pay gravel. Development then proceeds with construction of the main haulage drifts, ventilation drifts, cross-cut entries to the longwall panels, and ventilation entries. Options include single or multiple panel, one-sided or two-sided mining fields (fig. 6). Again, design must be site-specific and can be optimized by computer modeling. Examples of design efficiency curves for the longwall layouts are shown in figure 7. These curves plot length vs. width of a proposed mining field and can predict an optimum rate of return for a design plan once the variables are identified. Note that, where practical, the multipanel, two-sided scheme is generally most efficient, although access



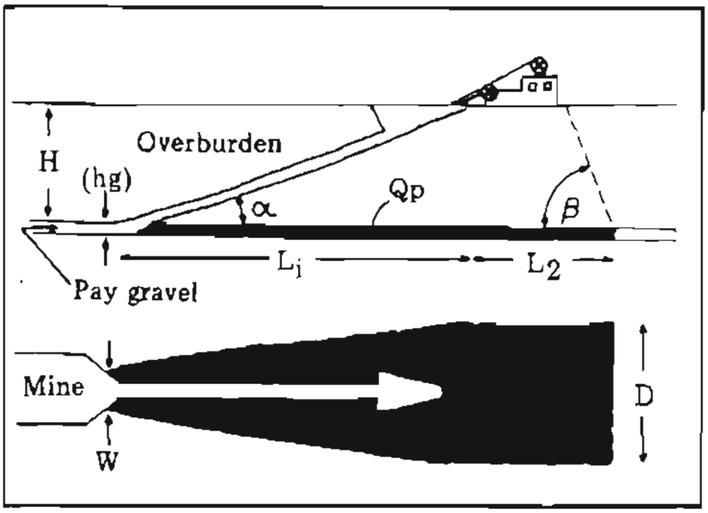
$L_{/w}$ = Mine-field length down dip of ramp.

L_p = Total length of mine field.

i_d = Percent dip of deposit.

$$L_{/w}(m) = 0.5(1-0.77 i_d) L_p$$

Figure 2. Plan view of proposed mine field in Siberia. Adapted from Sherstov and others (1981).



Horizontal length of inclined ramp:

$$L_i = (H+hg)\text{cotan}\,\alpha$$

Maximum width of foot pillar:

$$D = W + 2[(H+hg)\text{cotan}\,\beta]$$

Volume of gravel in foot pillar:

$$Q_p = hg[L_i(\frac{W+D}{2}) + L_2 D]$$

Figure 3. Length of inclined ramp, width of foot pillar, and volume of gravel in foot pillar for underground mine in Siberia. Adapted from Emelanov and others (1982).

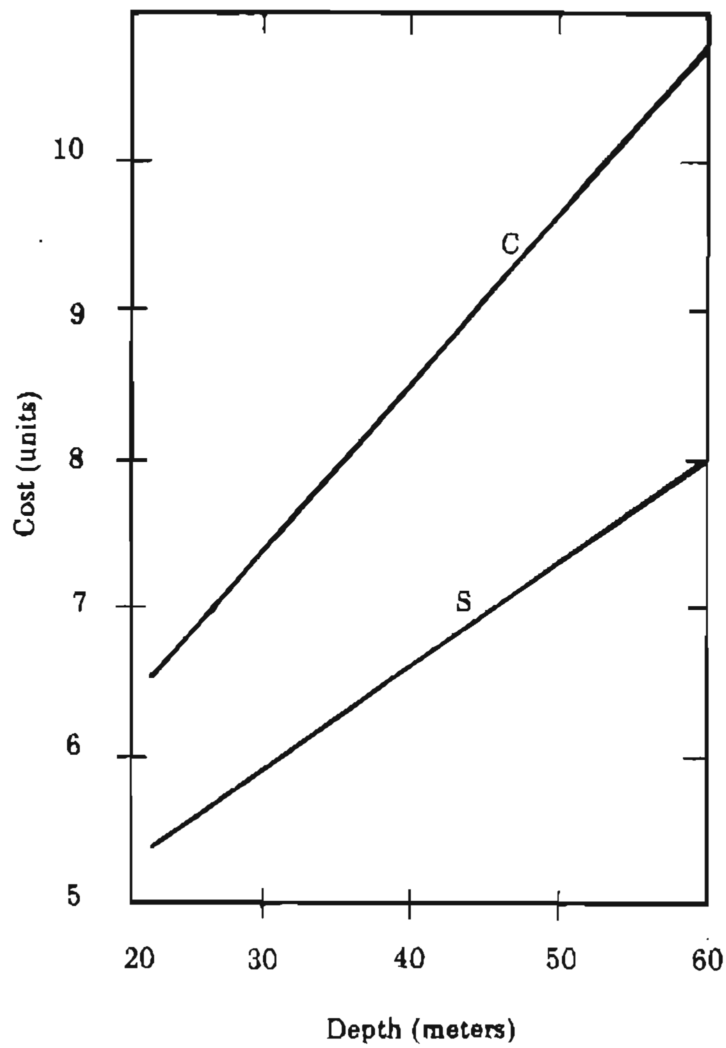
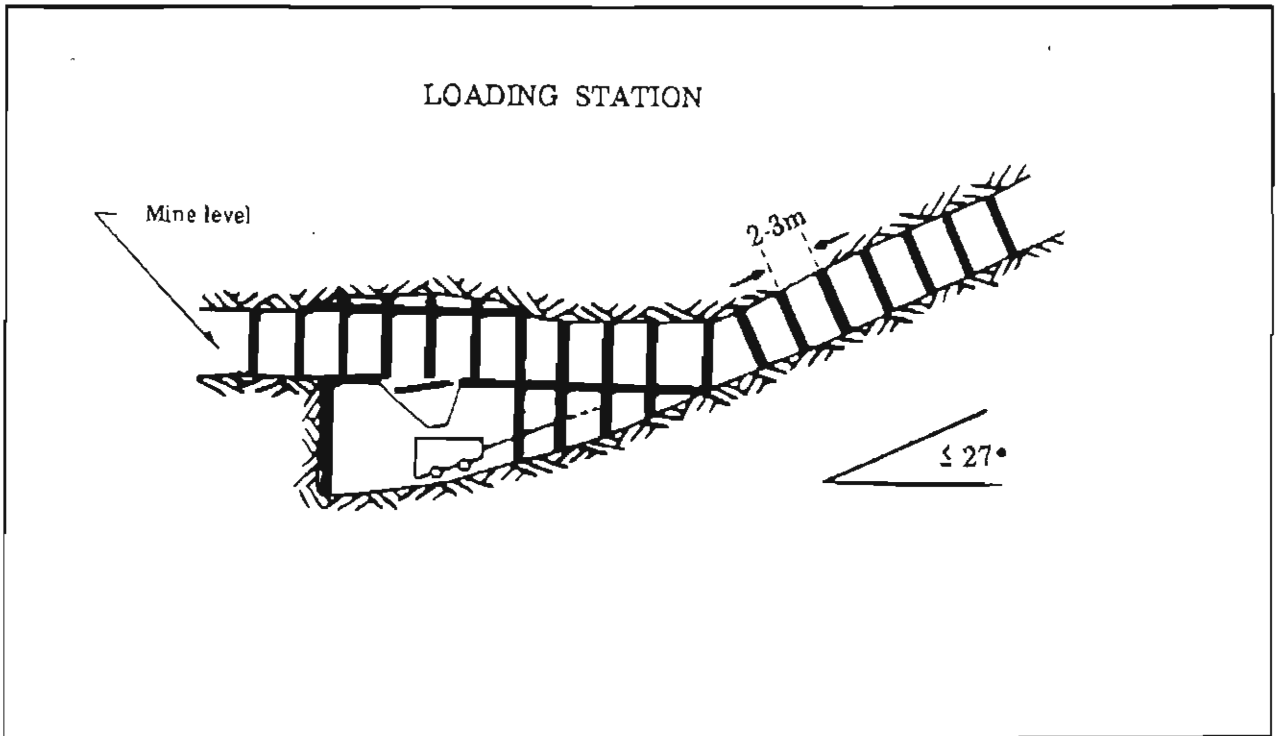


Figure 4. Cost of skip (S) vs. conveyor (C).

within the deposit is required. One-sided schemes are preferable if the pay gravel is of exceptional grade because foot pillar losses are minimized.

Production drilling and blasting have been based on standard air percussion drills; however, recent research has been directed toward improving production-drilling efficiency. In 1981, new electric-powered rotary drill jumbos were assigned to some longwall faces. These drills achieved penetration rates of about 2 ft/min for 1.6-in.-diam holes. Blasting is done with conventional dynamite; powder requirements are a function of face size and shape, ground temperature, ice content, and other factors that determine compressive and tensile strengths. Following the blast, gathering-arm loading machines feed stockpiles in the access drifts, which are then moved to the surface by scrapers. Maximum haulage distance for the scrapers is 150 to 200 ft. In some mines where headroom is sufficient, LHDs (Load Haul Dumps) and more recently adopted front-end loaders are alternatively employed.



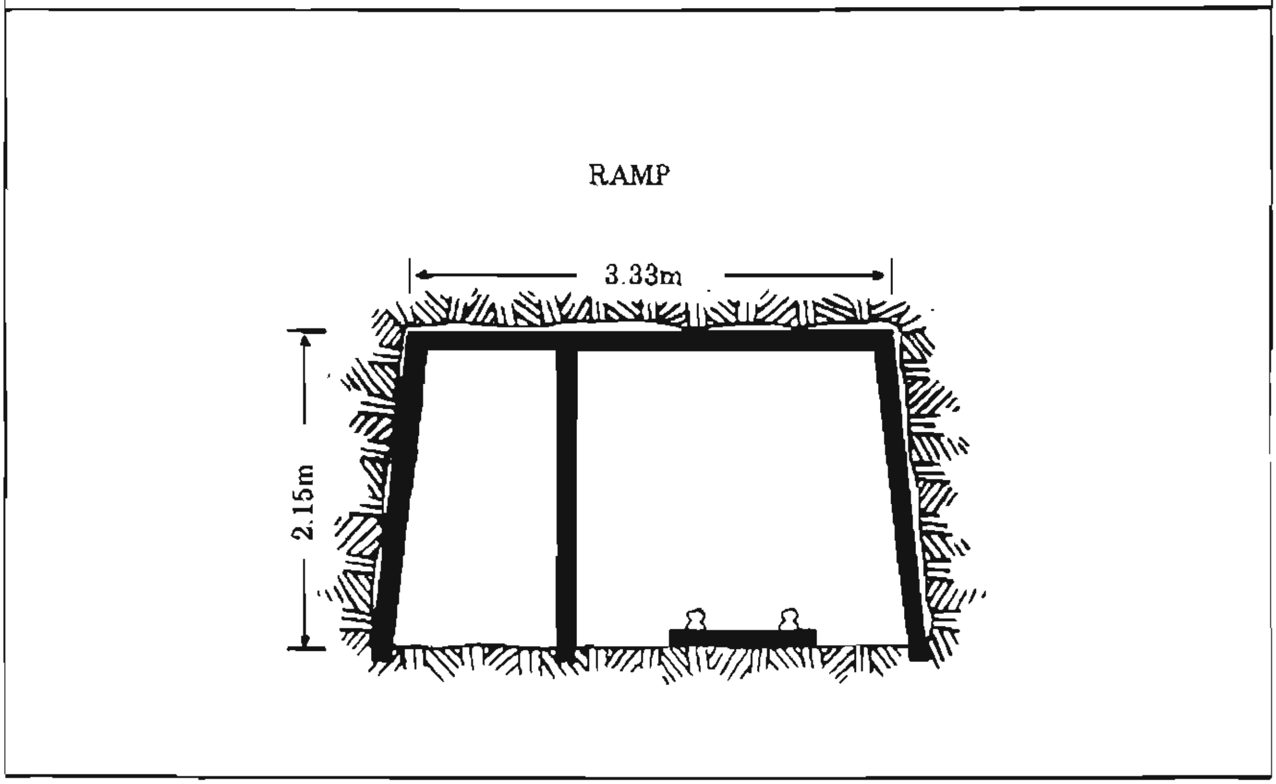


Figure 5. Cross-sectional view of loading station (240 m^3) and ramp used in Siberia. Adapted from Emelanov and others (1982).

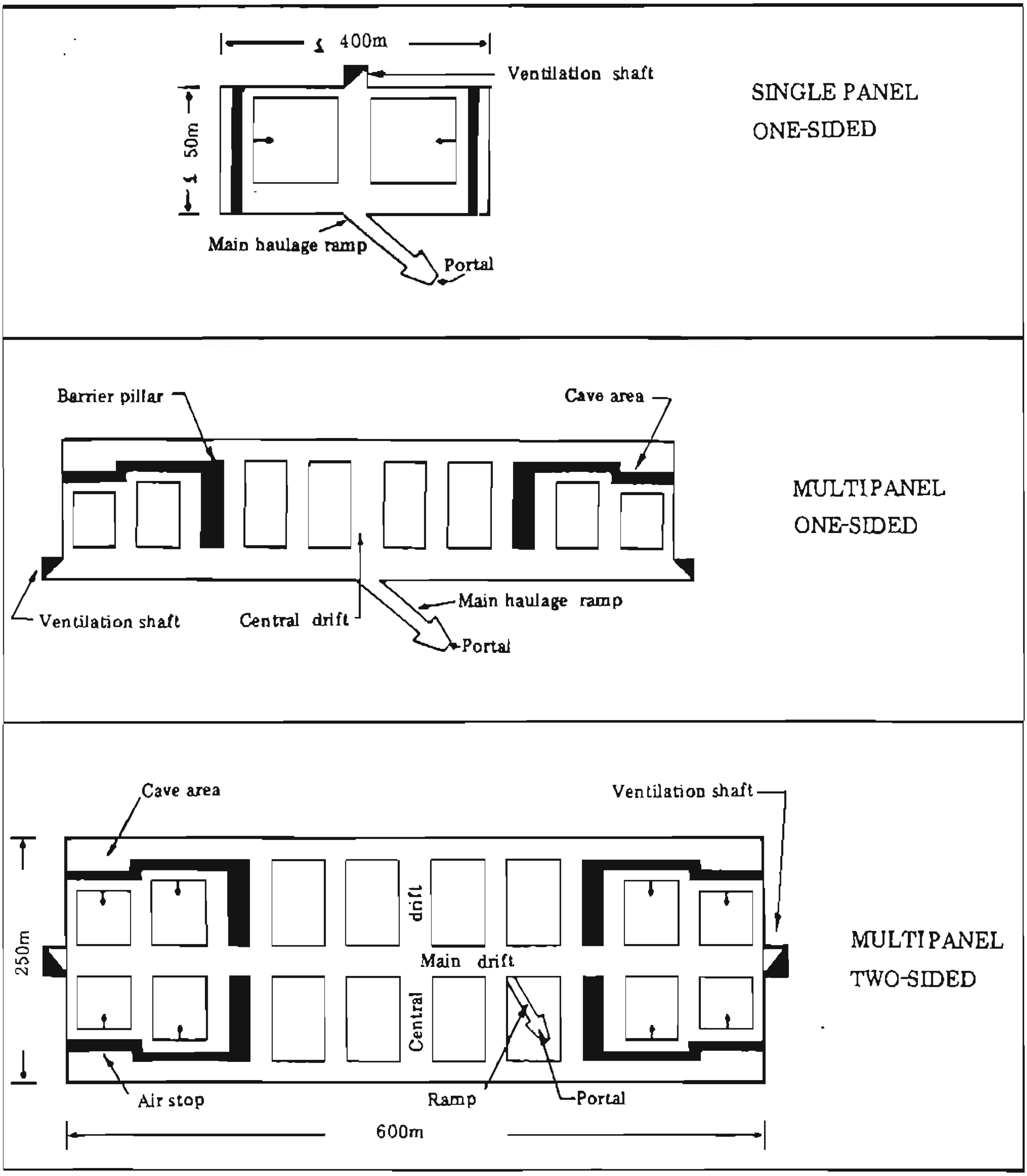


Figure 6. Plan view of mine designs in Siberia: 1) single-panel, one-sided mine capable of producing 150 to 220 m^3 of ore/day; 2) multipanel, one-sided mine capable of producing 400 to 700 m^3 of ore/day; and 3) multipanel, two-sided mine capable of producing > 600 m^3 of ore/day. Adapted from Emelanov and others (1982).

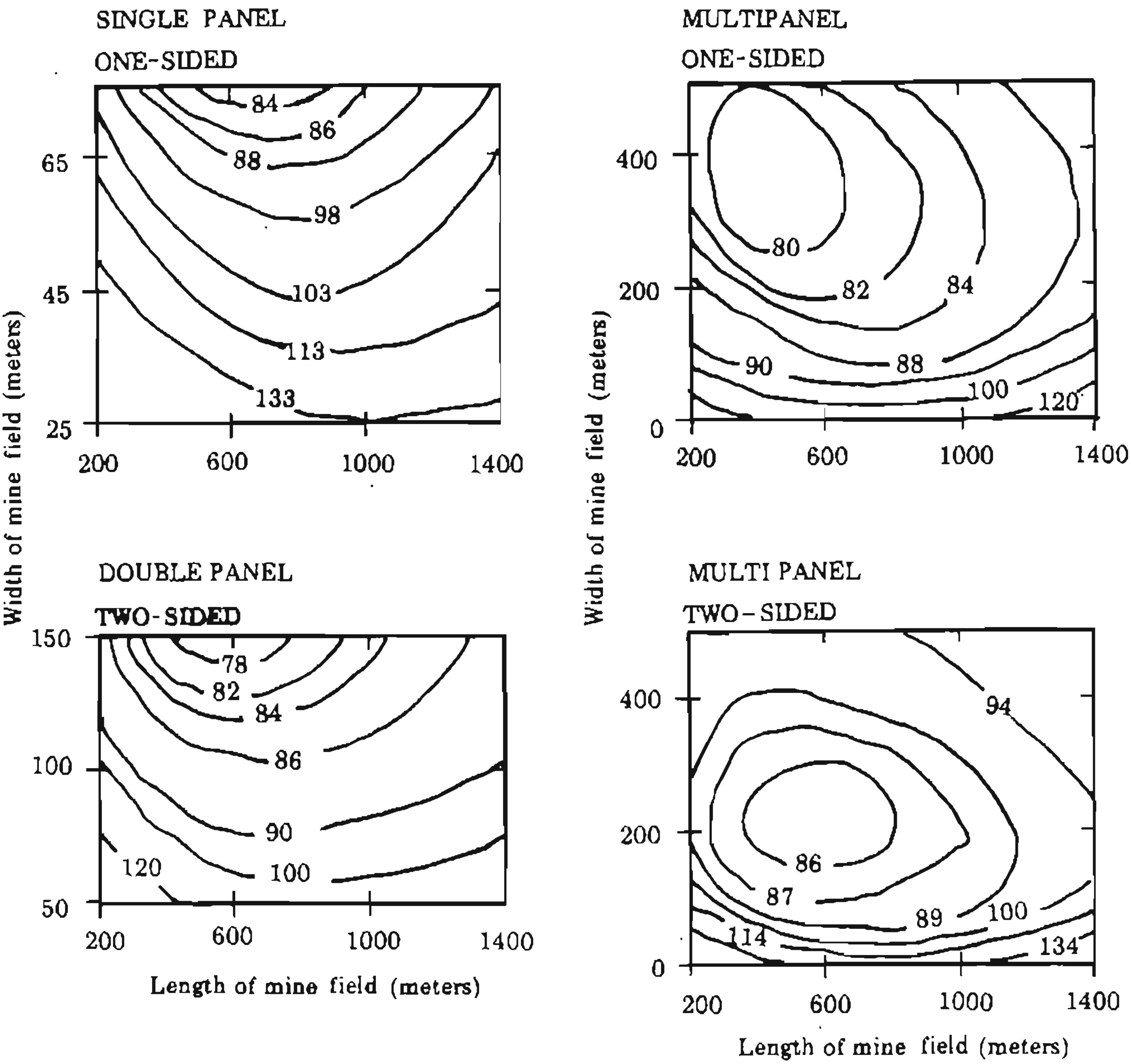


Figure 7. Cost-efficiency curves, in units, for longwall mine designs. Adapted from Emelanov and others (1982).

Unless the mine is relatively shallow (< 60 ft) and boreholes can be used effectively, the mine is ventilated with a system of outside air and return airways to an exhaust point. Standard specifications for mine ventilation can be referred to and need not be repeated here. Peculiar to permafrost mining, however, is the efficient and inexpensive use of water to create impervious icing over air stops, thereby greatly minimizing the leakage that renders ventilation systems ineffective.

Some underground placer mines have been operated throughout the summer despite higher costs. Special steps must be taken to restrict portal entry and to cool the mine air. Various schemes have been used to prevent degradation of the ventilation openings due to thawing. To some extent, the cold mine air can be recirculated by filtering. This air is mixed with outside makeup air that can be introduced and cooled by passing it through abandoned areas of the mine or through special air-cooling passages dug in the permafrost overburden. This tactic is particularly effective when the overburden contains a high percentage of ice. At least 6 m^3/min/miner of cooled outside air is recommended; additional air is necessary to compensate for diesel equipment.

In conclusion, if mineral grade is sufficient, similar mining opportunities in frozen ground may exist in North America. For example, many frozen, deeply buried pay channels are known in the Fairbanks area and several are being explored. At Livengood, a placer deposit that contains 18 million yd^3 of proven pay gravel has been suggested for underground development (Albanese, 1981), and in 1986, an underground room-and-pillar operation was initiated on Wilbur Creek.

The time today is too short to delve further into the subject, but it has provided an opportunity to take a very brief look at the accumulated technology developed since those Alaska old-timers sank their shafts along the Chatanika River valley. Additional data pertinent to the development and operation of underground placer mines is available in the journals we have referenced.

REFERENCES CITED

Albanese, Thomas, 1981, A hypothetical underground placer operation at Livengood: Mineral Industry Research Laboratory Report 52, p. 61-84.

Cieslewicz, W.J., 1987, Gold in northeastern Siberia: Society of Mining Engineers Preprint 87-19, 7 p.

Emelanov, V.I., Mamaev, U.A., Kublay, E.D., 1982, Underground mining of frozen placers: Moscow, Nedra.

Lubey, K.I., 1978, Underground mining technology for deep-seated placer deposits in the northeastern part of USSR: Moscow, Nauka.

Sherstov, V.A., Skuba, V.N., Lubey, K.I., Kostrometenov, K.I., 1981, Underground mining of placer deposits in Yakutzya: Siberia, Yakutzya.

Strishkov, V.V., 1981, The mineral industry of the U.S.S.R., _in_ Minerals yearbook, 1978-79 (volume III): U.S. Bureau of Mines, p. 957-1005.

STATE-OF-THE-ART, ON-SITE CYANIDATION AS APPLIED TO PLACER MINING

By
R.W. Clark, Consulting Mineral-processing Engineer
1611 Finch Way
Sunnyvale, California 92714

Can gold cyanidation increase the profits of your placer-mining operation? This paper discusses the feasibility of using the cyanidation process on a site-specific basis.

Alkaline cyanide in an alkaline solution is a weak solvent for gold (and contained silver); most mill operators use it to dissolve fine gold particles only. A practical maximum size is no greater than 50 microns (0.05 mm; 0.002 in.) on the thin dimension of a gold particle (too small to call a nugget, yet visible). In large lode-gold plants, where some coarser gold particles occur, it is common practice to capture the larger particles by a gravity concentrator, such as a jig, stake, pinched sluice, or Reichert cone, and separate them from associated heavy-mineral grains by amalgamation (as is common in placer operations). The remaining fine gold particles are then recovered by cyanidation.

The first step to determine if cyanidation should be added to your operation is to sample your gravity concentrates and plant tailings. These must be accurate samples that contain the full size range of particles of the stream in question, including the clay. Additionally, you have to know the volume/hr or tonnage/hr that these samples represent. The sample must be large to have much meaning (the size of the sample varies as the cube of the largest particles of gravel contained). To be representative, the sample should be collected over several days of normal operation, and it should consist of several barrels that contain gravels up to 12 mm (½ in.). A sample that consisted of only minus 1-mm material (0.04 in.; minus 16 mesh) would only have to be about 50 kg (100 lb).

The information you must gain from your sample is the amount of gold contained in the minus 147-micron (0.147-mm; minus 100-mesh) fraction. The coarse gold in the sample should be recovered by amalgamation and the fine gold by fire assaying. The size of the fire-assay sample should be large, preferably about 150 gm. From these assays and proper calculations, you will know the total remaining gold contained in your placer deposit. You will not likely recover all of this micron-fine gold by cyanidation, but it's the first step to determine if further testing is justified.

Your plant addition for cyanidation will cost several thousand dollars (1982) for each ton/24-hr day of fine sand treated. The smaller the size of the anticipated plant, the higher the per ton value. These very crude costs must be considered before taking the next step, which is cyanide bench testing.

The cyanide bench tests, also called jar tests, can be performed by any good metallurgical laboratory. They require only a few hundred grams of your minus 100-mesh sample (it would be best to use the part rejected from your fire assays). The testing laboratory will be able to report the percent-gold extracted and the estimated consumption of chemicals, such as sodium cyanide and lime. And, most important, you will have a good idea of how long it takes to dissolve the gold for the extraction-percent reported. Some good test procedures are described in Conwell (1980). It would be well worthwhile to review his work.

In most cyanidation operations, the gold particles require 24 to 48 hr for complete dissolution. For each 24-hr ton of solid that you treat, you will have to have tanks that contain 1 to 2 tons of very fine sand and an equal amount of water to produce a slurry of about 50-percent solids. The slurry in these tanks is agitated to keep particles in suspension. With the new draft-tube agitation concept, only low horsepower would be consumed. Usually a series of six or more tanks are required; costs can be saved if these can be shop fabricated and assembled.

On the basis of cyanidation tests, you now have conservatively adjusted the total amount of gold (and silver) you expect to recover over a certain number of years consistent with your present operating practice. If you still believe that this additional recovery may be profitable, you are ready for the next step in your evaluation. It only costs about one percent of a capital investment to conduct a study that will clearly show the cash flow of your proposed project. If it turns out to be unprofitable, you will have spent very little money.

The study begins with a flowsheet and material balance from which equipment can be sized, selected, and priced. With the foregoing, and a simple plot plan, a skilled engineering estimator can figure what your investment will be. You add to this the cost to run the addition: labor, power, chemicals, repairs, and so forth. Then you have an accountant conduct a cash-flow analysis.

We will start with the flowsheet. Normally there will be no grinding, because nature has liberated most or all of the fine particles of gold. Much will depend on results from earlier bench-test work. The point is that all gold particles will never be entirely free from other mineral grains; this is called 'locking.' In simple locking, where the particle of gold can be reached by the cyanide solution, there will be no problem, it will dissolve. Or if the rock encapsulating the gold particle is porous, there will be no problem, it will dissolve.

Only the fine mineral grains will stay in suspension, so all of the feed will have to be screened. The coarser the particle size, the more power required to suspend the particles in the slurry. The coarsest practical size today is about 300 to 400 microns (35 to 48 mesh). There are several good screens for this. The most popular are 'Derrick,' 'Tyler Hummer,' and 'Rotex.' The screening has to be done wet and additional water has to be added to the underflow to adjust the resultant slurry to 50-percent solids.

The slurry must enter the first leaching tank at a uniform rate, 24 hr/day, 7 day/wk to obtain good performance. Once the slurry is in the first tank, the movement from tank to tank is by means of air lifts. Feed sources can be from sluice tailings via pump or from an amalgamation barrel or from both. If the sluice tailings contain over 50-percent water, they will first have to go to a spiral classifier to remove coarse sand and gravel and then to a thickener, where flocculating agents can be used to keep the slurry size small. Thickened underflow can then be pumped to the screen. Amalgamation barrel tailings, usually discharged in batches, must be collected in a surge hopper or bin and, since wet, have to be fed by an interfolded-type screw feeder onto the screen.

In this paper, I will discuss the carbon-in-pulp (CIP) process for recovery of the dissolved gold and its ultimate reduction to metal. The older zinc-precipitation process (sometimes called the Merrill-Crowe process) will also usually work well, but the plant cost is higher. It is still practical for high-silver-content ores, but most placer ores only contain a minor amount of silver as an alloy with gold.

The carbon particles are added to the same pulp as the mineral particles, but movement of the carbon particles from tank to tank is in the opposite direction from the mineral particles because of the carbon's distinctly larger particle size. This creates a counter-flow action between the mineral particles and the carbon particles. At each air lift, the slurry of mineral and carbon particles is discharged onto a small screen. The screen undersize slurry advances to the next tank, while the screen oversize carbon particles are advanced, in reverse direction, to the preceding tank. Only part of the carbon is advanced by means of a splitter; the remainder returns to the same tank. Not all of the tanks will be in the counterflow system of CIP, and sometimes the first tank receives no cyanide, but these are process refinements beyond the scope of this paper.

As the cyanide dissolves the gold, the gold-cyanide molecules come in contact with carbon particles and are absorbed onto the carbon surfaces. We will not take time to discuss the theory because all we need to know is that it works and that the carbon must be in the activated state. The best carbon to use is made from coconut shells because it resists breakage. Fines from breakage mean gold losses, because they go through the screen on the final tank and on to tailings impoundment.

The carbon will be loaded up in normal practice to between 7,000 and 14,000 grains of gold/ton of carbon (200 to 400 troy oz). The value of this carbon at a gold price of $400/troy oz (ignoring the silver) is from $87.5 to $175/kg of carbon ($37.5 to $75/lb). At this value, there is a danger that your plant operators may try to 'high-grade,' so the system must be enclosed and the loaded carbon received in a locked storage bin.

The next step is to recover the gold from the carbon. There are two ways to do this. Very small operations will find it most cost effective to oxidize (burn) the carbon to ash, then flux and smelt to dore' bullion. Alternately, you may be able to sell the carbon to a smelter. For larger operations, it is common practice to chemically strip the gold from the carbon and reduce the gold to metal by electrodeposition on a steel-wool cathode or by replacement with zinc dust (Merrill-Crowe process).

If stripping and reduction is included in the flowsheet, the procedure is as follows. The original Zadra (1952) method was to pass an aqueous solution that contained 0.1-percent sodium cyanide plus 1.0-percent sodium hydroxide through the carbon column at 85°C for 24 to 60 hr or until the carbon was stripped down to a gold content of 150 gm/ton of carbon (about 5 troy oz). By raising the pressure above atmospheric, the temperature can be raised to 160°C, which allows stripping time to be reduced to between 2 and 6 hr. Another method also uses atmospheric conditions, a temperature of 85°C, and a chemical solution of 0.1-percent sodium cyanide and 1.0-percent sodium hydroxide, but this solvent is mixed with 20-volume-percent ethyl alcohol. The stripping time is reduced to 5 to 6 hr, but the method is a fire hazard (Zadra and others, 1952; Ross and others, 1973; and Heinen and others, 1976). A complicated method from South Africa (Davidson and Duncanson, 1977) involves using one bed volume of 5-percent sodium hydroxide plus 2-percent sodium cyanide for 2 to 6 hr of preconditioning at 90°C, followed by a hot-water wash of five bed volumes for 5 to 7 hr. The total cycle time can take up to 20 hr.

The pregnant solution, in closed circuit, passes into the reduction system (either electrolytic cells or zinc precipitation), while the resultant barren solution is returned to the carbon column for further stripping. Heat and chemical strength are replenished along the circuit. The gold-enriched steel-wool cathode or the zinc precipitates are dried, fluxed, and smelted, and excess zinc is leached with nitric acid.

After the carbon is stripped of gold and silver, it must be regenerated because it gradually looses its adsorption qualities. Reactivation is accomplished in an indirect-fired kiln. In addition, an acid leach is used to remove calcium salts. Small operations may eliminate these steps by sacrificing a part of the carbon to oxidation, thus trading higher carbon consumption for capital equipment.

Now we must turn our attention to disposal of the barren slurry from the last leach tank. The slurry can be air lifted to a screen to separate out the carbon, then sent to a tailings impoundment area. This area should be used for slurry only and should be large enough to contain the total volume of solids expected for the life of the placer deposit. In addition, it should contain an impervious seal to prevent seepage into the water table. The slurry can be deposited in such a manner that a clean lake is maintained. The clear solution is then pumped back to the cyanide plant to reclaim residual sodium cyanide and to increase the pH of the lime. Any overflow from the tailings pond must be neutralized by oxidation of the cyanide with chemicals, such as sodium hypoclorite (washing bleach). In arid climates, it may be necessary to intentionally remove a bleed stream, oxidize the cyanide, and discharge to waste. This prevents the buildup of mineral compounds that poison the cyanide process. The 'discharge to waste' may have to be sent to a separate evaporation pond.

To summarize, the following is a list of equipment likely to be included in such a plant (numbers correspond to those on fig. 1):

1 Bin[1] - Holds amalgamation tailings.
2 Screw feeder (interfolded type)[1] - Discharges ore onto screen.
3 Screen - Separates 35 to 48 mesh; oversize joins gravity-tailings stream, undersize moves to tank 1.
4 Tanks (6 or more) - Provide 24- to 48-hr retention time for 50-percent solids slurry. Include agitators (preferably draft tube), a sparge ring for bubbling air through slurry, and an air lift for advancing slurry to next tank.
5 Screens - For air-lift discharges. Have coarser openings than item-3 screen yet fine enough to retain carbon particles; contain appropriate chutes and splitters to direct products.
6 Gravity flume or Pump and line - Transports cyanide tailings to tailings pond.
7 Bin - Holds barren carbon.
8 Screw feeder - Feeds barren carbon from item-7 bin into highest numbered tank.
9 Bin - Receives gold-loaded carbon from lowest numbered tank that contains screen (probably not tank 1).
10 Eductor - Transfers gold-loaded carbon to stripping column on a batch basis.
11 Stripping column - Holds carbon during stripping operation.
12 Strip-solution tank - Holds barren strip solution.
13 Pump (strip solution) - Circulates strip solution through system.
14 Zadra-type electrowinning cells - Win gold from strip solution being plated out on steel-wool cathodes. System complete with rectifiers.
15 Bullion furnace - Smelts and reduces fluxed cathode to dore' bullion. Could be same equipment used to smelt sponge gold from amalgam retorting.
16 Kiln (indirect fired)[2] - Regenerates carbon after stripping. Complete with quench tank.
17 Eductor[2] - Transfers barren carbon back to item-7 bin.
18 Tank (carbon leach)[2] - Included in carbon-treatment circuit before or after reactivation in item-16 kiln. Typical leach is nitric acid.
19 Screen - Removes fines from regenerated and new carbon. (Some operators also attrition-scrap their carbon to break off corners that otherwise would break off in the CIP circuit and carry gold values to tailings.)
20 Blower - Supplies air to sparge rings in item-4 tanks.
21 Air compressor - General use.
22 Package boiler - Small electric unit to indirectly heat strip solution.
23 Pump - Returns tailings-pond solution to cyanide plant.
24 Tank (mill head) - Supplies barren solution to slurry.

---

[1] Used only if black sands are treated.
[2] Reactivation and leaching processes can be circumvented in smaller plants if part of the carbon is continuously oxidized for residual gold values. This practice, however, will increase new carbon consumption.

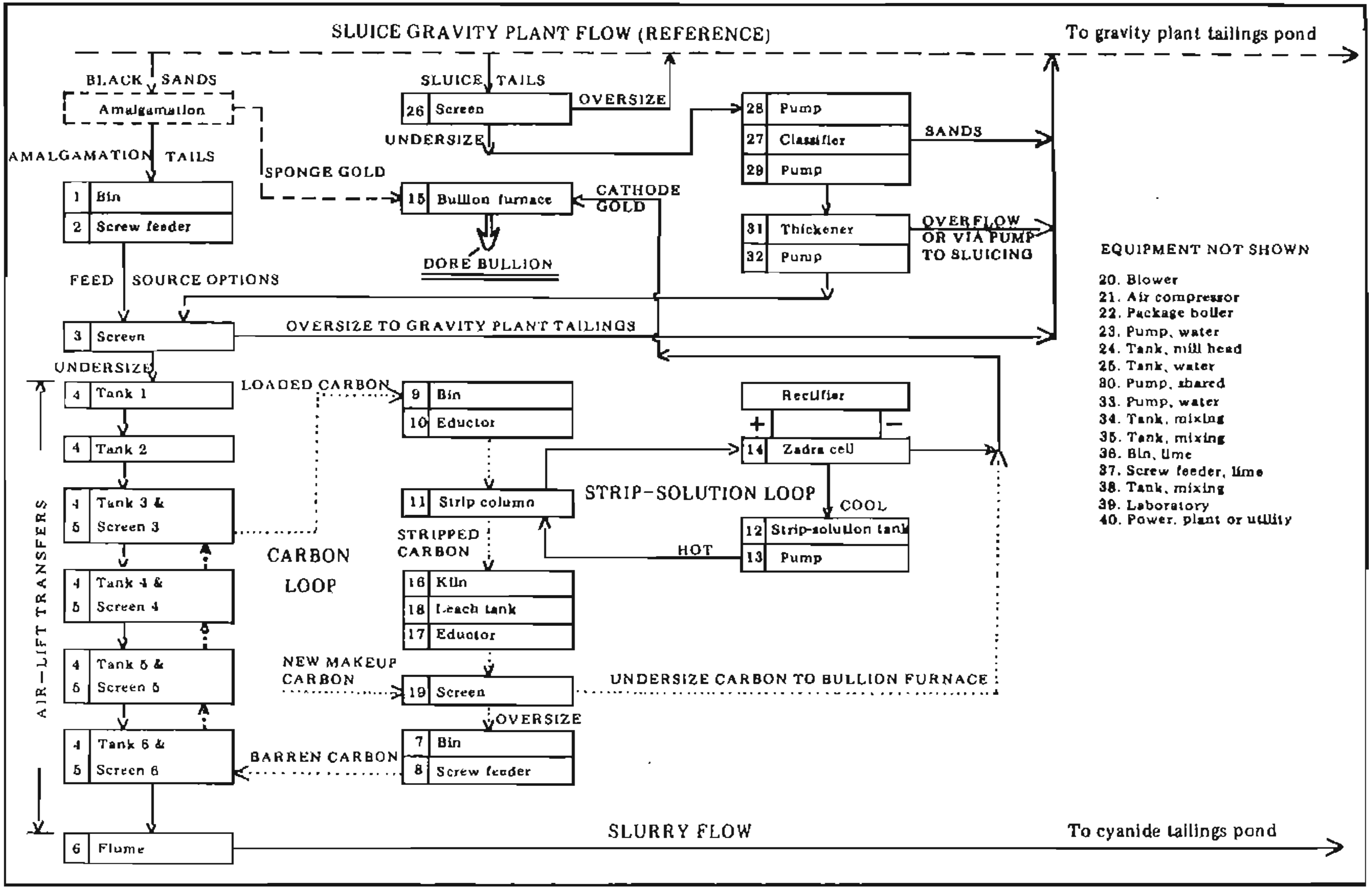


Figure 1. State-of-the-art, on-site cyanidation flow chart.

25 Tank (water) - Supplies process-water makeup (may not be required if rainwater collection is sufficient).
26 Screen (gravel)[3] - If sluice tailings are to be treated, removes gravel that is too large for item-27 classifier.
27 Classifer (spiral)[3] - Receives item-26 undersize via item-28 sand pump.
28 Pump (sand)[3] - Pumps sand and slimes to item-27 classifier.
29 Pump (sand)[3] - Pumps classifier overflow to item-31 thickener.
30 Pump (sand)[3] - Shared on-line spare for items 28 and 29 pumps.
31 Thickener[3] - Receives item-27 classifier overflow via item-29 pump, thickens slurry to 50-percent solids, then discharges it to item-32 pump.
32 Pump (sand and spare)[3] - Pumps underflow slurry from thickener into item-4 tank 1 via item-3 screen.
33 Pump (dirty water)[3] - Recovers thickener overflow for sluicing-water-supply system (optional).
34 Tank (mixing) - For makeup of cyanide solution. Complete with egg basket and delivery and circulation pumps.
35 Tank (mixing) - For flocculant preparation; used with item-31 thickener.
36 Bin - Stores hydrated lime.
37 Screw feeder - Feeds lime from item-36 bin into item-4 tank 1.
38 Tank - For sodium-hydroxide dissolution. Complete with agitator. (Use if reagent received as beads, otherwise use alternate equipment.)
39 Laboratory - Equipped to assay gold and silver by atomic-absorption and fire-assay methods. Also equipped for amalgamation tests and wet analysis of pH, lime, and cyanide strengths.
40 Power (plant motors and so forth) - Produced from utility or diesel generator.

The design of such a plant will require the services of skilled metallurgical and design engineers to properly size and arrange equipment and to include any flowsheet modifications that may be unique to your deposit. Laborers presently employed at a placer site can operate this type of plant as part of their routine duties; however, additional laborers will be required to operate the plant at nighttime and on the weekends.

REFERENCES CITED

Conwell, C.N., 1980, Gold recovery from placer concentrates by cyanidation: Western Miner, v. 53, no. 10, p. 34-40.

Davidson, R.J., and Duncanson, D., 1977, The elution of gold from activated carbon using deionized water: Journal of the South African Institute of Mining and Metallurgy, v. 77, no. 12, p. 254-261.

Heinen, H.J., Peterson, P.G., and Lindstrom, R.E., 1976, Gold description from activated carbon with alkaline alcohol solutions, in World mining and metals technology: New York, American Institute of Mining Engineers, p. 551-564.

[3]Used only if sands and slimes from sluice tailings are treated.

Ross, J.R., Salisbury, H.B., and Potter, G.M., 1973, Pressure stripping gold from activated carbon: American Institute of Mining Engineers' Meeting, Chicago, 1973, Proceedings.
Zadra, J.B., Engle, A.L., and Heiner, H.J., 1952, Process for recovering gold and silver from activated carbon by leaching and electrolysis: U.S. Bureau of Mines Report of Investigations 4843.

EVALUATION-DECISION PROCESS FOR SMALL-SCALE PLACER-GOLD MINING

By
M.J. Richardson, President
Consolidated Placer Dredging, Inc.
17961A Cowan
Irvine, California 92714

PURPOSE

This paper is intended to develop criteria for selecting the appropriate excavation and processing plant for small-scale placer-gold mining. While there has been a great deal of effort directed toward setting up gold mining in placer deposits over the past several years, few, if any, have resulted in commercially successful mines. There is, however, a similarity among unsuccessful operations that we have observed that characterizes them and has contributed to their failures.

PROPERTY CLASSIFICATIONS

It is difficult to generalize about mining properties and, in particular, placer deposits. When it comes to prescribing the type of equipment to mine a property, the recommendation will vary with different surface environments and soil substructures. The following are some of the more common conditions in which placer deposits are found:

1. Alluvial
   a. Rivers and streams, including their basins and terraces
   b. Tertiary gravels (elevated terraces)
   c. Beach sands
   d. Desert fans, flood plains
   e. Arctic permafrost, inland and coastal
2. Glacial moraine
3. Eluvial
   a. Desert concentrations
   b. Mountainous as in Serra Pelada, Brazil

EVALUATION

The most common mistake made in small-scale mining projects is the lack of competent evaluation prior to committing to the purchase of mining equipment and to mining itself. Proceeding to mine without prior evaluation is like shooting in the dark. Who would invest in a supermarket in a ghost town, hoping that once it was built people would show up? Just because someone may have found gold nuggets in a stream does not mean there is sufficient gold to amortize an investment in mining equipment. The goal of a placer evaluation is to establish, through proven means, the quantity of gold that exists and to determine if it is in sufficient volume and if it is present under conditions that would permit mining at a profit.

Mining gold to many people is synonymous with gambling, a belief that luck is somehow going to play a major part in the process and that a rational approach to an assessment of the prospect, as in other businesses, is not appropriate. When people view mining from that perspective, they should not be disappointed that the odds are against them. Professional mining companies, however, depend upon the competence of trained mining engineers and geologists to make their best estimates and, thus, are able to stay in business.

When it comes to evaluation, each type of placer deposit has its own peculiarities. One of the basic considerations, however, when choosing the method of sampling is that the procedure should simulate a proven system of recovery used in a gold-mining dredge. The first step of evaluation is to establish the presence of gold in the prospect and then, by inspection of the terrain with the aid of an experienced eye of a placer-mining engineer, estimate the potential volume of alluvial or eluvial material to support a mining operation. The next step is to evaluate the deposit with a churn drill to determine the tenor in milligrams per cubic yard (or cubic meter). Once an average tenor is established for the deposit, calculations can be made to determine if the potential return will be sufficient to support a capital investment in equipment.

Sampling

Our experience has been that the churn drill is the primary tool to sample a placer-gold deposit. This is true because of the considerable proven use of the churn drill in large dredging projects over many years, followed by production that verified the values found from drilling. With a churn drill, a miner is able to saturate a broad area down to bedrock, testing the soil horizons and accumulating data that can be evaluated on a large-volume basis with consistent results. Accuracy will depend on the practices used, the logging and observations, and calculations and adjustments based on experience with commercial-scale placer mining. An exception is in the case of the hammer, reverse-circulation drill, which we have used with a high degree of success in placers in recent years. It acts like a churn drill, insofar as the casing is driven downward rather than in a rotary fashion, except in the extraction process, which is accomplished by either compressed air or water pressure. The hammer drill is as much as 10 times faster than the engine-powered churn drill.

The alternatives of shafting, pitting, or bulk sampling are means that were followed before the churn drill became accepted into practice after its introduction in the mid-1800s. Once the large-scale placer-dredging firms found the reliability of drill sampling, they tended to discard the old methods of bulk sampling. After they introduced mineral jigs into their dredge recovery systems, the reliability of drill sampling was 100 percent of drill values. Sometimes bulk sampling of a particular area may be justified in conjunction with drilling as a check on the drill values and adjustment factors and to evaluate the bulk nature of the soil for mining purposes, but this should not be confused with and applied to a total sampling program.

Drilling Saturation

A grid system of drilling can block out the reserves and provide a basis for calculating the potential of the placer prospect. Proceed first with a scout drilling program to determine if the values of gold continue through the horizons, if there are false bedrock layers or other characteristics that should be known about the deposit, and the extent of the horizontal or geographical limits. On the basis of the sampling to establish sufficient tenors and their extent in depth and area of occurrence, it can be determined if there is a probability of a viable mine. If that question is confirmed, it can be said that there are probable reserves. The next step is to organize the grids within the boundaries that were delimited.

The development drilling that ensues will determine the limits or boundaries of higher tenors for a profitable mining operation. The spacing of lines and saturation of the area with drilling will depend on the classification of the deposit, its characteristics, and any proof of earlier mining that would add some confidence to the sampling. When this is completed and the results are adequate to support an investment and operation, the deposit can be defined as indicated reserves. On the basis of that data, a feasibility study of the proposed operation and the selection of equipment that would be most appropriate for mining and recovery can be conducted. Should the results be positive, the next step would be to conduct closer drilling that will saturate the area to a degree that is acceptable for investment and mining decisions. The ratio can vary for the number of drill holes that constitute an acceptable saturation and can be from a high of one hole per acre to as little as 10 per acre. Once this drilling is completed you have established proven reserves and can finalize the feasibility study and support financing or investment capital, provided that the results substantiate it.

FEASIBILITY DESIGN ANALYSIS

Primary Elements

Once the prospect evaluation has been completed, a final feasibility study can be prepared. The primary elements of the study are as follows:

1) Tenor of the gold in place (mg/yd^3).
2) Volume of material (yd^3) that must be excavated and processed to produce the total volume of gold (yd^3), applying the average tenor (troy oz.) from (1) above.
3) Character of the soil and environment that must be dealt with to recover the gold.
4) Equipment configuration to be used that is most suitable to the above conditions.
5) Financial limitations and strengths of investors in order to decide the magnitude of the operation, which will govern the choice in equipment and the plan to proceed.
6) Projected operation and related costs, based on the equipment decisions made and the projected price of gold, thus providing the essential dollar values to be used in a cash-flow analysis (table 1).

Table 1. Sample economic analysis of a placer-gold mining operation with a portable washing plant.

Assumptions:	
Average tenor of deposit (mg/yd^3)	250
Total proven reserves (yd^3)	8,000,000
Average production rate (yd^3/mo)	100,000
Total period of mining (yr)	6.7
Total investment with interest	$1,500,000
Cost of operation (per yd^3)	$1
Fixed price of gold (per troy oz)	$300
Revenue:	
Production x tenor x price of gold (100,000 x 0.40/31.1 x $300)	$241,000
Operating cost[a]:	
Production x cost of operation (100,000 x $1.00)	$100,000
Net operating profit	$141,000
Total net profit:	
Period of mining x net operating profit (6.7 x $141,000)	$9,400,000
Less capital investment	$1,500,000
Net before royalty or taxes	$7,900,000

[a]Includes labor and engineering, fuel and maintenance, utilities, equipment rental, replacement parts, insurance, personnel benefits, and administration costs.

Assumptions and Graph Description

The graph in figure 1 shows a division between a portable washing plant and a bucket-ladder dredge that is primarily involved with the excavation rate of the equipment. A fixed price of $300/troy oz of gold is used on the abscissa and computed against monthly production to produce revenue. The ordinate is the monthly production rate in cubic yards of a dredge or a washing plant that is fed by equipment such as a backhoe, dragline, or front-end loader. The three trend lines are for different tenor levels of the deposit: 150 mg/yd^3, 350 mg/yd^3, and 500 mg/yd^3. The operating cost line on the left is based on upgrading of dredging equipment as volume increases. Assumptions of the size of dredges, in terms of bucket volume (ft^3), are delineated on the right margin. The dredge-size selection is based on sufficient dredging operations and equipment as experienced over some 55 yr of operation with gold-mining, bucket-ladder dredges. Profitability can be measured horizontally at any point on the graph, from the operating cost line horizontally to the slope that is appropriate to the tenor of the deposit. Assumptions about equipment size are expanded on in figure 2.

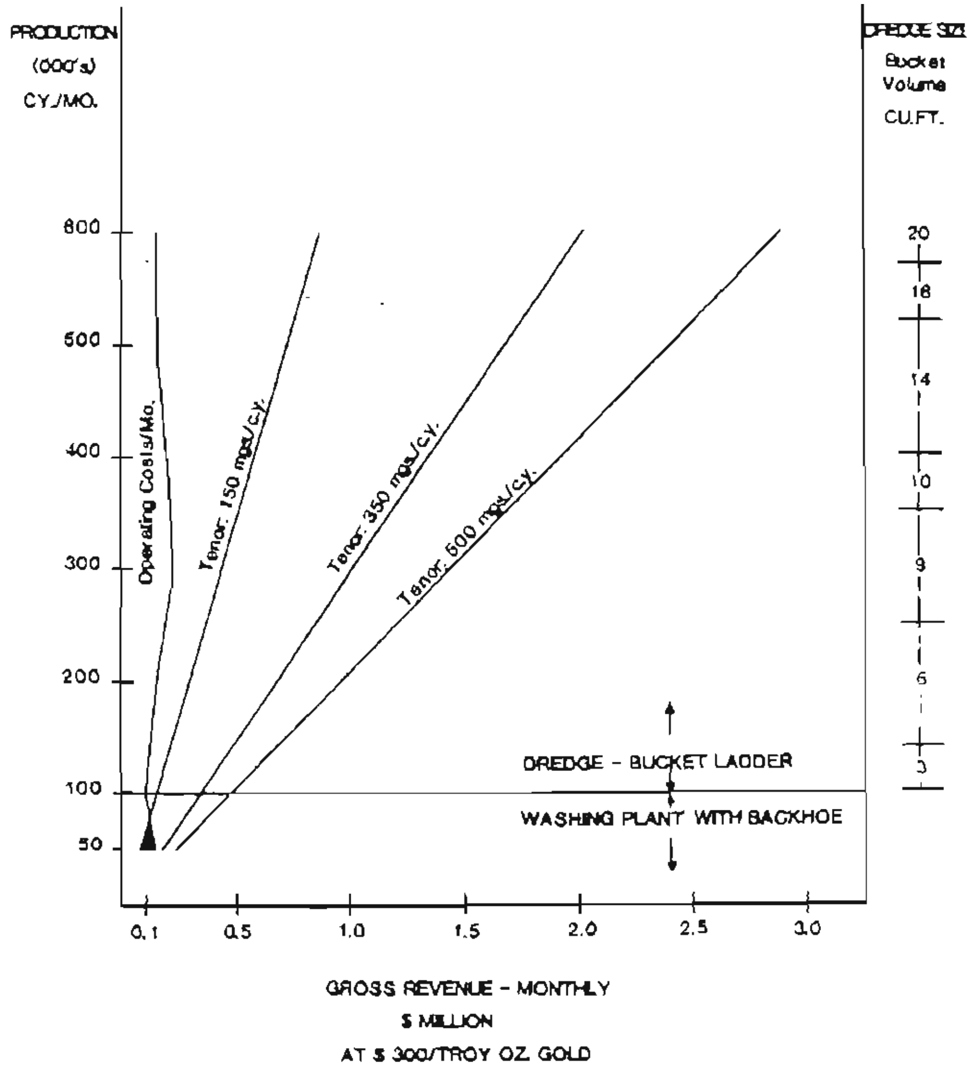


Figure 1. Placer-gold mining-equipment tradeoff.

Production Cutoffs: Washing Plant vs. Dredge

The cutoff line for the production rate of a washing plant has been selected as 100,000 yd^3/mo (figs. 1 and 2) and assumes an average production level of 200 yd^3/hr when operating, combined with an efficiency factor of 70 percent (that is, actual bucket fill and operating time vs. theoretical 100-percent fill and operating time). Our experience suggests that production much above that level introduces so much multiple handling of material, in the case of a portable plant, that it is more efficient to upgrade the equipment to a floating dredge, if possible.

There are also such factors as the excavation device and its ability to clean bedrock, which is usually limited to a backhoe when dealing with portable equipment, combined with the feeding capacity of the plant to deal with ever increasing volumes of material in a single dump. This is the difficulty with a dragline, for instance, because the plant can only operate efficiently

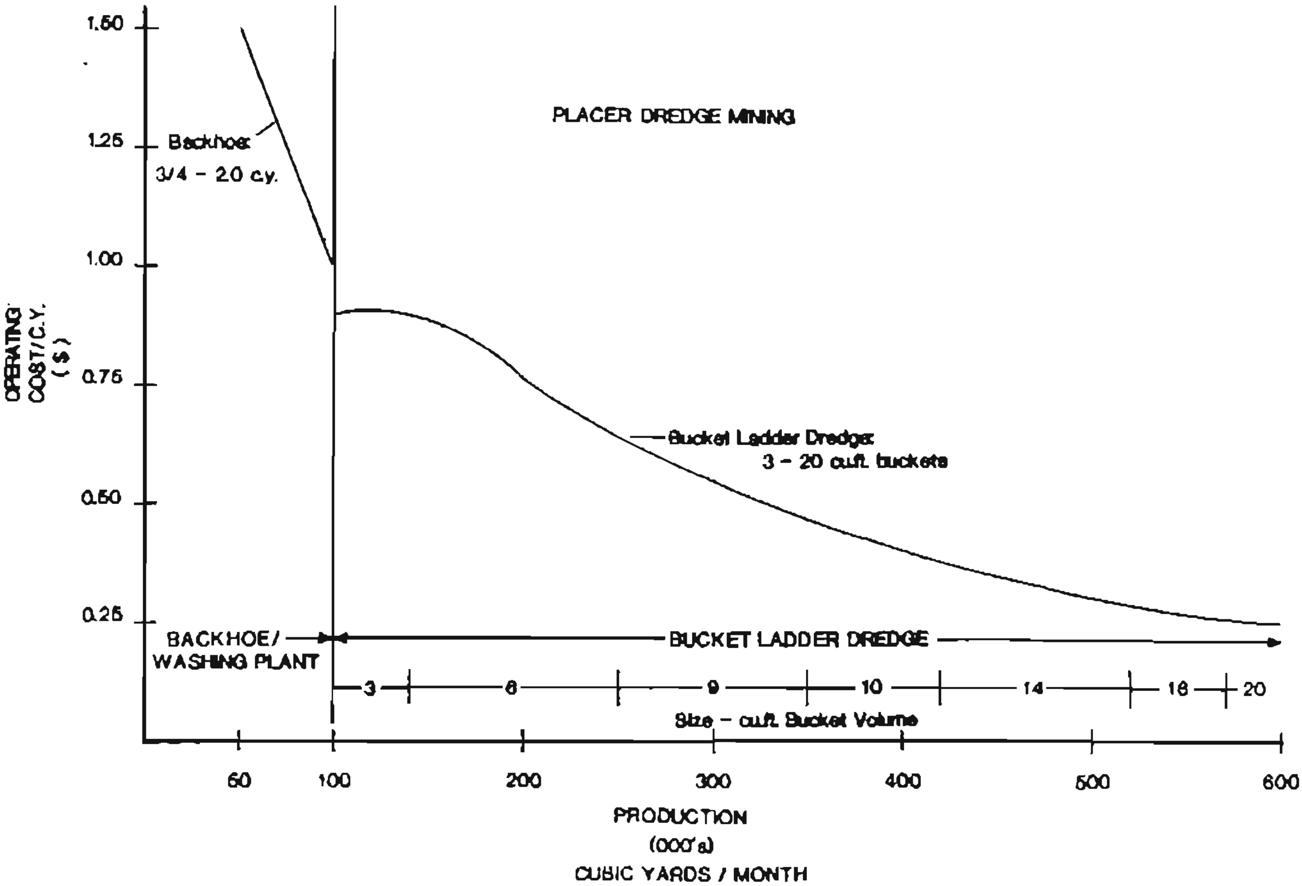


Figure 2. Cost of operation vs. production.

when there is some uniformity of flow of material through the system. The tendency with a dragline is to increase the volume of the bucket, but this compounds the problem of dump volume.

Not considered on figures 1 or 2 is the total proven gold-bearing volume of material in a deposit. If, for example, the tenor of a deposit was 500 mg/yd^3, but the total volume was only 300,000 yd^3, a capital investment in mining equipment would have to be balanced between minimizing the duration of mining and minimizing the capital cost of equipment. A 14-yd^3 bucket-ladder dredge, for example, would mine the deposit in 1 mo but would take 1.5 yr to mobilize, along with a total capital cost exceeding $5 million. A washing plant, on the other hand, if conditions would permit, would be expected to complete the mining in a few months. And the plant could be mobilized in < 6 mo for less than $1 million.

It is also important to note that the depth of the deposit is not considered in figures 1 and 2, but, again, the higher rate of dredging is assumed to be combined with a larger volume of material and a greater depth of deposit. When gold is lying near bedrock and its tenor at that level is high, its valve must be diluted when calculating the full amount of material to be removed. Thus, a lower tenor may be shown, which requires a larger dredge and higher volume producer to move the material in a reasonable period of time and at a lower cost per unit volume. Also, as the rate of dredging increases, it is expected that the cost per cubic yard will decrease. If it does not, then something is wrong with the design of the equipment or with the operation, or both, that should be corrected.

In this paper, I have attempted to make a case for a cutoff of 100,000 yd^3/mo for the production level of a portable washing plant, above which a dredge should be substituted. This means that the plant and equipment must be operating on a three-shift, 365 days/yr basis. Should that level of production not prove feasible, then production will be reduced accordingly. But it is important to note the scale of diminishing returns as plant production reduces. As the lines of tenor (fig. 1) begin to converge, profitability becomes a loss.

However, the consequences of high production in a portable plant (as far as the material-handling problem is concerned) are too often overlooked in the design stage. A high production level is essential to profitability and a cash-flow analysis should demonstrate that fact, but material-handling problems must be in balance with other considerations. While it might be possible to find a high-grade deposit in small canyons, for instance, it is usually difficult to adequately sample such a deposit and determine the average tenor and whether it is possible to maintain the necessary production. As the cost of operation line clearly shows (fig. 2), cost drops markedly once you shift to a dredge and further as production volume and size of dredge increase. The cost of operation of a small plant, therefore, must be kept under control and the rate of production optimized within practical limits of the equipment and the deposit.

From the curves shown in figures 1 and 2, we can conclude that, depending on tenor, as production drops below 100,000 yd^3/mo profit degrades. A production level of 50,000 yd^3/mo may be considered the minimum level for a commercial-size operation. The tenor at that minimum must be no lower than 400 mg/yd^3, combined with a floor of $300/troy oz of gold. It should be recognized, however, that in most placers a tenor that high is rare.

With the above criteria established, we can make a conclusion about the volume of the deposit. Unless there are some environmental or other restrictions that would prevent the use of a bucket-ladder dredge, it would be safe to say that a total volume of 8 to 10 million yd^3 would be the maximum for a portable plant. There would be gray areas to be considered that would establish a band of variability, but each case would finally be decided on its own set of conditions.

EQUIPMENT SELECTION

In this paper, I have excluded consideration of all other types of dredges in deference to a bucket-ladder dredge. This needs some qualification. The number of suction-type dredges, for instance, that have been installed in placer-gold mining operations over the years probably could not be counted, they are so numerous. We do not know, however, of a single suction dredge that has produced a profitable placer-gold mining operation. The suction dredge seems to be associated with promoter-oriented gold-mining deals that are devoid of exploration and competent mining engineers. (This is not intended to denegrate suction dredges in general, however, since they have performed, and continue to perform, a valuable function in nonmining, navigational dredging and in mining industrial minerals, such as phosphate and sand and gravel.) The fact remains that the suction dredge has its appeal when considering capital and maintenance costs in placer mining, which are both less than with the bucket-ladder dredge. This, in turn, helps to draw in the investors who would like to keep their investment and estimated risk at a minimum and can be easily persuaded that the technical capability is at least equal between dredge types. It is interesting to note that in a 1938 publication concerning placer mining with portable units, the same problem was discussed. So, as the saying goes, we are committed to repeat history when we don't read it.

History has shown that, in the vast majority of cases, the only commercially successful gold dredges have been bucket ladder. (Figures 3 and 4 illustrate large-scale bucket-ladder gold-mining dredges that are capable of producing between 300,000 and 500,000 yd^3/mo.) A suction dredge, regardless of its excavating device---if any, carries too much water (80-90 percent), does not clean bedrock, and disperses fine gold at the suction mouth. These problems are serious detriments to a profitable placer-gold operation.

In selecting the configuration of equipment for small-scale mining, which we have now assumed would be in the range of 50,000 to 100,000 yd^3/mo, we can assume that the excavation equipment, whether backhoe or other configuration, must be able to clean bedrock and efficiently feed the hopper of the washing plant. We will, therefore, at this point, look at the vital components of that plant.

Hopper Feed

The hopper must be of sufficient size to accept the volume of material that is discharged by the loader and not choke up as it disperses the material into the classifier. There must also be some water fed to the hopper to lubricate the surface, but, at the same time, the opening must be sufficiently constrictive that it doesn't permit too much material to enter the classifier at one time.

Classifier

The proven system for classification in placer mining is the revolving trommel screen. The advantage of this equipment is that cobbles and small boulders can be discharged into it and thus be washed of clay and other material clinging to the rock, which often contain gold. When a grizzly with spacing of < 8 in. is placed in front of the classifier to compensate for the latter being too weak to accept the shock loading of cobbles and boulders, values may be lost. Where the name of the game is saving gold, this can spell the difference between profit and loss. This is the problem, for instance, with shaking screens, which are often used by mining operators in handling excess water from suction dredges and avoiding the higher cost of a stoutly built trommel. Since a flat screen cannot normally sustain large

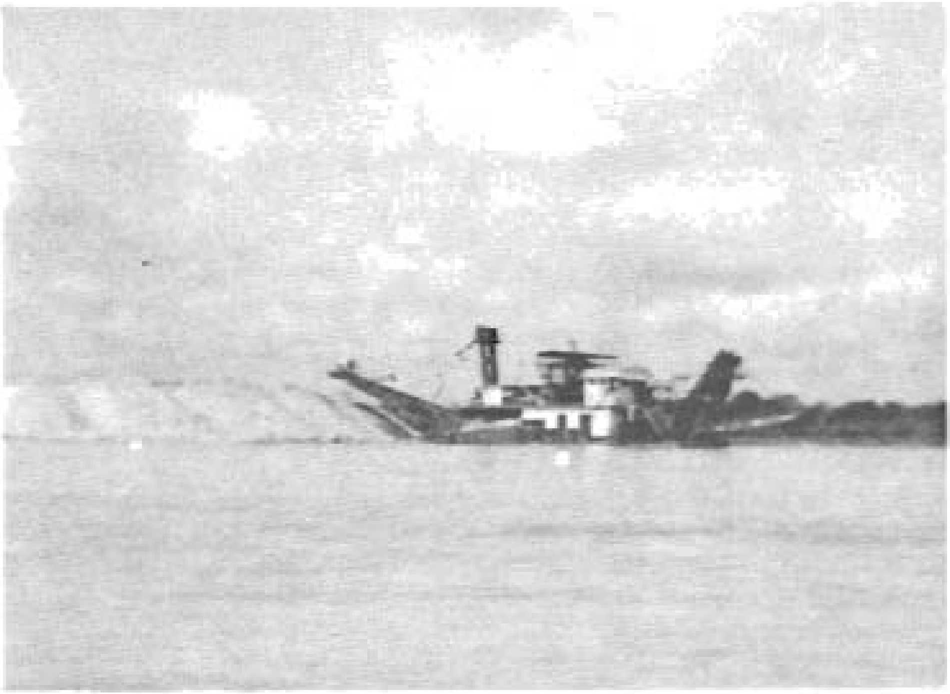


Figure 3. Bucket-ladder gold-mining dredge with 13.5 ft^3 buckets, Rio Nechi, Columbia.

Figure 4. Bucket-ladder gold-mining dredge (no. 5) with 9-ft^3 buckets, Nome, Alaska.

rocks or boulders falling on its surface, operators narrow down the grizzly spacings and thus lose a portion of the gold.

A revolving trommel with screen plates in the periphery is, therefore, the recommended approach. The tread rings, structure---or ribbing, the drive mechanism, and thrust bearings must all be built of sufficiently heavy material that they will last through the total period of the project under severe loading. Inside the trommel, a sparge pipe is installed that carries water under pressure and flows through nozzles that spray the material being tumbled to insure that it is thoroughly washed. If there is much clay, then lifting bars, and sometimes paddles and other devices, are necessary to further retard the material and break it up before the oversize passes out the rear to the discharge belt. There are several devices that have been proven, under various conditions, to handle most soils to insure the flow of undersized material through the screen holes.

The screen plates must be carefully designed to be able to sustain the abrasiveness of the material, as well as shock loading, without excessive wear. The essence of the whole function should be to make sure that all gold-bearing material is washed and diluted into a slurry that can flow through the screen holes before leaving the trommel. The holes can either be rounded or slotted, depending on the type of material being handled and the volume desired. They must, however, be tapered to prevent blanking of the holes. This is directly related to the configuration of the trommel itself, the diameter, and washing length. There are formulae that have been developed over the years to derive the length and diameter of trommels, but with variables in material composition and in the size of holes in the plates, a certain amount of controversy has been involved with this decision.

In reviewing the choices made over a long period of time concerning trommel sizing, there have been obvious mistakes made, probably because people were copying other operators who had gone before them and few possessed sufficient originality to figure out the problem correctly. One of the traps they got into was in coping with the use of riffles that required multiple flow lines from the trommel feed through the distributors. Depending upon the sizing of material that was being processed, the largest amount of material would flow out of the first third of the trommel, and the rest of the sluices or riffle boxes would be starved. This led to making the holes smaller in the first section of the trommel and gradually enlarging them at the lower sections; thus they ranged from as small as 1/4 in. at the upper end of the trommel to 3/4 in. at the lower end. When rectangular jigs were introduced, such as the Yuba and Pan American, miners effectively moved from riffles to jigs using a similar number of flow lines and the same gradations of screen holes. To correct the imbalance, a distributor splitter was developed, which rerouted the slurry, but the splitter did not address the central problem of inefficient use of the trommel surface. What the operators were doing was restricting the flow rate of the screens to try to solve a flow-line problem and wasting material in the construction of the trommels (that is, making them too large).

It was timely, therefore, when Norman Cleaveland, while dredge-mining tin in Malaya in the early 1950s, began experimenting with circular jigs.

They were not a new concept, but there was a basic problem in making them function properly, which he solved by installing wiping blades. Dispersing the material over the bed removed the tendency for the slurry to pyramid as it flowed from the center of the jig. By providing a larger jigging area in one space with one flow line, the number of total flow lines was reduced, which paved the way for greater efficiency and higher flow rates through the dredge with a given trommel size. He also experimented with slotted holes in the screen plates with some success, but this development was thwarted by the higher cost of machining plates vs. drilling round holes. This problem has been solved in some cases by using rubber screen plates that can be punched or cast in slotted and tapered form.

With the foregoing considerations, the decision can be made as to the diameter and length of the trommel, along with the types of screen plates and size and shape of holes, to achieve a flow rate that is necessary to handle the material. The slope of the screen along its axis is another point of decision. The slope is intended to maintain the flow of material through the trommel as the material is being washed, but not allow the material to move so fast that it is not sufficiently scrubbed. The speed of rotation can be varied and will effect the flow rate as well. The size of screen openings also depends on the type of primary mineral jig used and the jig's capability to handle larger particle sizes.

Mineral Jigs

The next major function to be considered in the processing plant after classification is the jigging for concentration of the gangue. The mineral jig, contrary to some opinions, is not a recovery system but only one of concentration. I hasten to add, however, that it is also a gold saver, insofar as it performs its function properly---to keep fine gold from passing over its bed into the discharge sluices---and is maintained in its best operating efficiency.

In 1979, Mr. Cleaveland made a further improvement to his jig patent and evolved the second-generation jig, which has been named the 'Mk II Cleaveland Circular Jig.' The main improvement was to eliminate the mechanical wiping blades and replace them and the center drive system with a feed box and shrouds. This innovation not only eliminates the nuisance of maintaining the blade operation but opens up the formerly blanked-out area of the center, which is the optimum location for effective jigging. The addition of the feed box provides supplemental classification of the material as well. By varying the diameter of the jig bed, the volume of feed can be varied and a control over the number of flow lines can be influenced. The size of material particles that can be handled by the circular jig will vary, but 1/2 in. has proven to be a practical limit. A 9-ft-diam primary jig can usually handle the production that is prescribed for our maximum-sized portable washing plant. This production level is 150 to 200 yd^3/hr. Should the proportion of smaller particles be such that the material discharged onto the jig bed exceeds 100 yd^3/hr for any sustained period of time, changes would have to be made.

In most cases, the constraints of the project and the material that can be efficiently handled, both in production and discharge, by a portable washing plant, whether on dryland or floating on a barge, will usually limit the feed to the primary jig to something < 100 yd^3/hr. At the other end of the production spectrum, considering a 50 yd^3/hr plant with perhaps 30 yd^3/hr onto the primary jig, a 6-ft jig could be used for the primary. These decisions can be gauged from table 2, which shows the different flow rates of circular jigs. In making a decision to scale down the production level of the plant, it should be noted that the overall plant cost would not be reduced proportionately and the ability to later scale up production would be made difficult is not impracticable. For that reason, serious consideration should be given to the building of a plant sufficiently large with the object of later increasing production, rather than scaling it down at the beginning.

Table 2. Specifications of circular jigs vs. rectangular jigs.

Diameter (ft)	Area (ft^2)	Hutches	Capacity[a] (yd^3/hr)	Lip length (ft)	Ratio[b] D/E
			Circular Jig		
18	254.3	4	381.5	56.5	6.8
9	63.6	1	95.0	28.3	3.4
6	28.3	1	42.4	18.8	2.3
3	7.1	1	10.6	9.4	1.1
		Rectangular Jig (Yuba, Pan American)			
4-cell	49.0	4	49.0[c]	7.0	7.0

[a] Varies with composition of material: the coarser the grain size, the higher the ratio D/E, within limits up to 3/4 in. For circular jigs, assume factor of 1.5 times the jig-bed area.

[b] Concentration is function of exposure time of particles to jigging action. Deceleration of slurry when radiating outward to edge, or lip, of jig further contributes to jigging efficiency. However, as distance increases beyond a practical distance in proportion to diameter and size of material, efficiency decreases. The ratio, therefore, of flow rate to lip length gives some indication of the balance between distance and exposed lip length for discharge; that is, the lower the number, the greater the efficiency of jigging.

[c] The channeled flow of slurry through the rectangular jig has several disadvantages: added friction on the sides; increased flow to compensate for the friction, resulting in fine gold lost; and a shorter lip length. These combine to reduce the capacity to $\leq$ 1:1.

The feeding of a circular jig requires only one flow line; thus, on our portable washing plant, the trommel housing can feed into one discharge hole and directly into the feed box of the jig or into a sump and then pumped into the jig. This choice can be influenced by several considerations. By mounting the trommel above the jig, the overall height of the plant must be higher. When the plant is situated on a floating barge, this can be advantageous, depending on the digging and feeding point on shore. By using a sump, the overall height can be lower and the jigs mounted alongside. The extra power required for pumping is a consideration, but, if there is clay in the deposit, the additional scrubbing action of the centrifugal pump can improve recovery. Another advantage is the ease of maintenance and operation of the primary jig when it is alongside and not cramped underneath the trommel.

After the action of the primary jig, the hutch product, which flows through a single spigot at the bottom of the cone, is pumped to the feed box of the secondary jig, which is a 3-ft-diam circular jig. At this point, the concentration of gold is high enough that a screened enclosure is necessary for security purposes. Thus the hutch discharge hose should be armored and passed through the enclosure wall into the feed box. Acting as an additional concentrator, the 3-ft jig discharges its product directly into the recovery unit, which is an amalgam system. Both jigs discharge the lighter material into a sluice and then into the pond, or a ditch---when operating on dry ground. To aid in reclamation, this may be replaced with a sump for pumping the fines into the tailings piles of the oversized material from the trommel.

Amalgamation System

There seems to be little understanding today among many small operators of the use of mercury for amalgamation of gold. To have a secure, inexpensive means of recovery on a continuous basis on board the plant, a mercury amalgamation system is essential. Any time that the final concentrate has to be accumulated for transportation to a separate location to extract the gold, security becomes a problem and costs are compounded. The last phase of concentration should be in an enclosed area that offers some deterrence to theft. The jack pot, auger riffle and silvered copper-plate systems offer such a means of continuous processing that can be operated with a minimum of disruption and labor.

In the amalgam system, gold is recovered in three stages and becomes successively finer grained in each. Cleanup is made periodically with minimum disruption to the mining operation and with maximum security. At the same time, mercury is recovered and prevented from entering the disposal or tailings area. Coupled with this system is the handling of the amalgamated gold. This requires heating to evaporate the mercury, which passes in a vapor through a sealed hood and condensor into a container for recycling. The gold is left behind in sponge form for melting and pouring into a bullion bar for shipment to the refinery.

When considering each subsystem or component of the washing plant, the need to carefully analyze and choose the most cost-effective method of operation cannot be overemphasized. With the narrow range of profitability of a portable unit, anything that is introduced that compounds costs can easily tilt the balance and result in a losing operation. The handling of gold in the final recovery stage is very important to that aspect and can, in itself, spell the difference between profit or loss. Security ultimately demands special considerations in the design stage, since all else becomes subordinate if the gold is recovered only to be 'high graded.' A portable washing plant without major provisions for continuous recovery, coupled with ultimate security, is not addressing the vital elements of placer-gold mining.

OPERATIONS

Excavation System

Once we have settled on the proper design configuration of the recovery plant, the remaining decisions will involve the proper excavation and feeding equipment. The most successful of these in small-scale placer operations has been the backhoe. It is the best tool for cleaning bedrock, which is basic to placer mining since most values are found in the lower zone of the deposit. The backhoe can be supplemented as necessary with a dozer or front-end loader, or both. Other advantages of the backhoe are its uniform speed and ability to feed directly into the hopper of the plant, making it especially useful. Using 3/4- to 2-yd^3 buckets, volumes up to 200 yd^3/hr can be realized. The dragline, on the other hand, will usually require too large a bucket because it is slower than the backhoe; thus, it can dump too much material for the unit to handle. The front-end loader usually means a dryland operation and multiple handling of material and too large a bucket, as with the dragline. While at times this cannot be avoided, such as when working in narrow canyons, the value or tenor of the deposit must be high to carry the additional costs of operation. Another problem with this type of equipment is its hydraulic controls and repair rate when attempting to operate on a 24-hr basis.

Efficiency

With any placer operation, there is a hope to achieve 75 to 80 percent running time or better. This means sufficient backup spares and repair facilities to minimize downtime. This goal is essential to profitability and needs careful planning in the original layout of the logistical aspects of the operation. With these noble objectives stated, it is appropriate that I quote at this point Norman Cleaveland from his paper 'Ocean Mining Systems' presented in October 1967 at the Ocean Mining Symposium, Los Angeles:

Basic Lore of Gold and Tin Dredging

1. Each property is a problem in itself.
2. Most any dredging problem can be solved, except the no-gold or no-tin problem.
3. Partial solutions to problems are usually far more costly in the long run than proper solutions.
4. 'How will it function at 3 o'clock in the morning?' is the first question that should be asked about any proposed new design or procedure.
5. Prospecting with a dredge is a very extravagant method of prospecting.
6. As the old Burma Shave signs might have said: To make your old dredge really go, keep yardage high, bedrock clean, and losses low.

And a further quote from Cleaveland from the same symposium:

"Faulty prospecting and inadequate appraisal of prospecting results have been the major cause of dredge-mining failures in the past. It is difficult to overemphasize the importance of prospecting and the associated studies that should be made."

Personnel

Underlying the above comments is the importance of people in the overall equation. Where does one find experienced personnel in placer-gold dredging necessary to direct a profitable enterprise? This is a fast disappearing breed of men. But the woods are full of people who have had experience with unprofitable, small-scale placer-mining operations anxious to help someone do it the same way they did. Nothing is particularly new in this kind of problem since there were many more failures in placer-gold mining than successes when such operations were in their peak in the 1930s. What is lacking today is the source of experienced personnel that the successful operations used to produce. The only alternative is to train young men for placer work using the few knowledgeable personnel available.

Depending on the nature of the terrain, a major problem to solve in small-scale backhoe mining is the handling of an open pit operation, which is more akin to hardrock or sand-and-gravel operations. Avoiding a plan that will mean multiple handling of material must be considered essential, unless the deposit is particularly high grade; that is, over 300 mg/yd^3. Being able to float the washing plant and feed it from shore solves many problems. The tailings can be discharged into the pond, and the water, if kept clean, can be recirculated through the plant with minimal pumping. Maintaining the suction of a slime pump at the bottom of the pond will not only keep the water clean for recirculation but will aid in increasing production, since the material has to be removed anyway.

Few dryland placer-mining operations have proven to be profitable, but this is primarily from the past and during times when the backhoe had not yet been perfected. The fact remains, however, that difficulties are compounded when the plant is not floating and must, therefore, be carefully planned by experienced placer-mining engineers.

In Brazil, many floating portable washing plants exist today, mainly in the mining of tin, that are fed from shore by backhoe (fig. 5). Their profitability can be attributed to the richness of the deposits and the stabilizing of the price of tin. There is no reason to doubt that similar procedures, combined with the special circumstances of gold recovery not common to tin, can be applied to placer-gold operations in Alaska and Canada, given the shallow depths necessary for the backhoe.

CRITICAL QUESTIONS

When making the decision about placer-gold mining equipment for small-scale projects and attempting to evaluate the various products that are being promoted, the following questions should be asked:

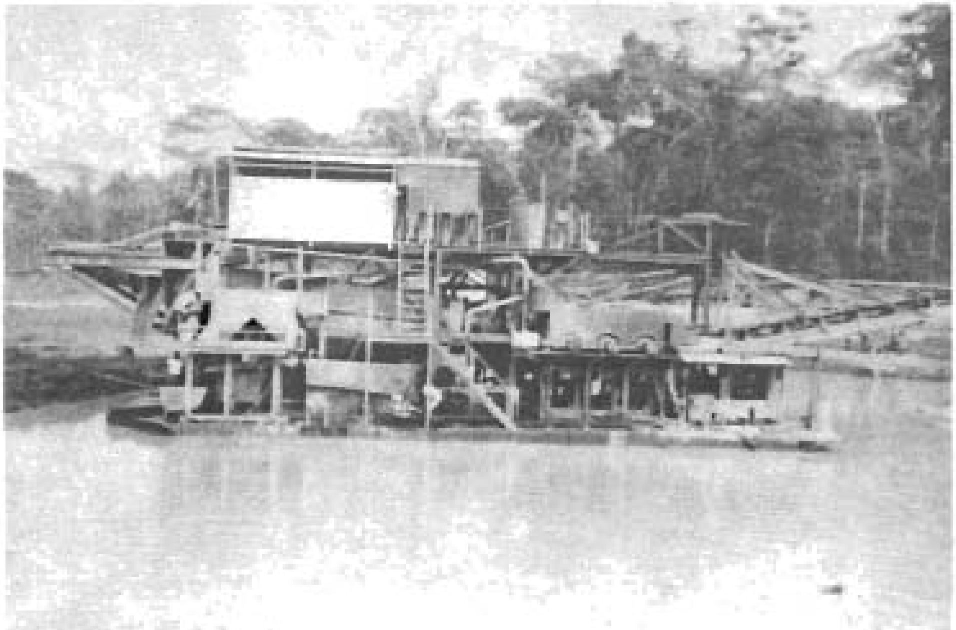


Figure 5. Floating washing plant for tin recovery operating in Brazil. Plant fed from shore by backhoe.

1. Can the system continually process over 50,000 yd^3/mo of material without significant interruption (that is, over 70-percent operating time)?
2. Will the disposal of oversize and sluice discharge be handled adequately and provide for reclamation with a minimum of multiple handling of material?
3. Is an amalgamation system included that provides continuous gold recovery and does not discharge mercury into the environment?
4. Can the system operate for < $1/$yd^3$ including excavation equipment, overhead, and G&A expense?

Answer the above questions successfully and you may be on track for a profitable mining project.


TOMORROW

CAN PEO SOLVE THE PLACER-MINING PROBLEM?

By
Bernie Schneider, Research Chemist
U.S. Bureau of Mines, Box L
University, Alabama 35486
and
Annie Smelley, Research Chemist
U.S. Bureau of Mines, Box L
University, Alabama 35486

The Bureau of Mines has been developing techniques for using commercially available water-soluble materials as flocculating agents. These techniques have been used on various ores in our laboratory at the Tuscaloosa Research Center. At the request of the Bureau of Mines and in cooperation with the Alaska Bureau of Mines office, we conducted a limited number of laboratory studies regarding the use of PEO (polyethylene oxide) on water from Alaska placer-gold mines. We took eleven water samples from the mines between July and October 1985 (table 1). The first five were taken at the end of the final settling pond but before the water went into the creek. The next four were taken out of the creek below the mine discharge. The tenth sample was taken at the end of the sluice and the eleventh from the dredge pond. Total suspended solids were determined on each of the untreated samples (table 2). Suspended matter ranged from 140 to 25,000 mg/l; turbidity ranged from 175 to 22,580 NTU (tables 2, 3, and 4). We used the same flocculation test on all of the Alaska samples (table 5).

In our laboratory tests, we measured 100 ml of the Alaska water into a 200 ml beaker. We then added PEO from a burret and stirred the solution with a magnetic stirring bar, maintaining about a ½-in. vortex. In most cases, big flocs formed. We continued adding PEO until the floc started to consolidate and finally settle. At this point we decanted the clear water. In every case, the mass consolidated to the point where it could be picked out of the water. We weren't able to do very many analyses of the solid material.

Flocculation tests were conducted on all samples to determine the best PEO concentration (table 6). The decrease in TSS (total suspended solids) and the increase in clarity were typical of all treated samples (table 7). The optimum dewatering condition, using PEO, was determined for each of the 11 samples (table 8) and for placer-mining effluent sampled in 1981 (table 9). Solids in the dewatered samples did not completely flocculate at first, but after a couple of weeks, settling was complete. Apparently conditions changed from anerobic to aerobic. During the dewatering tests, we added PEO in increments of 0.001 to 0.08 lb/gal. These amounts seem appropriate because we used them repeatedly with consistently good results. We also added calcium---the best two cation additives were Calfloc I (table 10) and Betz 1110 (table 11)---which reduced the amount of PEO required. In each test, we measured turbidity of the sample at 1- and 24-hr settling periods.

Table 1. Water samples taken from Alaska placer-gold mines during July - October 1985.

Sample[a]	Mining district	Location
1	Circle	Deadwood Creek
2	Circle	Crooked Creek
3	Circle	Gold Dust Creek
4	Fairbanks	Gilmore Creek
5	Hope	Resurrection Creek
6	Circle	Deadwood Creek
7	Circle	Mammoth Creek
8	Circle	Eagle Creek
9	Circle	Deadwood Creek
10	Hope	Resurrection Creek
11	Nome	Dredge-pond discharge

[a]Samples 1-5 were taken from the final settling pond; samples 6-9 were taken from the creek below the mine discharge; sample 10 was taken at the end of the sluice; and sample 11 was taken from the dredge pond.

Table 2. Suspended and settleable matter in untreated water samples taken from Alaska placer-gold mines during July - October 1985.

Sample[a]	Total suspended matter (mg/l)	Initial turbidity (NTU)	Settleable matter (ml/l)
1	9,850	19,000	2.0
2	1,300	3,100	0.05
3	140	175	1.2
4	4,610	11,700	0.1
5	1,720	3,260	0.3
6	3,100	4,750	1.4
7	1,280	750	1.7
8	2,000	3,750	1.4
9	290	520	1.0
10	25,100	22,800	25
11	4,320	16,000	110

[a]See table 1 for sample location.

Table 3. Results of standard settling test on water samples taken from Alaska placer-gold mines during July - October 1985.

Sample[a]	Total suspended matter (mg/l)	Nonsettleable matter (mg/l)	Settleable matter (ml/l)
1	9,850	1,230	2.0
2	1,300	1,200	0.05
3	140	50	1.2
4	4,610	4,500	0.1
5	1,720	1,670	0.3
6	3,100	2,250	1.4
7	1,280	280	1.7
8	2,000	1,380	1.4
9	290	80	1.0
10	25,100	8,270	25
11	4,320	160	110

[a]See table 1 for sample location.

Table 4. Turbidity of untreated water samples taken from Alaska placer-gold mines during July - October 1985.

	Turbidity		
Sample[a]	Initial (NTU)	1 hr (NTU)	24 hr (NTU)
1	19,000	9,000	1,950
2	3,100	1,940	468
3	175	66	36
4	11,700	6,420	2,700
5	3,260	1,485	540
6	4,750	2,325	702
7	750	918	49
8	3,750	810	657
9	520	133	39
10	22,800	4,890	1,368
11	16,000	270	6

[a]See table 1 for sample location.

Table 5. Mineral composition of water samples taken from Alaska placer-gold mines during July - October 1985.

Mineral	Sample[a] 1	2	3	4	5	6	7	8	9	10	11
Quartz	M	M	M	m-M	M	M	M	M	M	M	M
Chlorite	M	M	m	m	m-M	M	m-M	m	M	m-M	M
Mica and illite	m-M	m-M	m	m	m	m-M	m-M	m	m-M	m	M
Kaolinite	t	m-M	m	m-M				m			m
Plagioclase	m		t-m	t	m		m		t-m	m-M	
Mixed-layer clays	m-M	m				m			m		
Smectite				M	m-M						
Gibbsite		t-m									
Goethite			t-m	m				t-m			

[a]See table 1 for sample location.
M - Major
m - Minor
t - Trace

Table 6. Standard PEO dilution series.

Test	PEO Conc. (%)	PEO Dosage (lb/1,000 gal)	Turbidity (NTU)	Final solids content (%)
1	0.25	0.17	72	53
2	0.10	0.10	79	51
3	0.05	0.07	129	51
4	0.01	0.03	132	48
5	0.005	0.03	153	47
6	0.001	0.02	105	44

Table 7. Turbidity (NTU) of untreated water samples vs. treated water samples at 1- and 24-hr settling periods. Samples taken from Alaska placer-gold mines during July - October 1985.

	1 hr		24 hr	
Sample	Untreated	Treated	Untreated	Treated
1	9,000	42	1,950	3
2	1,940	60	468	37
3	66	30	36	6
4	6,420	66	2,700	49
5	1,485	57	540	31
6	2,325	28	702	9
7	918	15	49	4
8	810	27	657	11
9	133	2	39	5
10	4,890	66	1,368	44
11	270	27	6	4

[a]See table 1 for sample location.

Table 8. Best results of dewatering tests on water samples using PEO. Samples taken from Alaska placer-gold mines during July-October 1985.

	PEO		Turbidity	
Sample[a]	Conc. (%)	Dosage (lb/1,000 gal)	1 hr (NTU)	24 hr (NTU)
1	0.01	0.03	69	30
2	0.001	0.02	66	--
3	0.005	0.02	78	27
4	0.05	0.08	102	66
5	0.01	0.03	150	75
6	0.01	0.02	48	21
7	0.001	0.01	48	21
8	0.01	0.02	67	34
9	0.001	0.02	60	24
10	0.005	0.02	186	75
11	0.001	0.01	30	6

[a]See table 1 for sample location.
-- No data

Table 9. Best results of dewatering tests on placer-mining effluent sampled in Alaska in 1981.

Description	Initial solids content (%)	PEO Conc. (%)	PEO Dosage (lb/1,000 gal)	Turbidity (NTU)	Final solids content (%)
Livengood Creek, below sluice discharge	1.21	0.05	0.07	--	23
Livengood Creek, below gate from tailings settling pond	0.17	0.01	0.005	79	28
Miller Creek, sluice water	0.40	0.01	0.005	93	39
Jack Wade Creek, east of Chicken, sluice water	0.58	0.001	0.017	90	22
Sourdough Creek, sluice water	1.73	0.005	0.013	63	35

-- No data

Table 10. Results of dewatering tests on water samples using PEO and Calfloc T. Samples taken from Alaska placer-gold mines during July - October 1985.

Sample[a]	Calfloc T Conc. (%)	Calfloc T Dosage (lb/1,000 gal)	PEO Conc. (%)	PEO Dosage (lb/1,000 gal)	Turbidity 1 hr (NTU)	Turbidity 24 hr (NTU)
1	0.05	0.11	0.01	0.02	42	3
2	0.01	0.04	0.001	0.01	60	37
3	0.01	0.02	0.005	0.02	36	7
4	0.01	0.06	0.05	0.09	66	49
5	0.01	0.02	0.01	0.01	57	31
6	0.01	0.02	0.005	0.02	33	13
7	0.01	0.04	0.001	0.01	15	4
8	0.01	0.02	0.05	0.10	27	11
9	0.01	0.03	0.001	0.01	2	5
10	0.01	0.08	0.005	0.05	66	44
11	0.01	0.01	0.05	0.06	27	4

[a]See table 1 for sample location.

Table 11. Results of dewatering tests on water samples using PEO and Betz 1180. Samples taken from Alaska placer-gold mines during July - October 1985.

Sample[a]	Betz 1180 Conc. (%)	Betz 1180 Dosage (lb/1,000 gal)	PEO Conc. (%)	PEO Dosage (lb/1,000 gal)	Turbidity 1 hr (NTU)	Turbidity 24 hr (NTU)
1	0.01	0.03	0.01	0.03	60	3
2	0.01	0.04	0.001	0.01	75	40
3	0.01	0.04	0.005	0.04	30	6
4	0.01	0.03	0.005	0.03	105	60
5	0.01	0.02	0.01	0.01	60	31
6	0.01	0.05	0.005	0.02	28	9
7	0.01	0.03	0.001	0.01	18	5
8	0.01	0.03	0.05	0.08	43	17
9	0.01	0.03	0.001	0.01	15	8
10	0.01	0.02	0.005	0.06	75	36
11	0.01	0.04	0.005	0.04	18	24

[a]See table 1 for sample location.

1986 FIELD SEASON

This summer, we plan to conduct field tests at four different mine sites in Alaska. We are presently negotiating with mines in Cripple Creek, Fairbanks, Livengood, and one on the Kenai Peninsula, and assuming that no one objects, we will be here this summer. We will be using a mobile plant that consists of an 8 by 8 screen and necessary pumps, mixing tanks, and PEO make-up equipment (fig. 1). We have not been able to test the screen at our Tuscaloosa lab because of a lack of adequate sample material. In particular, we are uncertain as to how fast material will move down the screen. The type of solid will influence this action. We have made special provisions to change screen inserts. Changing the size of the screen openings will allow us to cover all bases before we start.

An important point to remember is that flocculated material can be mixed with coarse material, such as gravels; for example, we have mixed it with small-size refuse from a coal plant. The dewatered refuse was easy to discharge. This may be a benefit for placer operations and lands reclamation.

Water flowing through the screen always contains some solids. In our plant, the solids will flocculate and settle rapidly. We will monitor the solids and determine what happens when they come off the screen at a certain percent solid, say 20 to 40 percent and possibly up to 70 percent. We will allow a few solids to come through the screen, so that they can be used in PEO makeup. Water from the screen underflow will go to a small settling pond, where it will be recycled back to the sluice box and added to the miner's water system.

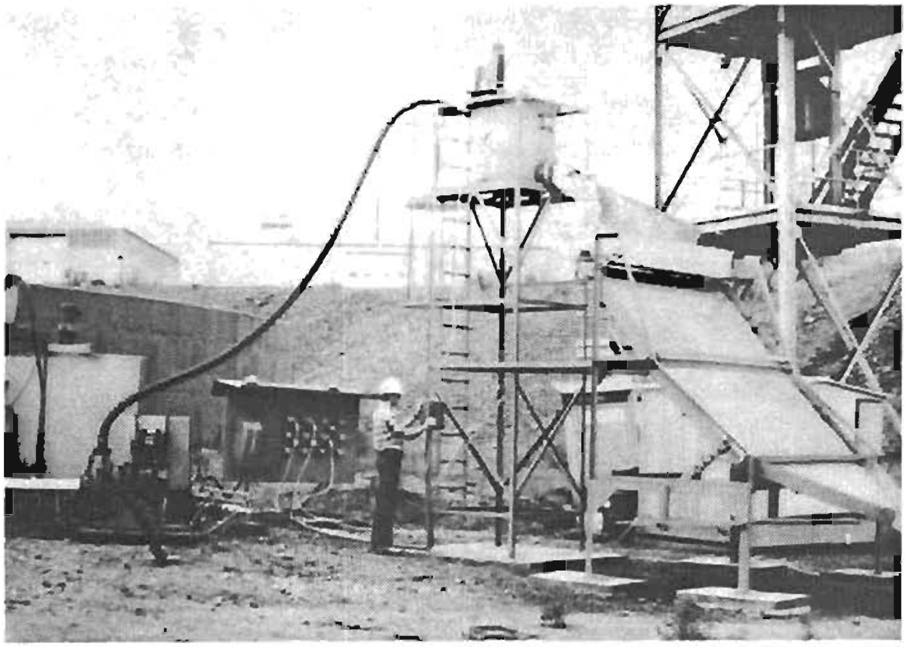


Figure 1. Mobile plant.

Dirty water from the sluicing operation is taken from a sump and pumped to a mixing tank for PEO makeup. The tank contains a stirrer, and we add the solid PEO base to water. The rest of the mobile plant consists of blending tanks.

When we go to a mine site it is essential that we set up our tanks and start operating in a day or so. Potentially, we will be a self-contained unit, complete with a generator and an electric panel. The only thing we will ask of the miners is that they set up a small pit (about 100 ft^2) to collect our water. PEO sells in this country for $5.12/lb and in Japan for $2.73/lb. The Alaska tests will use PEO that costs $2.73/lb FOB Fairbanks. We have talked to salesmen, however, who are willing to sell PEO at $0.65/lb, if bulk quantities are purchased.

QUESTIONS AND ANSWERS

(Q): What is the environmental effect of PEO?
(A): I have a report with me that lists all the FDA requirements and FDA laws. PEO has passed as biodegradable and is acceptable from an environmental standpoint. In fact, PEO is approved by the FDA for use in food consumption.

(Q): What is the effect of water temperature on flocculation?
(A): We have run successful tests in Kentucky when the temperature was 33°F.

(Q): What is the effect of PEO on the pH of water?
(A): There is no change in pH.

(Q): Do tests show any effect on metals? Did you ever dry the sediment and check it for metal content?

(A): We never tested placer sediment for metals because we had such a small amount of sediment that we wouldn't have had enough for the analysis. We do, however, plan to test the effects this summer. Arsenic content has no effect on pH; therefore, it won't have any effect on the flocculant. We have, however, done some testing where we added lime or Calfloc T, and this has a tendency to drive the PEO dosage down. Therefore, if the PEO concentration is too high, we might add lime, but at this point it is not in the plans.

(Q): Are you going to use an active mining operation for your demonstration work?

(A): We will come onto an active property and take water from the raceway between the sluice box and the first pond, so we will be taking a part of the operator's production. The capacity of the mobile plant is unknown at this time, but we do know that we have been able to handle from 400 to 600 gal/min. Let's say we did get up to 800 gal/min. If the overflow is 1,600 gal/min, then we would be taking half of it. Our pond will be completely isolated from the miner's pond. We will know exactly what effect we are having without trying to guess how we are doing. We will be taking on a real site, taking real water, real slurry, and treating it as a real miner would.

(Q): Are you going to procure waivers from DEC so the miner is not subject to a violation?

(A): The miners we will be working with will have a waiver for us to operate. We will have no effect upon the miners. If we mess up their water, they are not liable.

(Q): What will happen during the winter at 60 and 70°F below zero?

(A): Freezing helps to consolidate the sludge.

(Q): Please discuss operating and capital costs of a PEO plant.

(A): If we use 0.01 to 0.02 lb PEO/1,000 gal, the cost for PEO would be $0.02 to $0.06/1,000 gal. Equipment cost is $20,000. We used a 55-gal drum instead of a highly controlled mixer at a commercial operation in North Carolina. When a cheaper screen is used, the plant could be built at a lower cost. It would be the same technically, but the equipment would be different.

(Q): How long do you plan on spending at each mine this summer?

(A): About three weeks.

(Q): Do you think five NTU is a suitable standard?

(A): You have to know the background reading of a particular stream. If the background is zero, you are correct. It is a feasible criterion. It also depends on what part of the stream you are using. In addition, if you are discharging 50 percent of the stream flow, you are going to have cleaner water than if you are discharging only 10 percent of the stream flow.

(Q): Being biodegradable, if you put sludge on a tailings pile, will it grow grass? If it gets into the stream bed, would this block a stream bed worse than turbidity would?
(A): In certain areas, like Florida, you can put sludge on a tailings pile, go home for 3 wk, come back, and cut down a 'jungle.' It should not change the quality of the stream bed.

(Q): Will the flocculant sludge be allowed to set up at the end of settling ponds or placed in the tailings?
(A): Due to space limitations, we feel the maximum benefit is to mix the sludge with the tailings.

(Q): Would you say that in 10 yr you would be able to build a home on this floc where it has settled down and have a well put down where this floc is?
(A): Absolutely.

(Q): What does it taste like?
(A): Extensive studies of the effect of PEO on fish have been conducted by Union Carbide. The study shows no effect on the fish.

(Q): Are there any PEO effects on iron, arsenic, or other similar metals?
(A): We have not addressed any of these problems.

(Q): Is there a correlation between TSS and turbidity?
(A): We were not able to do these tests because of our limited samples, but we expect to do some studies on those relationships.

(Q): Will samples in the field be better than lab tests?
(A): Field-sample results are usually much better, but until placer-water field tests are run, we will not know.

(Q): What did you find when you ran tests on Gilmore Creek waters?
(A): It was one of the more difficult ones. We did get some more turbidity. Yes, we can settle material from this water, but you may not be able to afford it!

(Q): In the Allied Chemical tests, the reagent preparation water had to be crystal clear before adding the flocculent agent, is your system the same?
(A): We use recycled water and have had no problems with this in the past.

(Q): Will you have problems with pumps?
(A): I don't think so.

(Q): Can you clean up Gilmore with your system?
(A): Yes, we can clean it up, but it would be costly. We would just have to run tests and see what we would get out of the final settling ponds.

SEDIMENTATION WITH CENTRIFUGAL SEPARATION IN A RECYCLE PROCESS

By
J.A. Chapman, Graduate Student
Department of Environmental Quality Engineering
University of Alaska-Fairbanks 99775
and
R.A. Johnson, Professor
Department of Environmental Quality Engineering
University of Alaska-Fairbanks 99775

INTRODUCTION

The regulatory climate in the Alaska placer-mining industry is in a state of flux. The mining community is not satisfied with the present NPDES (National Pollutant Discharge Effluent Standard) mining permit because they feel that the turbidity standard is too strict. Although some parties would like to see the standard reduced, most environmental and Native groups are concerned with the degree of enforcement of current standards. Court hearings on the Federal water-quality standards were held in Fairbanks in February 1986.

State standards for sediment and turbidity discharge to streams are as follows (Alaska Department of Environmental Conservation, personal commun., 1986): 1) there shall be no noticeable increase in sediments, including settleable solids, above natural conditions in the receiving stream; 2) turbidities in the receiving stream shall not exceed 5 NTU above natural conditions when natural conditions are < 50 NTU; and 3) turbidities may exceed natural conditions by as much as 10 percent to a maximum of 25 NTU when natural conditions are > 50 NTU. State standards are applied after the effluent has mixed with the receiving stream, commonly considered to be 500 ft downstream of the discharge. The current NPDES permit stipulations are somewhat less complicated in that they are applied to the mine effluent. To comply with the Federal stipulations, settleable solids must be < 0.2 ml/l; turbidities, < 5 NTU.

Settling ponds are currently the most common treatment technique used by placer miners to meet water-quality standards. Recycling, however, has been proposed as a replacement for once-through settling, because it can reduce the amount of effluent. In many industries, solids are removed from a liquid stream through the use of a hydrocyclone. In this paper, we evaluate a water-treatment system that combines all three techniques without the use of chemical coagulants or flocculants.

SETTLING THEORY

Soil particles may be removed from placer waters by allowing them to settle through the water column. The velocity at which a given particle settles is dependent on the size, shape, roughness, and density of the particle and on the viscosity and density of the fluid. This is true for the settling of discrete particles in a quiescent fluid.

The terminal settling velocity for a sphere can be described by equations of classical fluid mechanics (Weber, 1972). These equations lead to the relation for terminal velocity:

$$V = [(4g/3C_D)((P_s - P_l)/P_l)d]^{.5} \quad (1)$$

where

V = Terminal velocity (cm/s)
g = Gravitational constant (980 cm/s)
C_D = Drag coefficient [C = 24/Re + 3/Re0.5 + 0.34, for RE < 10,000; Re = $(dP_l V)/\mu$; and μ = absolute viscosity (g/cm/s)]
P_s = Density of solid (g/cm)
P_l = Density of liquid (g/cm)
d = Diameter of particle (cm)

Where laminar flow prevails, at Re < 1, the relationship between C and Re is

$$C = 24/Re$$

and the terminal settling velocity simplifies to

$$V = (g/18\mu)(P_s - P_l)d^2 \quad (2)$$

which is the commonly cited Stokes' law. Stokes' law can generally be applied to soil particles that have a diameter of about 100 microns or less.

Using the settling velocity of a particle, one can determine whether the particle will be removed in a quiescent pond. For complete removal of particles of a given size, their settling velocities must be greater than the surface overflow rate (SOR) of the pond (Weber, 1972):

$$SOR(cm/s) = Q/A$$

where

Q = Volumetric flow rate through pond
A = Surface area of pond

Particles that settle at a velocity less than the surface overflow rate will be removed fractionally in proportion to the ratio of their settling velocity to the surface overflow rate. This assumes that particle sizes are distributed evenly throughout the water column at the pond inlet. The total removal of a heterogeneous distribution of particles in a pond can then be estimated by the equation (Weber, 1972)

$$R = (1 - f) + \sum_{i=1}^{n} (V_i/SOR)x_i \quad (3)$$

where

R = Total fraction removed
f = Fraction of particles with settling velocity < SOR
V_i = Settling velocity, < SOR (cm/s)
x_i = Fraction of particles with settling velocity Vi

It is possible to introduce significant error in calculated settling velocities unless shape differences are taken into account (Brown, 1949). If particle sizes of minus 200 mesh (< 74 microns) are determined from a hydrometer test, which is commonly the case, a size is obtained that relates to a sphere that settles according to Stokes' law. This minimizes the error introduced by varied shapes and densities when calculating removals. For plus 200-mesh particles (> 74 microns), the effect of particle shape would not have to be calculated, because the pond would generally be sized to remove particles much smaller than 200 mesh.

HYDROCYCLONE

Hydrocyclones are conical-shaped devices that are tangentially injected with a fluid under pressure. The tangential injection causes a rotation of the fluid and high centrifugal forces in the vessel. The centrifugal forces cause particles that have a greater density than the carrier fluid to radiate outward. A fraction of the inlet flow concentrated with the solids exits as the underflow of the cyclone; cleaner fluid moves inward and exits as the overflow.

As the diameter of the hydrocyclone decreases, its ability to separate smaller particles increases. It was the intent of this study to evaluate a hydrocyclone that was designed to remove the smallest particles capable with known technology without the addition of coagulants or flocculants. A 12-mm cyclone was chosen with a maximum flow rate of 1 gal/min at a 70 psig (pounds per square inch, gage) pressure drop. Although large banks of cyclones are used in some industries, this may not be economical for placer-mining operations. Larger sized cyclone systems may find other treatment schemes more useful than the one described in this investigation. Treatment schemes that use cyclones in combination with flocculating agents have been investigated by Lin (1979).

The intrinsic separation efficiency (eff) for the hydrocyclone was presented by Johnson and Lindley (1982):

$$eff = (e - R)/(1 - R)$$

where

e = Mass of particles in underflow divided by mass in feed
R = Ratio of underflow to feed flow rate

It is a measure of the ability of a cyclone to separate over and above that which is attributable to hydrodynamic entrainment.

TEST FACILITY

The test loop consisted of two settling basins and a Dorr-Oliver 12-mm hydrocyclone in series (fig. 1). A vibratory feeder delivered paydirt at a controlled rate to a shaker box, where spray jets washed the soil. The shaker box was lined with 12-mesh screen so that plus 12-mesh material (> 1.68 mm) did not enter the first basin but was removed as tailings. Water flowed through the first basin and entered the second one; total flow was

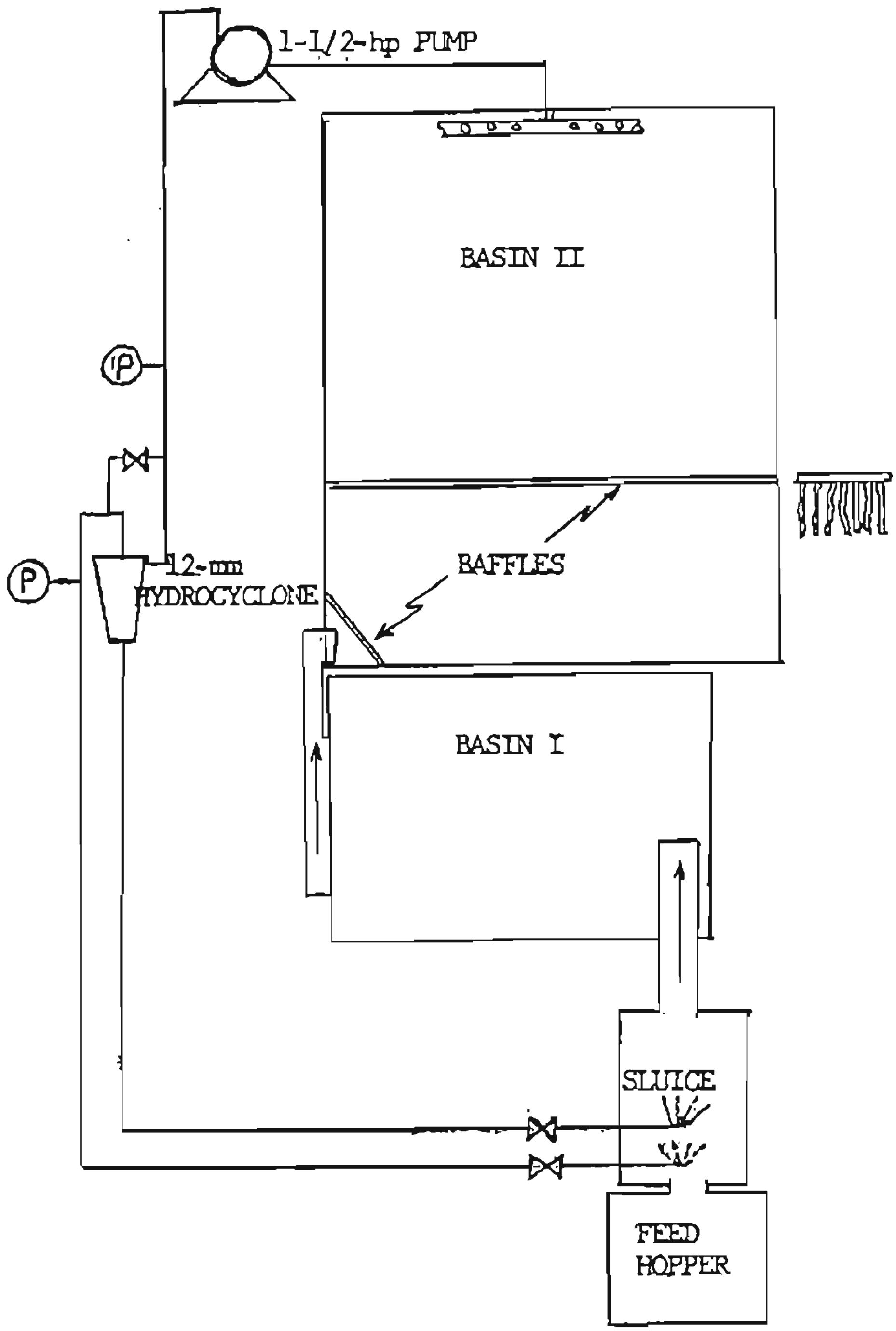


Figure 1. Schematic diagram of test loop.

circulated with a 1- to 1½-hp centrifugal pump from the second basin and either passed through or bypassed the hydrocyclone. When data were collected on the hydrocyclone, overflow and underflow were returned to the spray jets. The system was operated at 100-percent recycle with a water duty (defined as the cubic yards of paydirt washed per 1,000 gal of water) of 0.11 to 0.15, or a feed rate of 0.3 to 0.4 lb/gal.

By using a laboratory-scale facility, we were able to perform tests much more economically than by conducting full-scale investigation in the field. In addition, our results provide a more realistic simulation of field conditions than results from quiescent settling conducted in a batch mode.

TEST PROCEDURES

Sieve analysis and hydrometer tests were performed on the paydirt to determine particle-size distribution. Turbidity, total solids, total suspended solids, settleable solids, water temperature, and pH measurements were performed in accordance with standard methods on water samples taken at various points in the system. Viscosity was measured with a rotating spindle Brookfield viscometer. Particle-size distribution of sediments deposited in the second pond was obtained by a hydrometer test and by a rapid sediment analyzer. Sediment loadings on the basins were measured volumetrically.

Three runs were performed. Runs 1 and 2 were conducted with soil that contained 4-percent clay. Run 1 lasted for four recycles; Run 2, for six. Run 3 was conducted with a soil that contained 6-percent clay, and the test lasted for four recycles.

RESULTS AND DISCUSSION

Settleable solids results are presented in table 1. Basin II had a surface overflow rate of 120 gal/day/ft^2, which is sufficient to remove 10 micron particles under ideal settling conditions. Photomicrographs of the effluent showed that the largest particles leaving the basin were 20 to 25 microns. Analysis of the basin sediments show that 100 percent of the material was finer than 20 to 25 microns and 15 to 20 percent was finer than 10 microns.

Table 1. Settleable solids results for experimental water-treatment system.

	Effluent			Hydrocyclone		
Test	Sluice (ml/l)	Basin I (ml/l)	Basin II (ml/l)	Feed (ml/l)	Overflow (ml/l)	Underflow (ml/l)
1 (4% clay)	10.0	3.1	0.8	0.8	<0.1	2.1
2 (4% clay)	7.6	2.1	0.5	0.5	<0.1	1.1
3 (6% clay)	11.0	N/A	1.3	1.3	<0.1	2.5

N/A - Not applicable.

Total suspended solids and turbidity levels in the recycled water reached steady-state values in the first one to two recycles (figs. 2 to 5). The increase in values for run 3 (figs. 3 and 5) after the second recycle is probably due to an increase in feed rate. Some of the spikes and minima in these curves, particularly for the influent values, are caused by an unsteadiness in the paydirt feed rate. Note how these fluctuations are damped by the time the flow reaches the effluent basin. Moreover, such unsteadiness is not entirely undesired since it simulates field conditions. The steady-

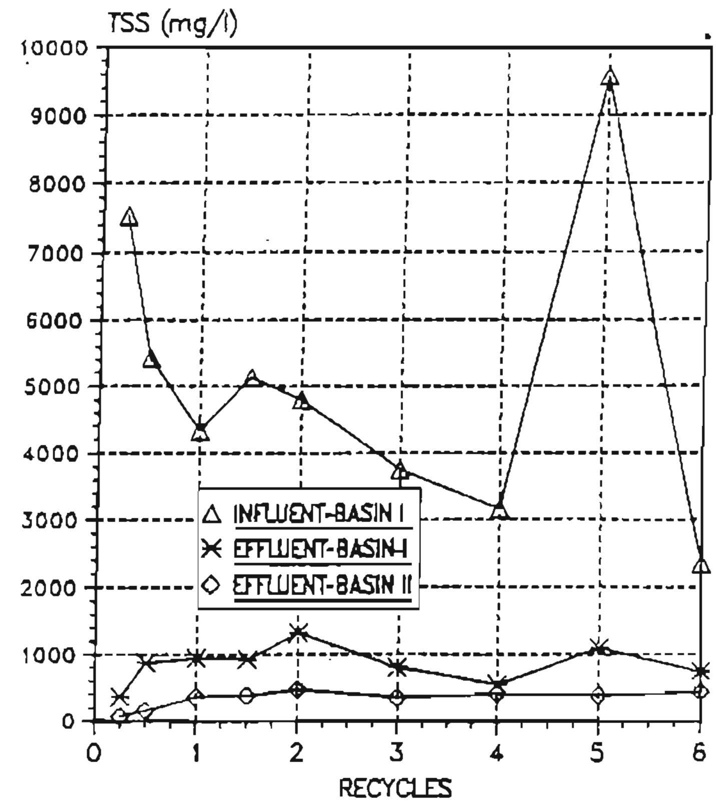


Figure 2. Total suspended solids (TSS) vs. recycle, Run 2 (4-percent clay).

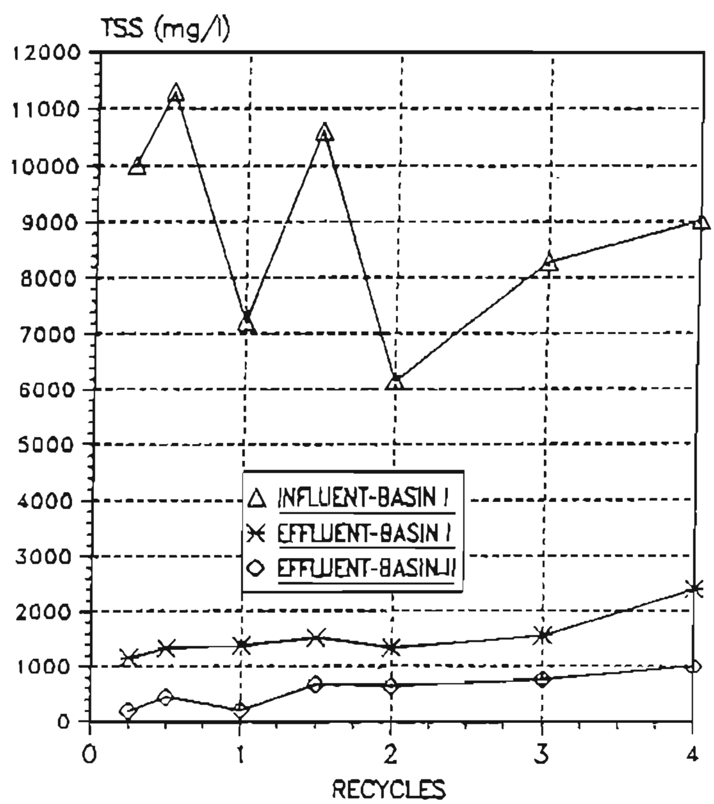


Figure 3. Total suspended solids (TSS) vs. recycle, Run 3 (6-percent clay).

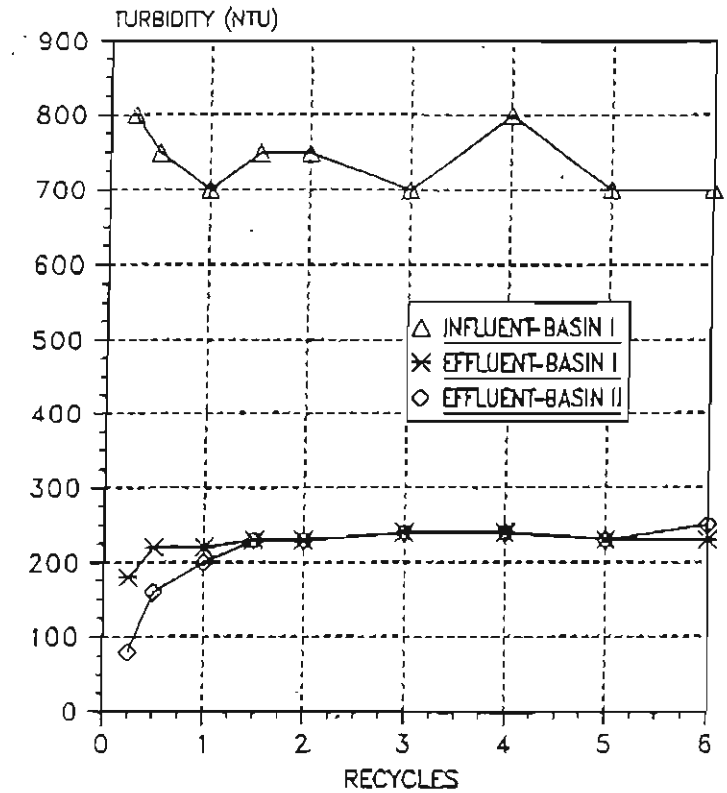


Figure 4. Turbidity vs. recycle, Run 2 (4-percent clay).

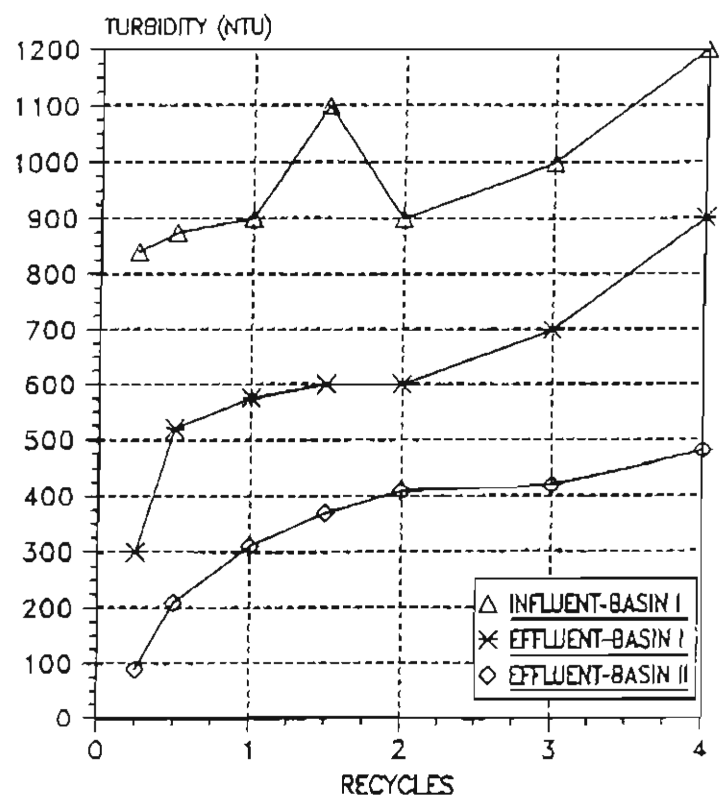


Figure 5. Turbidity vs. recycle, Run 3 (6-percent clay).

state values were higher for the soil containing 6-percent clay than for the soil containing only 4-percent clay, indicating that total suspended solids and turbidity are dependent on the percent fines in the soil.

If flocculation were not occurring, one would expect the total suspended solids levels to increase with the number of recycles. Data presented in figures 2 and 4 show that this is not the case. To estimate the amount of flocculation, we plotted the difference between total solids and total suspended solids (figs. 6 and 7). This difference represents essentially the colloidal plus dissolved matter and is found not to increase with time. If flocculation were not occurring, we would expect the difference to increase with time, as indicated by the curves labelled 'predicted.' The fact that this increase does not occur leads one to suggest that flocculation is occurring. We should note that about 300 mg/l of the total solids minus total suspended solids value can be attributed to the hardness of the laboratory water used in the runs. No measurable increase in viscosity was observed for either of the two paydirts during the course of the runs.

Removal efficiencies for the hydrocyclone are presented in table 2. The hydrocyclone was very effective in removing settleable solids but had a lesser effect on total suspended solids and an even smaller effect on turbidity. The settleable solids levels of the overflow were always less than 0.1 ml/l (table 1); however, turbidities were in the range of 200 to 400 NTU, well above the 5 NTU standard.

Table 2. Removal efficiency of hydrocyclone in experimental water-treatment system.

Test	Total Suspended solids (%)	Settleable solids[a] (%)	Turbidity (%)
1 (4% clay)	11	100	21
2 (4% clay)	41	100	19
3 (6% clay)	57	100	42

[a]Settleable solids levels of the hydrocyclone overflow were < 0.1 ml/l.

The 12-mm hydrocyclone is designed to remove smaller particles than the settling basins and represents optimal cyclone technology for fine particle removal. Photomicrographs of the hydrocyclone overflow show significant amounts of minus 5-micron particles, which are too small to separate effectively even with the very small cyclone tested. The fact that water quality standards are not achieved in the hydrocyclone overflow demonstrates that hydrocyclones alone will not be effective as a final cleanup step for placer mines with significant clay content in the soil. They could, however, be used in conjunction with other technologies to replace settling ponds in areas where the local topography precludes the use of the ponds.

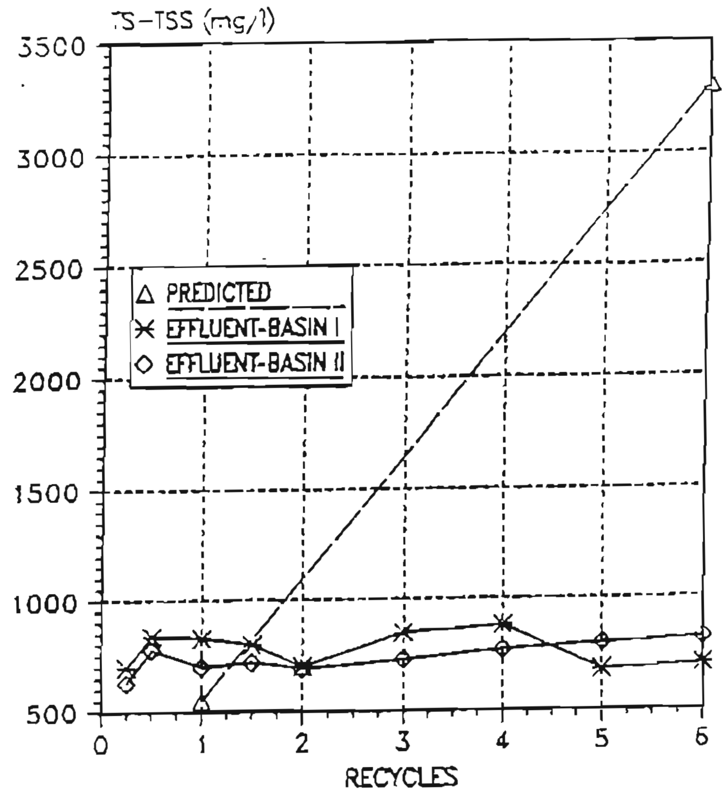


Figure 6. Total solids (TS) minus total suspended solids (TSS) vs. recycle, Run 2 (4-percent clay).

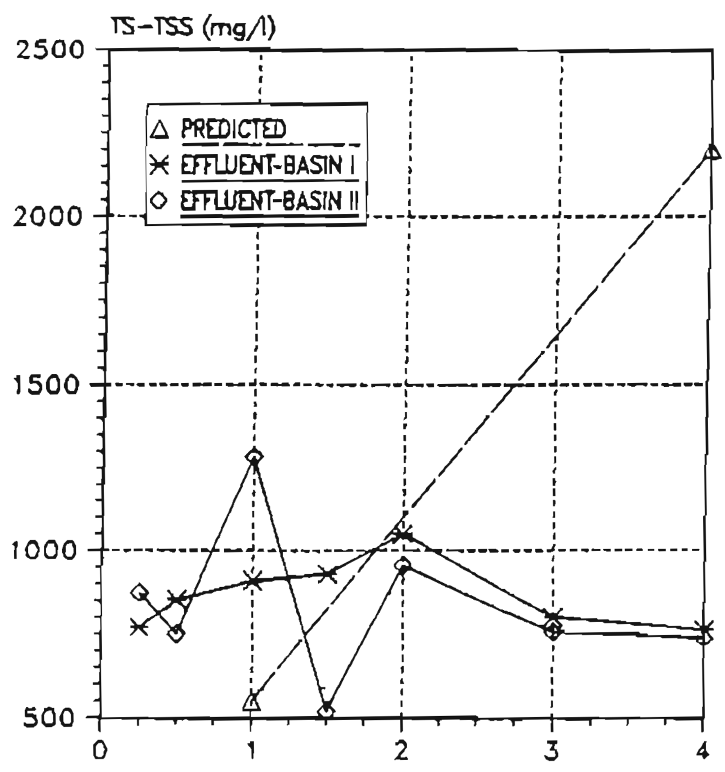


Figure 7. Total solids (TS) minus total suspended solids (TSS) vs. recycle, Run 3 (6-percent clay).

A computer model, based on equation (3), was developed to predict removal efficiencies of suspended solids. It incorporated ideal settling and a heterogeneous distribution of particles. Modifications to this model, based on empirical observations of short-circuiting, resulted in a predicted particle size in the overflow from the second settling basin of 20 microns.

Results from our computer model were compared with field results collected by Shannon and Wilson at the Gilmore mine site, July 1984, and with our laboratory results (table 3). The model is not accurate enough to predict final effluent quality of a given paydirt and pond system, but it can be used to determine pond sediment loadings. Figure 8 shows the configuration of sediments deposited in the first basin at the conclusion of Run 1 (4-percent clay); sediments were distributed equally over the second basin floor and were < 0.5 cm thick. It was estimated, by determining the volume of sediment in each basin at the end of Run 1, that 94 percent of the particulates removed in the run settled out in the first basin. The computer predicted that 97 percent of the particulates would settle out in the first basin.

Table 3. Computer prediction vs. laboratory and field results of total suspended solids (%) removed from process water in experimental water-treatment system.

Study	Total suspended solids removed: Observed (%)	Total suspended solids removed: Predicted (%)
Test 1 (4% clay)	84	93
Test 2 (4% clay)	91	93
Test 3 (6% clay)	90	91
Gilmore mine[a]	86-98[b]	97

[a]The surface overflow rate used for the multipond system at the Gilmore site was based on the pond with the lowest surface overflow rate, excluding the first pond because it was a recycle pond.
[b]Dependent on method of calculation.

Deviations from ideal settling can be caused by several factors, including short-circuiting, wind, infiltration, seepage, and thermally induced currents. The degree of short-circuiting in the laboratory ponds was determined with dye tests (Weber, 1972). Simple baffles decreased the amount of short-circuiting significantly. The measured detention times were 6 percent and 14 percent of the theoretical detention time for unbaffled and baffled, respectively. Improved pond flow patterns should decrease the maximum size of particles leaving a pond and are likely to improve overall removal.

Wind can generate waves on a pond and vertically mix the water column, which suspends particles of a certain size. Upward velocities of wind-generated orbital currents can be calculated using the U.S. Army Coastal Engineering Center's approximations (Jokela and others, 1983). For a 10 knot wind blowing over a 250 m pond, wave forecasting curves yield a wave height

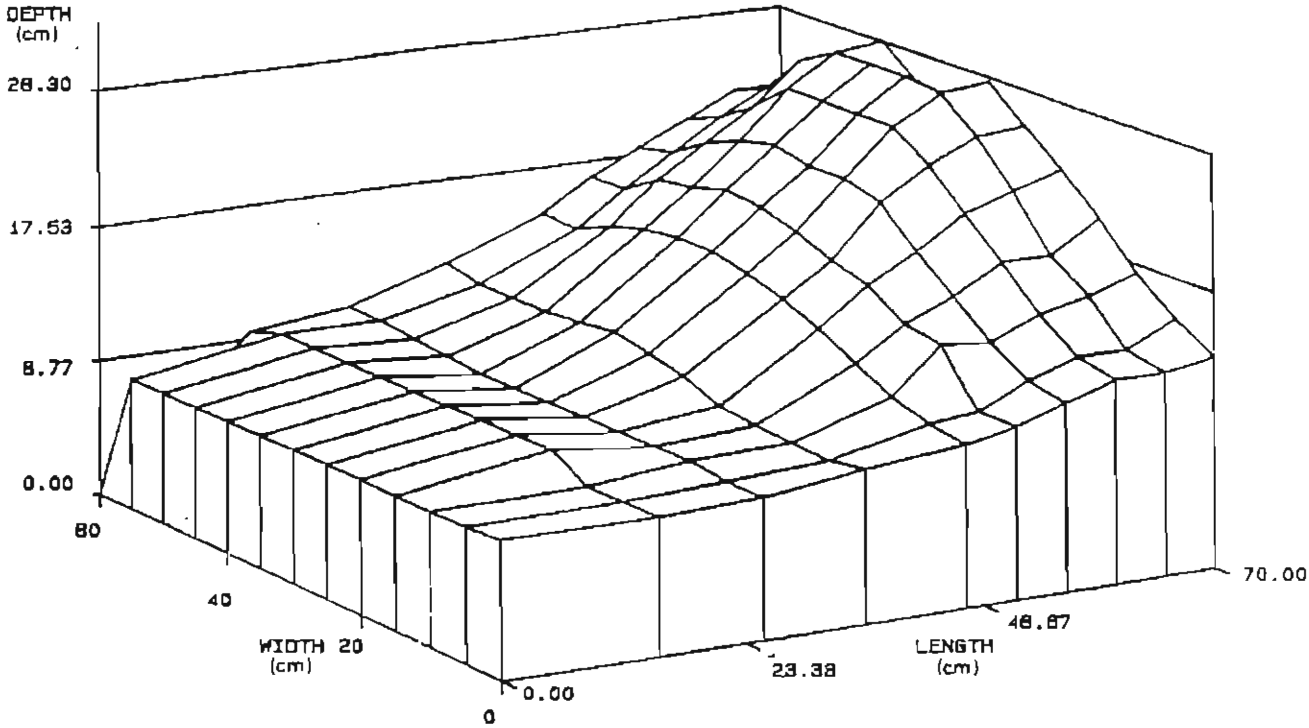


Figure 8. Sediment buildup in Basin 1 at end of Run 1 (4-percent clay). View from right-hand side of overflow.

of 3.6 cm and a wave period of 0.75 s (Wiegel, 1964). An associated deep-water wave length for this wave would be

$$L = gT^2/2\Pi$$
$$= 0.9m$$

where T is the wave period. The diameter of the wave orbit at 1 m depth is

$$2b = H\ \exp(2\Pi y/L)$$
$$= 3.6\ \exp(2\Pi(-1)/.9)$$
$$= 0.0033\ \text{cm}$$

where H is the wave height and y is the depth. The resulting orbital velocity is

$$2\Pi B/T = 0.014\ \text{cm/s}$$

Velocities of this magnitude would suspend a 13-micron particle. At a depth of 0.5 m, the orbital velocity would be 0.45 cm/s, which would suspend a 75-micron particle.

Field data from the Gilmore mine site (Shannon and Wilson, Inc., 1985) show that particles with diameters of 20 to 35 microns leave with the final pond effluent. Even with severe short-circuiting, estimated by using a surface overflow rate 20 times the theoretical rate, 100-percent removal of

12-micron and larger particles would be expected. The presence of particles in the effluent two to three times this size could be explained by the orbital velocities of wind-generated waves. This implies that the quality of the effluent in the final pond may be dependent on weather conditions.

The size of the wind-generated waves and the magnitude of the associated orbital velocities are dependent on the wind speed and fetch. The fetch, or the length of unobstructed water subjected to wind, could possibly be reduced by the installation of baffles that extend above the water surface.

SUMMARY AND CONCLUSIONS

Recycling had no effect on the viscosity of the 4- and 6-percent clay-content soils, which has important implications for gold recovery. Total suspended solids and turbidity did not continue to increase with recycle but did seem to be affected by water duty and clay content. The concentration of submicron particles also remained constant during recycle. This indicates that coagulation or flocculation was occurring naturally. Hydrocyclone technology, when used as a final treatment step without coagulants or flocculants, can meet NPDES stipulations for settleable solids, but it is unlikely that turbidity standards can be met if the paydirt has a significant clay fraction. Wind and short-circuiting are limiting factors on the performance of settling ponds. Simple baffling, however, can reduce short-circuiting significantly.

ACKNOWLEDGMENTS

Funding for this study was provided by the U.S. Geological Survey through the Institute of Water Resources, University of Alaska, Fairbanks (UAF). Special thanks are due to David Maneval (UAF) and the Mineral Industry Research Laboratory (UAF) for providing equipment and working space and to the Environmental Quality Engineering Department (UAF) for providing guidance and facilities. We would also like to express our gratitude to Mr. Cacey Patton (Bedrock Mining Company), who provided us with paydirt and a commentary on the placer miner's situation.

REFERENCES CITED

Brown, C.A., 1949, Sediment transportation in engineering hydraulics: New York, John Wiley, p. 770-855.

Johnson, R.A., and Lindley, K.A., 1982, Use of hydrocyclones to treat seafood-processing waste waters: Journal of Water Pollution Control Federation, v. 54, no. 12, p. 1607-1612.

Jokela, J.B., Bredthauer, S.R., and Coffin, J.H., 1983, Sedimentation in glacial lakes: Cold Regions Environmental Engineering Conference, 1st, Fairbanks, Alaska, 1983, Proceedings, p. 124-148.

Lin, H.K., 1979, Application of hydrocyclones for the treatment of waste water in gold-placer mining: Fairbanks, University of Alaska, unpublished M.S. thesis, 97 p.

Shannon and Wilson, Inc., 1985, Placer Mining Waste Water Treatment Technology Project, phase 2: Juneau, Alaska, unpublished report for Alaska Department of Environmental Conservation, 148 p.

Weber, W.J., 1972, Physiochemical processes for water-quality control: New York, John Wiley, 640 p.
Wiegel, R.L., 1964, Oceanographical engineering: New Jersey, Prentice-Hall, 532 p.

A MICROCOMPUTER-AIDED SCHLUMBERGER SOUNDING TECHNIQUE FOR PLACER-GOLD EXPLORATION

By
S.L. Huang, Associate Professor
School of Mineral Engineering
University of Alaska-Fairbanks
Fairbanks, Alaska 99775
and
R.C. Speck, Associate Professor
School of Mineral Engineering
University of Alaska-Fairbanks
Fairbanks, Alaska 99775

INTRODUCTION

Gold mining, the foundation of Alaska's heritage and economy, has played a major role in the exploration and development of the state. It's importance after World War II, however, declined due to rising operating costs until, during the 1960s, nearly all gold dredging was discontinued. In the 1970s, removal of federal restrictions on the price of gold and the resulting increase in world market price stimulated a revival of gold-mining activity in the state. Since 1981, gold production has increased by 30 percent. Exploration expenditures, however, which had increased steadily until 1981, declined by 69 percent in 1982 (Eakins and others, 1983). This rapid drop in exploration expenditures was due to a worldwide recession and high prospecting costs and also reflects decisions by major mining companies to proceed toward production of the gold deposits.

Despite cyclic and temporary changes in exploration activities, overall national priorities will make it increasingly important to find new gold reserves. It is certain that as the general demand for gold increases, the chance of finding new, profitable deposits decreases; consequently, prospecting techniques will become more sophisticated.

The electrical-resistivity method was first used by Schlumberger in 1912. Since then, it has proven to be among the most effective geophysical means for shallow subsurface investigations. Using the method, depth to bedrock, type of deposit, and other material properties can be determined. In addition, the equipment is easy to operate in the field and is easily obtainable from geophysical-equipment firms.

Before the advent of the microcomputer, complex data reduction and interpretation associated with the technique tended to impede its usefulness. This report describes a computer program that will allow quick analysis of the resistivity measurements. As with all geophysical methods, the results should be verified with well borings. As experience is gained, however, more confidence can be given to the accuracy of the results.

SCHLUMBERGER ELECTRIC RESISTIVITY

The specific electric resistivity of rock controls the quantity of current that will pass through a rock when an electric-potential difference is applied. Field measurements of the resistivity properties of subsurface formations, therefore, afford an opportunity for distinguishing one rock type from another and for determining the depth to each rock formation.

The Schlumberger configuration measures the potential gradient at the midpoint of the survey spread. It consists of four probes: two closely spaced potential-monitoring electrodes positioned midway between two widely spaced current-inducing electrodes (fig. 1). Only the current electrodes are moved during the field survey. The potential electrodes remain in their original position as long as the distance between the two inner probes is less than two-tenths the distance between the inner and outer probes. For this array, the apparent resistivity (ρ_a) can be determined by

$$\rho_a = \pi R[(4a^2 - b^2)/4b] \cong \pi Ra^2/b \qquad (1)$$

where

R = Resistance, measured as induced voltage (V) divided by current (I)
a, b = Distances (shown in fig. 1)

This measurement (ρ_a) is neither the true resistivity of a stratum nor the average resistivity of a subsurface formation but is an idealized value that can be used for survey-data interpretation.

Vertical Electrical Sounding

To apply the Schlumberger configuration to a vertical electrical sounding (VES) survey, electrode setup stations are selected along a traverse of an area for which subsurface information is desired. At a setup station, the current electrodes (fig. 1, positions A and B) are moved away from the center point of the configuration in selected increments as the depth of the survey increases. Ultimately, the current-penetration depth to strata is generally equal to the distance from a current electrode to the configuration center point (fig. 1, length a). The distance between the potential electrodes (fig. 1, length b) is usually set at 10 ft. Once the instruments obtain an adequate amount of information at one station, they are moved to the next station along the traverse, and the procedure is repeated.

The VES survey produces different apparent resistivity readings for different subsurface conditions. The method is particularly valuable when exploring for metallic deposits. VES data are usually interpretated by curve-matching techniques in which a set of theoretical curves is used in conjunction with a set produced by field measurements. The theoretical curves are computed for a particular layering structure. If a match can be obtained, the subsurface structure is assumed to be similar to the theoretical structure. Although the principle behind the method is straightforward, it requires a great deal of practice; thus, the technique can be frustrating to an inexperienced interpreter.

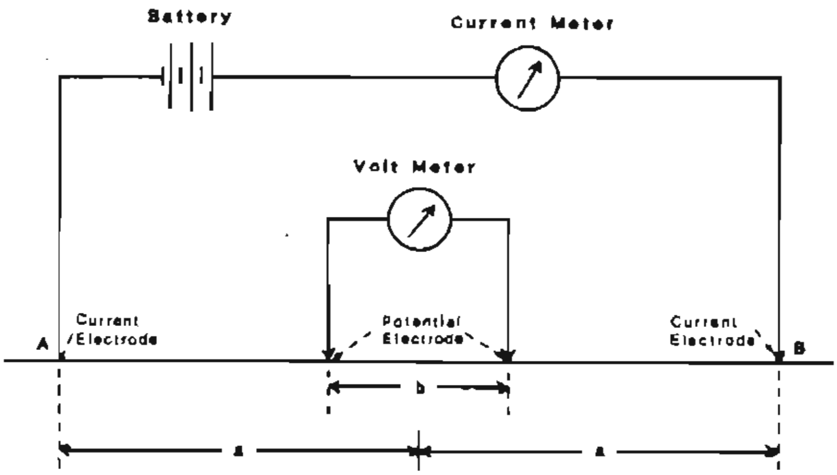


Figure 1. Arrangement of Schlumberger current (A and B) and potential electrodes.

Direct analysis of VES data gained wide acceptance with the development of the main-frame computer. A digital computer can process large numbers of VES curves in a very short period of time and fit them to theoretical layering conditions. The introduction of the microcomputer greatly enhances the merits of the direct interpretation technique.

Electric Profiling

Where large subsurface variation exists or in areas where a rapid survey is needed, electric profiling is conducted. This procedure is particularly suitable when prospecting for lode-gold bodies and when identifying dipping structures. During this type of survey, the distance between electrodes must remain constant. Because the depth of current penetration is related to the electrode spacing, the depth of investigation will be constant for all stations. From station to station, as the survey proceeds, changes in the composition or orientation of subsurface strata within the depth of interest will be reflected by changes of apparent resistivity as calculated by equation (1). The lateral variations of measurements may be constructed as a map that shows a series of equipotential lines, which indicate the distribution of potential deposits.

Regardless of the type of survey procedure, the selection of electrode spacing should be made with caution. Spacing that is too wide will make it difficult to obtain reliable field data; spacing that is too narrow will require unnecessary field time.

VES DATA ANALYSIS

Curve-matching Technique

Interpretation of VES data by a curve-matching technique is limited to a relatively simple, plane-layered geologic structure. This technique involves the following procedure:

1. Plot the observed apparent-resistivity curves on a transparent sheet.
2. Superposition the observed apparent-resistivity curves on the theoretical Schlumberger curves.
3. Shift the field curves, keeping the axes of the graphs parallel, until a good match is obtained.
4. Determine depth by interpolation.

The interpretation principle for a more complex structure is similar to the above procedures, except it is more complicated. Figure 2 (Heiland, 1946) shows an example of curve matching prepared for a two-layer structure. Resistivity of the upper layer is 1,575 ohm-ft; resistivity of the lower stratum is assumed to be infinite. From the interpretation of field data, depth to the top layer at station 13 is interpolated as 105 ft.

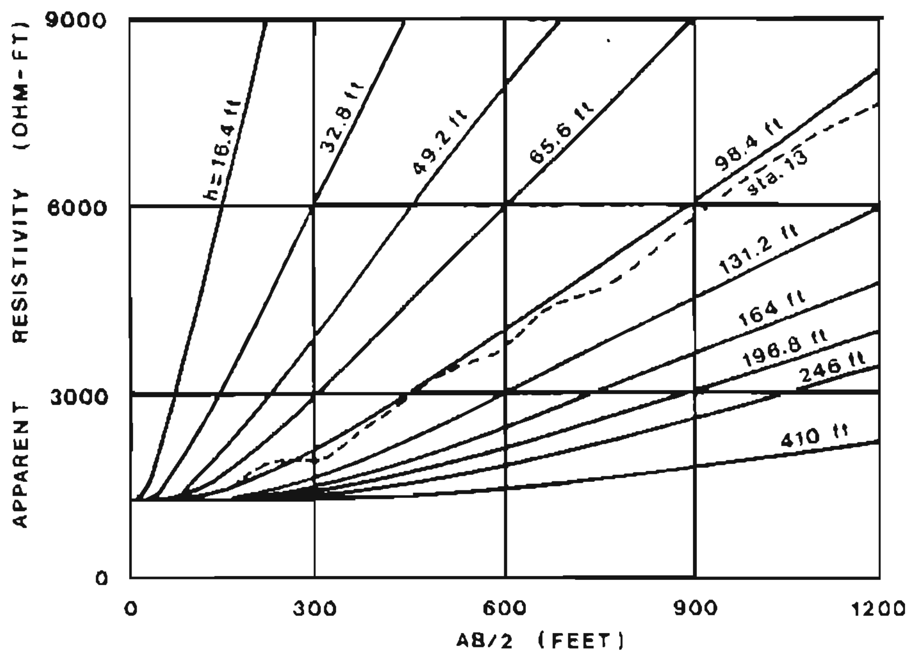


Figure 2. Schlumberger curve-matching method for a two-layer subsurface condition. Modified after Heiland (1946).

Computer-aided Technique

The mathematical determination of the layered structure from a set of field Schlumberger VES measurements can be obtained by finding the potential field induced by a point-current source. To determine the potential field, two assumptions must be made: 1) electric current flux does not exist in the resistivity layer; and 2) electric potential varies continuously across a boundary plane. The solution of the above assumptions gives the following equation:

$$V = (\rho I/2\pi)\{(1/r) + 2 \int_0^\infty K(\lambda)J_o(\lambda r)d\lambda\} \tag{2}$$

where

V = Induced voltage
ρ = Resistivity of formation
I = Current
r = Distance from any current electrode
K = Kernel function
J_o = Bessel function of zero order
λ = Integration variable

The apparent resistivity in a Schlumberger configuration can be numerically determined by the equation below:

$$\rho_a = \rho\{1 + 2r^2 \int_0^\infty K(\lambda)J_1(\lambda r)\lambda d\lambda\} \tag{3}$$

where

J_1 = Bessel function of 1st order

The computer program (app. A) described in this paper was primarily designed to provide an approximate solution for equation (3). It was originally written in FORTRAN by Zohdy (1974) for a main-frame computer and later modified in BASIC by the authors. The program consists of four subroutines, including analysis of spacing, Kernel function, curves, and least deviation of field data from the fitting curve. It calculates depth and resistivity of subsurface structures by fitting field data to a set of theoretical conditions. The computer then selects the theoretical model with the least deviation from the observed measurement as the most probable field condition. Step-by-step input statements for the VES computer program are listed in figure 3 to allow user verification of the program.

Table 1 summarizes the simulated Schlumberger sounding data based on conditions indicated in figure 3. Table 2 shows the output of the example shown in figure 3. The output includes thickness, depth, and resistivity of the subsurface structure. Deviation of the simulated data from the field-sounding curve can be estimated by the program. If the deviation is large, the calculations should be repeated until the accuracy of estimation is acceptable.

Table 1. Simulated Schlumberger VES reading[a].

Resistivity (ohm-ft)	1/2 spread (ft)
50,069.57058097547	15.5
63,226.80447371651	22.7509
73,630.83485384769	33.3937
74,566.4596171476	49.0153
60,397.39750241566	71.9446
35,396.52507961623	105.6
13,308.43469762605	155
2,744.071480481635	227.509
583.577126809063	333.937

[a]Based on conditions listed in figure 3.

* * * * * * *

RESISTIVITY---SCHLUMBERGER SOUNDING

TITLE: EXAMPLE RUN FOR SCHLUMBERGER VES SURVEY

INPUT APPARENT RESISTIVITY (Ohm-Ft), 1/2 ELECTRODE SPREAD (Ft):

INPUT -1 FOR RESISTIVITY AND SPREAD WHEN COMPLETE

RESISTIVITY	1/2 SPREAD
37000	15.5
70000	31
90000	46
74000	61.5
64000	76
40000	92
26000	107
18000	122
11000	137
8800	152
6500	167
5000	182
3800	197
2950	212
2150	227
1980	242
1700	257
1680	272
1700	287
1600	302
-1	-1

ITERATION AGAIN (Y/N)? Y

INTERATION UPWARD OR DOWNWARD (UP/DOWN)? UP

INCREMENTS FOR DEPTH AND RESISTIVITY: .05 .05

TOTAL NUMBER OF SUBSURFACE LAYERS: 3

DEPTH (Ft) AND RESISTIVITY (Ohm-Ft):

DEPTH	RESISTIVITY
5.4	22655
21.6	226550
999999	566

* * * * * * *

Figure 3. Example illustrating input procedure for VES computer program.

Table 2. Subsurface layering configuration[a] estimated by the VES computer program.

Thickness (ft)	Depth (ft)	Resistivity (ohm-ft)
5.4	5.4	22,655
16.2	21.6	226,550
999,977	999,999	566

[a]Based on conditions listed in figure 3.

FIELD EXAMPLE

To evaluate the accuracy of the computer processing, VES surveys were conducted at two sites (fig. 4) at the U.S. Army Cold Regions Research and Engineering Laboratory (CRREL) permafrost tunnel. The sounding technique had a maximum spread of 600 ft. Potential electrodes were spaced at 10 ft, and current electrodes were moved outward in increments of 15 ft. Three control points were measured at site I and four at site II. Plots of the theoretical and field VES curves for the survey sites are shown in figures 5 and 6.

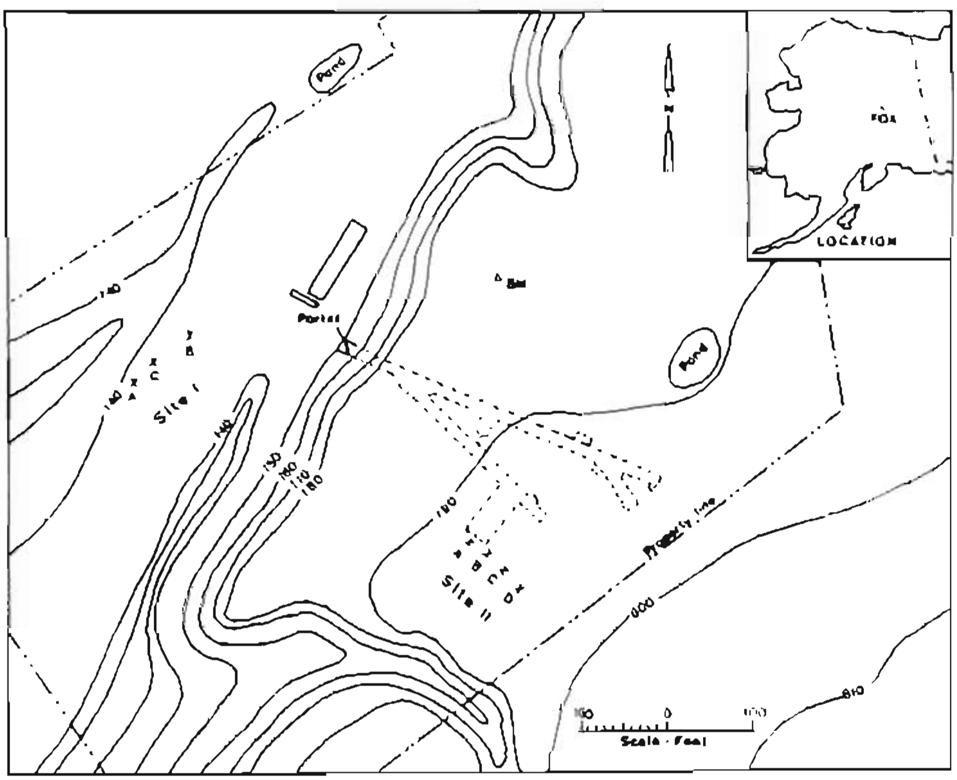


Figure 4. Map showing VES survey sites at CRREL permafrost tunnel, Fox, Alaska.

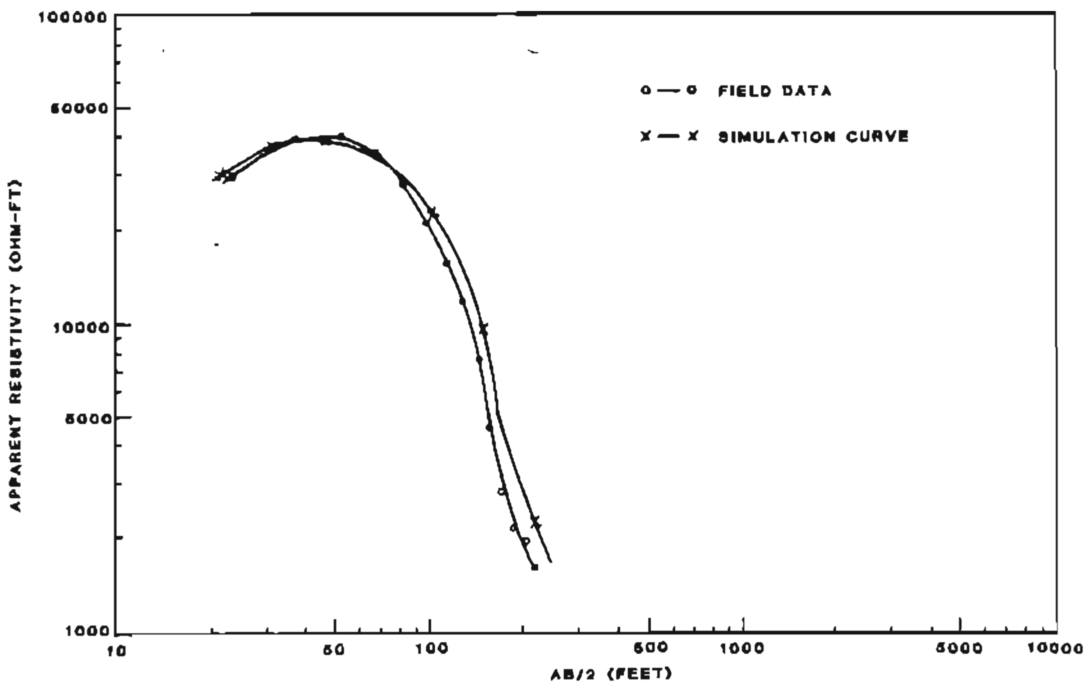


Figure 5. Diagram showing theoretical and field VES curves for station B, site I. AB/2 = One-half current-electrode spacing.

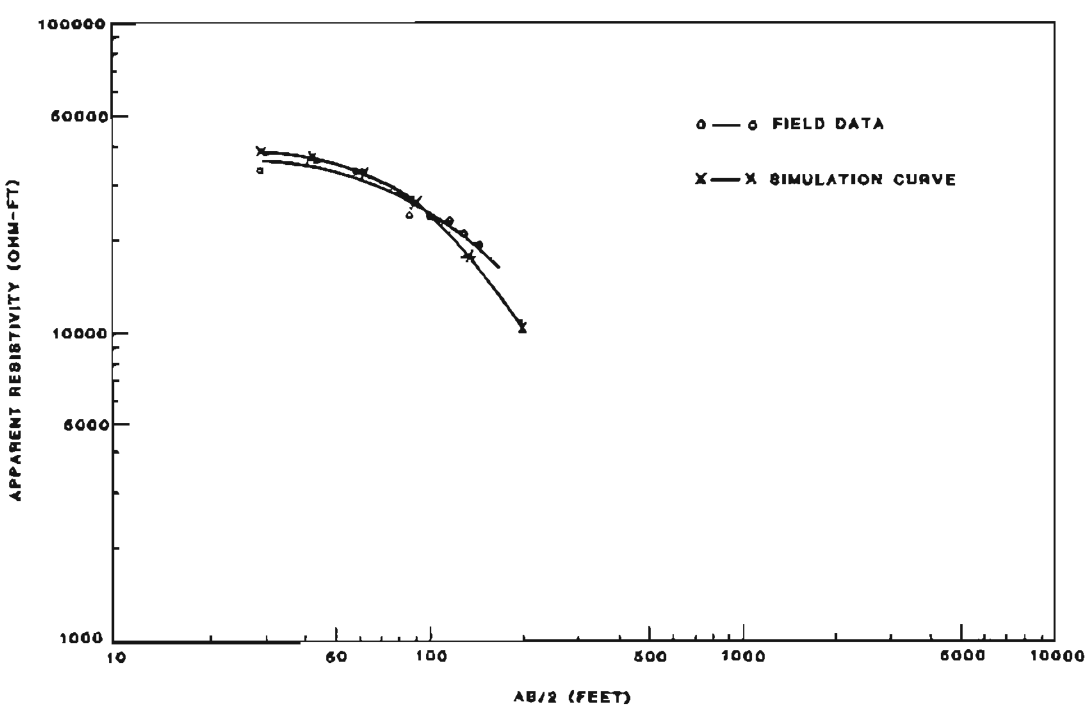


Figure 6. Diagram showing theoretical and field VES curves for for station A, site II. AB/2 = One-half current-electrode spacing.

Three distinct lithologic layers---frozen silt, gold-bearing gravel, and schist bedrock---are known to underlie the area of the permafrost tunnel. Results of the computer simulation (table 3) indicate that three subsurface layers exist at site I. The near-surface layer has an electrical resistivity of 5,000 to 23,000 ohm-ft and is about 2.5- to 5.5-ft thick. This relatively low resistivity may have been caused by ice melting during the survey. The layer underlying the low resistivity zone has a rather high resistivity of 150,000 to 225,000 ohm-ft, probably due to low water content. Thickness of the second layer is about 9 to 16 ft. The deepest layer has a very high conductivity that could be caused by weathering of bedrock to clay. Thickness of the third layer could not be determined because of the limited spread of the survey.

Table 3. Results of electric-resistivity surveys at at site I, CRREL permafrost tunnel.

Thickness (ft)	Depth (ft)	Resistivity (ohm-ft)
Station A		
5.4	5.4	22,655
16.2	21.6	226,550
- -	- -	556
Station B		
5.6	5.6	10,439
13.4	19.0	153,519
- -	- -	110
Station C		
2.4	2.4	5,000
8.8	11.2	100

- - Unable to determine because of limited spread of survey.

Table 4 shows the results of the computer interpretation of field measurements taken at site II. Because of the topography at the site, it was not possible to set up the necessary stations to identify the gravel-bedrock contact. Survey results, however, indicate that there are two subsurface layers. The upper layer has a resistivity that ranges from 33,000 to about 39,000 ohm-ft. Thickness of the near-surface layer ranges from 50 to 60.5 ft. The underlying layer has a relatively low resistivity of 5,000 to 9,000 ohm-ft.

DISCUSSION AND CONCLUSIONS

Observations from field surveys show that the electric resistivity method can quickly detect subsurface strata with certain accuracy, although ground conditions, such as disturbance from permafrost, can introduce deviation from the true subsurface configuration. Computer applications are

Table 4. Results of electric-resistivity surveys at site II, CRREL permafrost tunnel.

Thickness (ft)	Depth (ft)	Resistivity (ohm-ft)
Station A		
60.5	60.5	38,885
- -	- -	5,000
Station B		
50.0	50.0	35,443
- -	- -	5,933
Station C		
55.0	55.0	33,750
- -	- -	9,000
Station D		
54.0	54.0	33,000
- -	- -	8,800

- -Unable to determine because of topography at survey site.

convenient for processing large amounts of data in a very short period of time. Information provided by the computer, however, should be used with caution. For example, the computer solution in this study indicated that the depth to bedrock at site I, U.S. Army CRREL permafrost tunnel, was located 11 to 22 ft below the ground surface. This measurement is less than the actual 25 ft to bedrock. In addition, the simulated VES curves showed the silt-gravel boundary at site II at a shallower depth (50 to 60 ft) than the actual depth (60 ft) of the boundary. (See figure 7).

When interpreting field data, the computer simulates a vertical sounding curve based on a theoretical subsurface configuration. Three parameters---number of layers, depth, and resistivity---must be defined by the investigator before the computation begins. For each solution attempt, the combination of the three parameters that has the least deviation from the field data will be selected by the computer for the model calculation. This best-fitting model, however, may not truly represent the field-layering arrangement. By changing the theoretical subsurface configuration, such as subdividing one layer into several sublayers, the best-fitting model will be changed accordingly. Therefore, computer interpretation of the vertical resistivity sounding data should be made in conjunction with known information.

ACKNOWLEDGMENTS

We wish to express our gratitude to the U.S. Bureau of Mines for financial support. We also greatly appreciate the assistance from Ms. Lucy Trant.

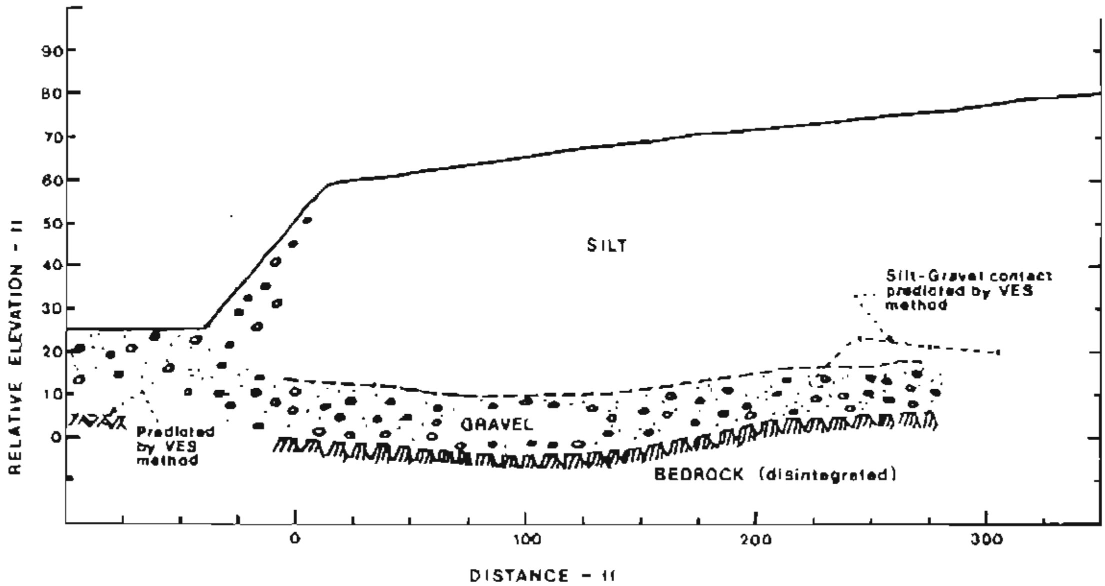


Figure 7. General geologic cross section of study area at CRREL permafrost tunnel, Fox, Alaska.

REFERENCES

Eakins, G.R., Bundtzen, T.K., Robinson, M.S., Clough, J.G., Green, C.B., Clautice, K.H., and Albanese, M.A., 1983, Alaska's mineral industry - 1982: Alaska Division of Geological and Geophysical Surveys Special Report 31, 63 p.

Heiland, C.A., 1946, Geophysical exploration: New York, Prentice-Hall, 1,013 p.

Zohdy, A.R., 1974, A computer program for the calculation of Schlumberger sounding surves by convolution: National Technical Information Services (NTIS) Publication 23.2056, 11 p.

APPENDIX A

Electric Resistivity Program for the Schlumberger Sounding Survey

```
1000 REM *************************************************************
1010 REM ***                                                      ***
1020 REM *** ELECTRIC RESISTIVITY PROGRAM FOR THE SCHLUMBERGER ***
1030 REM *** SOUNDING SURVEY.  PROGRAMMED BY SCOTT L. HUANG     ***
1040 REM *** 5/20/84.  PORTION OF THE PROGRAM IS MODIFIED FROM ***
1050 REM *** THE VES PROGRAM (ZOHDY, 1974).                     ***
1060 REM ***                                                      ***
1070 REM *** THE RESISTIVITY PROGRAM CALCULATES THE DEPTH AND   ***
1080 REM *** RESISTIVITY OF SUBSURFACE FORMATION BY ASSUMING    ***
1090 REM *** A THEORETICAL CONDITION. THE PROGRAM WILL AUTOMA- ***
1100 REM *** TICALLY ESTIMATE A FIELD APPARENT-RESISTIVITY      ***
1110 REM *** CURVE, WHICH HAS THE LEAST DEVIATION FROM THE      ***
1120 REM *** ACTURAL FIELD SURVEY CURVE.  TEN ITERATIONS WILL   ***
1130 REM *** BE CONDUCTED FOR EACH COMPUTER RUN.                ***
1140 REM ***                                                      ***
1150 REM *************************************************************
1160 PRINT TAB(15);"*********************************************"
1170 PRINT TAB(15);"*                                           *"
1180 PRINT TAB(15);"*   RESISTIVITY -- SCHLUMBERGER SOUNDING    *"
1190 PRINT TAB(15);"*                                           *"
1200 PRINT TAB(15);"*********************************************"
1210 LPRINT TAB(15);"*********************************************"
1220 LPRINT TAB(15);"*                                           *"
1230 LPRINT TAB(15);"*   RESISTIVITY -- SCHLUMBERGER SOUNDING    *"
1240 LPRINT TAB(15);"*                                           *"
1250 LPRINT TAB(15);"*********************************************"
1260 PRINT:PRINT:LPRINT:LPRINT
1270 DIM RESIST(50),THICK(50),VV#(50),XAMDA#(50),V(50),VES1#(50),VES2#(50)
1280 DIM DEPTH(50),VES#(50),RADIUS(50),XK#(50),DF(50),RHO(50)
1290 DIM AMDA(50),VV1#(50),VV2#(50),XVES1#(50),XVES2#(50)
1300 DIM SPAC(50),RSQ#(10),DEPTH1(50),RHO1(50),APRH#(50)
1310 PRINT "TITLE FOR THE SURVEY: ";:INPUT TITLE$
1320 LPRINT "TITLE -- ";TITLE$
1330 J=1
1340 PRINT "INPUT APPARENT RESISTIVITY (Ohm-Ft). 1/2 ELECTRODE SPREAD (Ft):"
1350 LPRINT "INPUT APPARENT RESISTIVITY (Ohm-Ft). 1/2 ELECTRODE SPREAD (Ft):"
1360 PRINT "INPUT -1 FOR RESISTIVITY AND SPREAD WHEN COMPLETE"
1370 LPRINT "INPUT -1 FOR RESISTIVITY AND SPREAD WHEN COMPLETE"
1380 PRINT:LPRINT
1390 LPRINT TAB(10);"RESISTIVITY";TAB(39);"1/2 SPREAD"
1400 PRINT:LPRINT
1410 INPUT APRH#(J),SPAC(J)
1420 IF SPAC(J)=-1 THEN 1450
1430 LPRINT TAB(13);APRH#(J);TAB(43);SPAC(J)
1440 J=J+1: GOTO 1410
1450 NOO=J-1: RADMIN=999999!: RADMAX=0
1460 FOR JJ=1 TO NOO
1470 IF RADMIN >= SPAC(JJ) THEN RADMIN=SPAC(JJ)
1480 IF RADMAX <= SPAC(JJ) THEN RADMAX=SPAC(JJ)
1490 NEXT JJ
1500 PRINT:LPRINT:PRINT:LPRINT
1510 PRINT "ITERATION  (Y/N) ";:INPUT TR$:PRINT
1520 LPRINT "ITERATION AGAIN (Y/N)? ";TR$:LPRINT
1530 IF TR$="N" THEN 2230
1540 FINAL=0
1550 PRINT "ITERATION UPWARD OR DOWNWARD (UP/DOWN) ";:INPUT ITE$
1560 LPRINT "ITERATION UPWARD OR DOWNWARD (UP/DOWN)? ";ITE$
1570 IF ITE$="UP" THEN UD=1
```

```
1580 IF ITE$="DOWN" THEN UD=-1
1590 PRINT:LPRINT
1600 PRINT "INCREMENTS FOR DEPTH AND RESISTIVITY: ";:INPUT INCD,INCR
1610 LPRINT "INCREMENTS FOR DEPTH AND RESISTIVITY: ";INCD,INCR
1620 LPRINT:PRINT
1630 PRINT "TOTAL NUMBER OF SUBSURFACE LAYERS: ";: INPUT LAYERS
1640 LPRINT "TOTAL NUMBER OF SUBSURFACE LAYERS: ";LAYERS
1650 PRINT:LPRINT
1660 PRINT "DEPTH (Ft) AND RESISTIVITY (Ohm-Ft): "
1670 LPRINT "DEPTH (Ft) AND RESISTIVITY (Ohm-Ft): "
1680 FOR I=1 TO LAYERS
1690 INPUT DEPTH1(I),RHO1(I)
1700 PRINT TAB(1);DEPTH1(I);TAB(20);RHO1(I)
1710 LPRINT TAB(1);DEPTH1(I);TAB(20);RHO1(I)
1720 NEXT I
1730 FOR LS=1 TO 10
1740 RSQ#(LS)=0
1750 FOR I=1 TO LAYERS
1760 DEPTH(I)=DEPTH1(I)*(1+INCD*(LS-1)*UD)
1770 RHO(I)=RHO1(I)*(1+INCR*(LS-1)*UD)
1780 NEXT I
1790 GOSUB 2240: GOSUB 2560: GOSUB 2830
1800 IF FINAL=0 THEN GOSUB 3580 ELSE 1920
1810 NEXT LS
1820 MIN#=999999!
1830 FOR LS=1 TO 10
1840 IF RSQ#(LS) <= MIN# THEN 1850 ELSE 1860
1850 MIN#=RSQ#(LS): FINAL=LS
1860 NEXT LS
1870 FOR I=1 TO LAYERS
1880 DEPTH(I)=DEPTH1(I)*(1+INCD*(FINAL-1)*UD)
1890 RHO(I)=RHO1(I)*(1+INCR*(FINAL-1)*UD)
1900 NEXT I
1910 GOTO 1790
1920 PRINT:PRINT:PRINT:LPRINT:LPRINT:LPRINT
1930 PRINT "*** SIMULATED SURFACE LAYERING CONFIGURATION ***"
1940 LPRINT "*** SIMULATED SURFACE LAYERING CONFIGURATION ***"
1950 PRINT:PRINT:PRINT:LPRINT:LPRINT:LPRINT
1960 PRINT TAB(10);"THICKNESS";TAB(26);"DEPTH";TAB(38);"RESISTIVITY"
1970 LPRINT TAB(10);"THICKNESS";TAB(26);"DEPTH";TAB(38);"RESISTIVITY"
1980 PRINT TAB(12);"(Ft)";TAB(27);"(Ft)";TAB(39);"(Ohm-Ft)":PRINT
1990 LPRINT TAB(12);"(Ft)";TAB(27);"(Ft)";TAB(39);"(Ohm-Ft)":LPRINT
2000 FOR I=1 TO LAYERS
2010 PRINT TAB(10);THICK(I);TAB(25);DP(I);TAB(40);RHO(I)
2020 LPRINT TAB(10);THICK(I);TAB(25);DP(I);TAB(40);RHO(I)
2030 NEXT I
2040 PRINT:PRINT:PRINT:LPRINT:LPRINT:LPRINT
2050 PRINT "*** SIMULATED SCHLUMBERGER VES DATA ***":PRINT:PRINT:PRINT
2060 LPRINT "*** SIMULATED SCHLUMBERGER VES DATA ***":LPRINT:LPRINT:LPRINT
2070 PRINT TAB(10);"1/2 SPREAD";TAB(39);"RESISTIVITY"
2080 LPRINT TAB(10);"1/2 SPREAD";TAB(39);"RESISTIVITY"
2090 PRINT TAB(13);"(Ft)";TAB(40);"(Ohm-Ft)":PRINT
2100 LPRINT TAB(13);"(Ft)";TAB(40);"(Ohm-Ft)":LPRINT
2110 FOR L=1 TO NRADII
2120 PRINT TAB(11);RADIUS(L);TAB(35);VES#(L)
2130 LPRINT TAB(11);RADIUS(L);TAB(35);VES#(L)
2140 NEXT L
2150 PRINT:PRINT:PRINT:LPRINT:LPRINT:LPRINT
2160 PRINT "*** STATISTICAL INFORMATION ***"
```

```
2170 LPRINT "*** STATISTICAL INFORMATION ***"
2180 PRINT:PRINT:PRINT "LEAST SQUARE ROOT OF DEVIATION: ";RSQ#(FINAL)
2190 PRINT "AT ";FINAL;" ITERATION"
2200 LPRINT:LPRINT:LPRINT "LEAST SQUARE ROOT OF DEVIATION: ";RSQ#(FINAL)
2210 LPRINT "AT ";FINAL;" ITERATION"
2220 GOTO 1510
2230 END
2240 REM **************************
2250 REM *** SPACING SUBROUTINE ***
2260 REM **************************
2270 THICK(1)=DEPTH(1)
2280 FOR I=2 TO LAYERS
2290 THICK(I)=DEPTH(I)-DEPTH(I-1)
2300 NEXT I
2310 XRATIO#=EXP(LOG(10)/6)
2320 RADIUS(1)=RADMIN
2330 I=2
2340 RADIUS(I)=XRATIO#*RADIUS(I-1)
2350 IF RADIUS(I) >= RADMAX THEN 2380
2360 I=I+1
2370 GOTO 2340
2380 NRADII=I
2390 RADMAX=RADIUS(I)
2400 XMIN=(RADMIN/XRATIO#^10)*1.05
2410 XMAX=(RADMAX*XRATIO#^6)*1.05
2420 XAMDA#(1)=XMIN
2430 I=2
2440 XAMDA#(I)=XRATIO#*XAMDA#(I-1)
2450 IF XMAX-XAMDA#(I) <= 0 THEN 2480 ELSE 2460
2460 I=I+1
2470 GOTO 2440
2480 NRAD=I
2490 NN=LAYERS
2500 SUNN=0
2510 FOR I=1 TO NN
2520 SUNN=SUNN+THICK(I)
2530 DP(I)=SUNN
2540 NEXT I
2550 RETURN
2560 REM *************************
2570 REM *** KERNEL SUBROUTINE ***
2580 REM *************************
2590 FOR J=1 TO NRAD
2600 V(J)=1
2610 K=LAYERS-1
2620 KK=LAYERS+1
2630 AMDA(J)=-2/XAMDA#(J)
2640 FOR I=1 TO K
2650 M=KK-I
2660 L=LAYERS-I
2670 P=RHO(M)*V(J)
2680 AK=(RHO(L)-P)/(RHO(L)+P)
2690 PROD=AMDA(J)*THICK(L)
2700 IF PROD < -50 THEN PROD=-50
2710 Q=AK*EXP(PROD)
2720 ABQ=ABS(Q)
2730 IF ABQ <= 0 THEN 2740 ELSE 2760
2740 V(J)=1
2750 GOTO 2770
```

```
2760 V(J)=(1-Q)/(1+Q)
2770 NEXT I
2780 NEXT J
2790 FOR J=1 TO NRAD
2800 VV#(J)=V(J)*RHO(1)
2810 NEXT J
2820 RETURN
2830 REM ************************
2840 REM *** CONVES SUBROUTINE ***
2850 REM ************************
2860 XK#(1)=.0148
2870 XK#(2)=-.0814
2880 XK#(3)=.4018
2890 XK#(4)=-1.5716
2900 XK#(5)=1.972
2910 XK#(6)=.1854
2920 XK#(7)=.1064
2930 XK#(8)=-.0499
2940 XK#(9)=.0225
2950 FOR I=1 TO 50
2960 VV1#(I)=0'
2970 VV2#(I)=0'
2980 NEXT I
2990 NODD=0
3000 NEVEN=0
3010 IF NRAD/2 = INT(NRAD/2) THEN 3050
3020 NODD=NRAD
3030 NN=(NODD+1)/2
3040 GOTO 3080
3050 NEVEN=NRAD
3060 NN=NEVEN/2
3070 GOTO 3160
3080 FOR J=1 TO NN
3090 VV1#(J)=VV#(2*J-1)
3100 NEXT J
3110 MM=NN-1
3120 FOR J=1 TO MM
3130 VV2#(J)=VV#(2*J)
3140 NEXT J
3150 GOTO 3200
3160 FOR J=1 TO NN
3170 VV1#(J)=VV#(2*J-1)
3180 VV2#(J)=VV#(2*J)
3190 NEXT J
3200 REM
3210 M=0
3220 L=1
3230 LL=9
3240 FOR J=L TO LL
3250 REM
3260 XVES1#(J)=VV1#(J)*XK#(J-M)
3270 XVES2#(J)=VV2#(J)*XK#(J-M)
3280 NEXT J
3290 REM
3300 SMVES1#=0
3310 SMVES2#=0
3320 FOR J=L TO LL
3330 SMVES1#=XVES1#(J)+SMVES1#
3340 SMVES2#=XVES2#(J)+SMVES2#
```

```
3350 NEXT J
3360 VES1#(L)=SMVES1#
3370 VES2#(L)=SMVES2#
3380 L=L+1
3390 LL=LL+1
3400 M=M+1
3410 IF LL > NRAD THEN 3430
3420 GOTO 3240
3430 IF NRAD=NODD THEN 3450
3440 GOTO 3520
3450 FOR J=1 TO NN
3460 VES#(2*J-1)=VES1#(J)
3470 NEXT J
3480 FOR J=1 TO MM
3490 VES#(2*J)=VES2#(J)
3500 NEXT J
3510 GOTO 3560
3520 FOR J=1 TO NN
3530 VES#(2*J-1)=VES1#(J)
3540 VES#(2*J)=VES2#(J)
3550 NEXT J
3560 REM
3570 RETURN
3580 REM ***
3590 REM *** LEAST SQUARE SUBROUTINE ***
3600 REM ***
3610 K=0
3620 FOR J=1 TO NOD
3630 FOR I=1 TO NRADII
3640 IF (SPAC(J)>=RADIUS(I))AND(SPAC(J)<=RADIUS(I+1)) THEN 3650 ELSE 3700
3650 XX#=VES#(I+1)-VES#(I)
3660 INTR#=VES#(I)+(SPAC(J)-RADIUS(I))/(RADIUS(I+1)-RADIUS(I))*XX#
3670 RSQ#(LS)=RSQ#(LS)+(INTR#-APRH#(J))^2
3680 K=K+1
3690 GOTO 3710
3700 NEXT I
3710 NEXT J
3720 RSQ#(LS)=RSQ#(LS)/K
3730 RSQ#(LS)=SQR(RSQ#(LS))
3740 RETURN
```

LONSDALEITE: GEOLOGIC ASPECTS AND PERSPECTIVES

By
E.I. Erlich, Consulting Geologist
595 South Forest, Apartment 311
Denver, Colorado 80222
and
Gennady Slonimsky, Consulting Geologist
595 South Forest, Apartment 311
Denver, Colorado 80222

LONSDALEITE - MINERALOGICAL AND EXPERIMENTAL DATA

A new mineral form of carbon, lonsdaleite, was first described by Bundy and Kasper in 1967. They had synthesized it at a temperature greater than 1,000°C and under static pressure that had exceeded 130 kbar. Another laboratory obtained the same transformation by methods of intense shock compression and strong thermal quenching[1]. Almost simultaneously, the mineral was discovered in meteorites from Canyon Diablo (Frondel and Marvin, 1967) and Goalpara (Hanneman and others, 1967).

The name, lonsdaleite, was chosen to honor the distinguished British crystallographer Professor Kathleen Lonsdale. The name was approved by the Commission of New Minerals and officially published in 1967 in 'The American Mineralogist' (Fleischer, 1967). When examined under an electron microprobe, the mineral was found to consist of carbon only. It belongs to the hexagonal class 6/m2/m2/m and the space group P6 3/mmc, Z:4. Its crystal structure is characterized by unit cell dimensions of 2.52 and 4.12 Å. In meteorites, the mineral was found as cubes and cubooctahedrons to 0.7 mm. The theoretical density of lonsdaleite is the same as cubic diamond, 3.51 g/cm^3. In fact, evidence indicates that the mineral is probably formed by the transformation of diamond to a hexagonal form. Lonsdaleite is pale brown yellow under the microscope and faintly birefringent with $\underset{\sim}{n}$ slightly higher than 2.404 (Roberts and others, 1975). Its strongest diffraction lines are 2.19 (100) and 1.26 (75).

Lonsdaleite was next discovered by Vdovykin (1970) in the North Haig and Dingo Pup meteorites of Western Australia. Because the first identifications of natural lonsdaleite were in meteorites, genesis of the mineral was commonly thought to be connected with shock metamorphism. It is interesting to note that it was Kathleen Lonsdale, whose name had been given to the mineral, who indicated the possibility of a nonshock-related origin for the mineral (Lonsdale, 1971). She pointed out that lonsdaleite might be generated under conditions of static pressure.

Two years later, there were reports of lonsdaleite from possible nonmeteoritic sources. Sokhor and others (1973) described schistose

[1]Patented by E.I. DuPont de Nemours and Company, 1965 (Netherland Patent 6506395).

diamonds in titanium placers of the Ukrainian Shield. The grains were flat and irregular and ranged from 0.05 to 0.3 mm. Birefrigence averaged 0.0035 and, in some specimens, reached 0.01. The authors indicated a high degree of deformation, which in their opinion suggested shock metamorphism. But the association of lonsdaleite with a generally constant mineral assemblage strongly contradicts meteoritic genesis of the mineral.

Four years later, another paper appeared that described lonsdaleite of a nonmeteoritic origin. The mineral was discovered in eclogites near Shubino Village, southern Urals, and Sal'naya Tundra, Kola Peninsula (Golovanya and others, 1978). Both occurrences were closely associated with the minerals moissanaite and graphite.

Extremely interesting are the latest findings of lonsdaleite in diamond placers in Yakutiya (Kaminsky and others, 1985). These discoveries indicate that lonsdaleite can occur as a result of phase transformations in normal diamonds. This possibility was confirmed experimentally (Sozin and others, 1983) in the process of dynamic synthesis of diamonds. The most interesting data on the subject, however, have been obtained during detailed geochemical studies of diamond crystals. These investigations indicate that parts of natural diamonds contain inclusions of lonsdaleite enriched by carbonate ions and hydrogen. The parts of the diamond crystal without such inclusions contain nitrate ions (Miliuvina, 1977). This leads to the suggestion that varieties of natural diamonds described as carbonado usually contain considerable amounts of the lonsdaleite phase. Thus, it is possible to conclude that the presence of lonsdaleite does not necessarily indicate meteoric impact.

GEOLOGY OF THE GREAT LONSDALEITE DEPOSIT IN THE POPIGAY DEPRESSION, SIBERIAN PLATFORM

All of the foregoing studies were of academic interest only until the beginning of the 1970s when the great lonsdaleite deposit was found in the Popigay depression, located on the northernmost apex of the Anabar Shield (fig. 1). The depression, about 100-km diam, cuts rocks of different ages, from Proterozoic and Cambrian dolomites, limestones, and sandstones to unconsolidated Permian rocks and Lower Triassic dolerites and trachydolerites. It is filled with massive and brecciated volcanogenic rocks that are nearly dacitic in composition[2]. Throughout the complex, there is widespread evidence for remelting of the host rocks, including the presence of allogenic breccia (a complex of host-rock fragments saturated by volcanogenic cement).

Because different theories---volcano-tectonic vs. meteoritic impact---have been proposed for the origin of the Popigay structure, it is not surprising that different nomenclature is used for rocks in the region. Below is a short review of the main types of rock; all details can be found in the most recent work of Masaytis and others (1976).

---

[2] SiO_2 content is about 63 percent with some abnormal alkalies, particularly 1.73 to 1.98 percent Na_2O and 2.54 to 2.72 percent K_2O. CaO content is about 3.17 to 3.68 percent and is almost equal to MgO content (Masaytis and others, 1976).

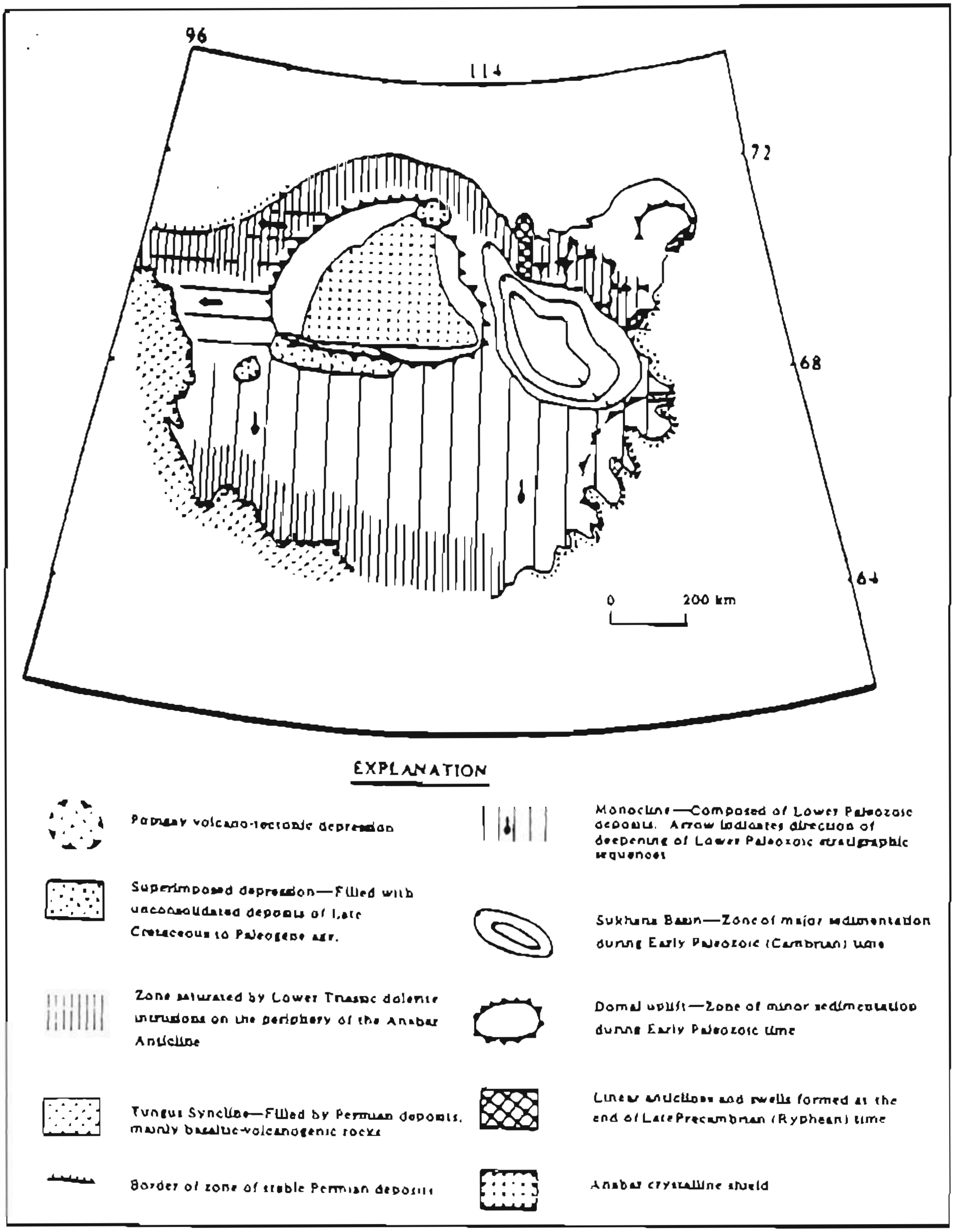


Figure 1. Popigay volcano-tectonic depression, northeastern Siberian platform. (See 'Diamonds in the Ice,' p. 14, for figure location.)

The central part of the Popigay depression is filled with andesitic to dacitic tuff breccia (suevites), which extends beyond the limits of the inner crater in long tongues. The rocks are characterized by different degrees of welding and corresponding variations in texture: in some cases, the rocks look like typical ignimbrites with glassy flamme. On the basis of fragment composition, the tuffs can be divided into vitric, lithic, and crystal types.

On the margins of the welded tuff breccia are unwelded rocks that form independent fields of volcanic sands (cuptoclastites). The sands also form separate lenses or layers within the tuff breccia and, in several cases, are transitional with the welded rock. They consist primarily (about 80 percent) of unsorted, unstratified fragments of quartz and feldspar that are < 1- to 2-mm diam. Among the other minerals present are pyroxene, hornblende, monazite, zircon, apatite, and tourmaline; lithic fragments include silicic volcanic rock, gneiss, carbonate rock, and siltstone.

Glassy aphanitic andesites and dacites are found in eluvium in watersheds of the Popigay structure and probably reflect previously existing cover rock. Masaytis and others (1976) gave these rocks a new name, tagamites or massive impactites, and considered them to be a pseudomagmatic formation. Occasionally these rocks are observed as dikes, tens of centimeters to several meters thick, that cut the tuff breccia. Some of the rocks have porosities that range from 15 to 20 percent. The pores are ellipsodial, reaching lengths of 5 to 8 mm. From 3 to 5 percent of the rock's volume is composed of large fragments of host rock that consist primarily of gneiss and quartzite.

Masaytis and others (1976) described two other types of deposits in the Popigay depression. The first is an allogenic breccia that consists of fragments and blocks from a meter to several tens of meters diam. These fragments are composed of different types of host rock, ranging from Archean basement rock to Cretaceous sedimentary-volcanogenic cover. The fragments are cemented by volcanic sands or coptoclastites, which contain smaller fragments of the host rock. The second deposit is an autogenic breccia that was developed in the crushed basement of the structure. It was formed as Archean rocks underwent different degrees of remelting and shock metamorphism. This breccia is found in the area of the resurgent dome in the central part of the depression and in the northwest corner of the Popigay structure. Describing these two localities, Masaytis emphasized unusual amounts of remelted rock and shock metamorphism.

Zonal structure in the depression is shown in figure 2. According to Polyakov and Trukhalev (1975) and later Masaytis and others (1976), the structure consists of two parts. The central part of the depression, about 72-km diam, is characterized by arcuate zones of uplifted basement rock, composed predominantly of crystalline rock breccia. The breccia has been cemented by the glassy products of remelted host rock (Masaytis' tagamites). A weak positive magnetic anomaly has been measured in this inner zone over a background of a practically neutral magnetic field, which is typical of the outer zone.

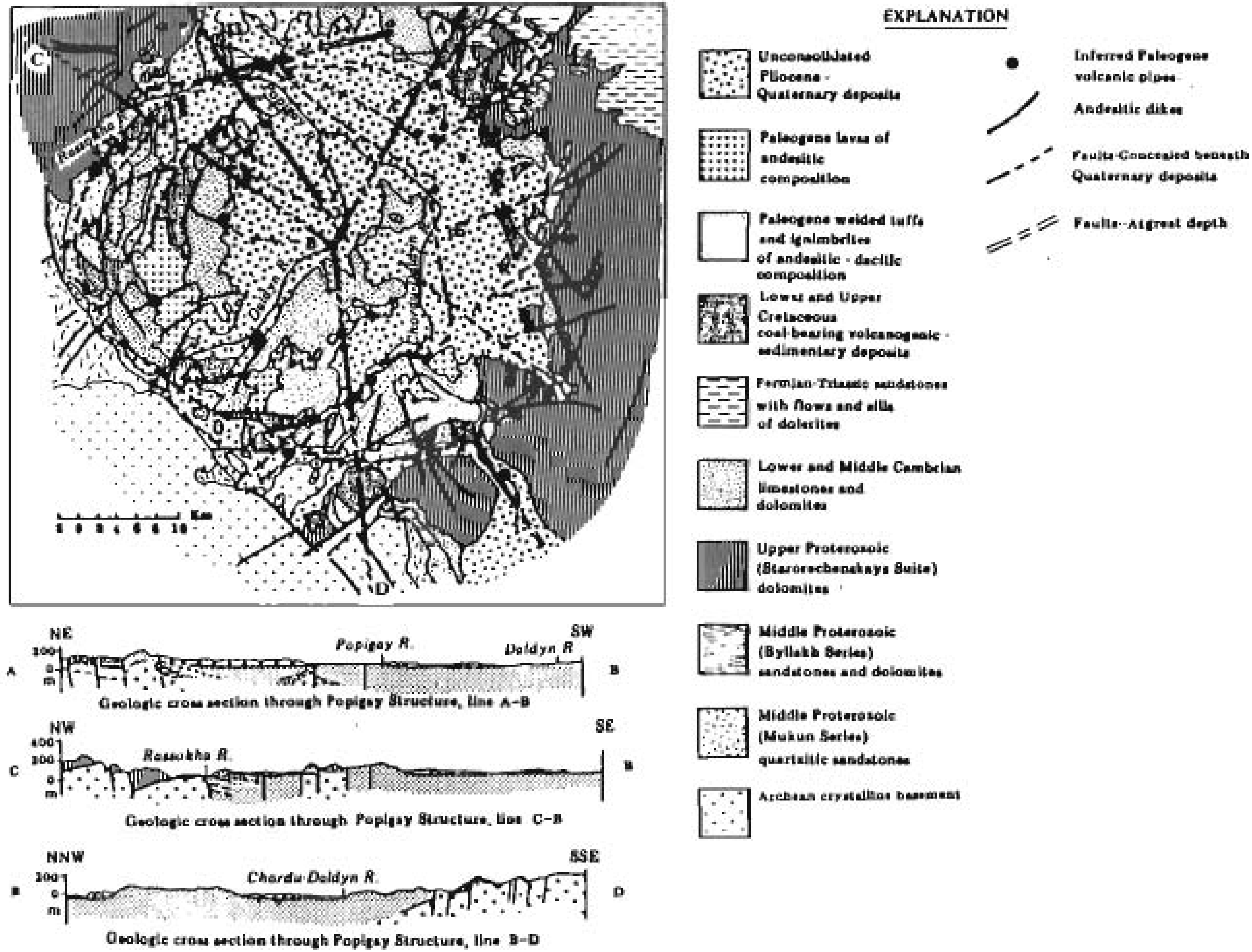


Figure 2. Geologic map and cross sections of the Popigay volcano-tectonic depression. (See figure 1 for map location.) After Polyakov and Trukhalev (1975).

The outer zone has been divided into two subzones. The inner subzone is formed by deformed cap rock overlain by tongues of tuffogenic rock; bodies of tagamites are also encountered. Within the subzone, klippen are extensively developed in the tuffogenic rocks. Gouging, wrapping, and crushing structures, up to 5-km wide, have been noted. The outer subzone is formed by cap rock with multiple overthrusts and upthrusted overthrusts, whose amplitude and plane slopes decrease in proportion to the distance from the center of the depression. Fragments of Cretaceous deposits and tuffogenic rock dip in arc-shaped depressions to 7.5-km wide.

The age of the volcanic fill varies according to different authors, but they agree that it was deposited during Cenozoic time and that part of the sequence accumulated from Neogene through Quaternary time (Zhabin, 1969).

METEORITIC IMPACT THEORY

The Popigay structure had always been considered a normal volcano-tectonic depression (Polyakov and Trukhalev, 1975) until a prominent petrologist from the National Geological Institute of Leningrad, Victor Masaytis, summarized several unusual features of the structure. He stated, in particular, that volcanic manifestations of so young an age were not typical for the northern part of the Siberian Platform. He also noted that the great diameter of the structure was unusual. Masaytis emphasized the abundance of breccia and the unusual quantity of host-rock melting. He described shatter cones and diapletic glass, characteristic of blast structures, and emphasized the presence of thrusts and upthrusted overthrusts (not the normal faults characteristic of volcano-tectonic structures) along the boundaries of the depression.

The presence of all these unusual features suggested to Masaytis that the Popigay structure is not a normal volcano-tectonic depression but rather an astrobleme; that is, the result of a meteoritic impact (fig. 3). To support his hypothesis, Masaytis obtained samples of high pressure minerals from the structure. During his investigation, he found many small (< 1 mm) grains of lonsdaleite in the breccia-textured rock. Pursuing the mineral data, he gave all of the rocks names in accordance with existing nomenclature for rocks of meteoritic craters: suevites and allogenic breccia for different types of pyroclastic rock, and the new local name, tagamites, for massive-textured rocks that occasionally form cement in brecciated host rock. Details of the study are presented in a book by Masaytis and others (1976).

Here it must be mentioned that the main purpose for geologic exploration in this part of Siberia was for diamonds. The discovery of the lonsdaleite 'diamonds,' a mineral three times harder than normal diamond, created a real sensation. Immediately, the National Geological Institute in Leningrad created a special division, with V.L. Masaytis as its chief, to investigate the Popigay structure. Khatanga Village was constructed as a base for field studies of the depression. Exploratory drilling and sampling confirmed the practically inexhaustible reserves of the lonsdaleite mineralization. The settlement, Mayak, was subsequently built for expedition workers. Lonsdaleite-bearing rocks were transported several thousand miles by airplane to extract the mineral with strong acids. Presently, beneficiation techniques for lonsdaleite are being investigated.

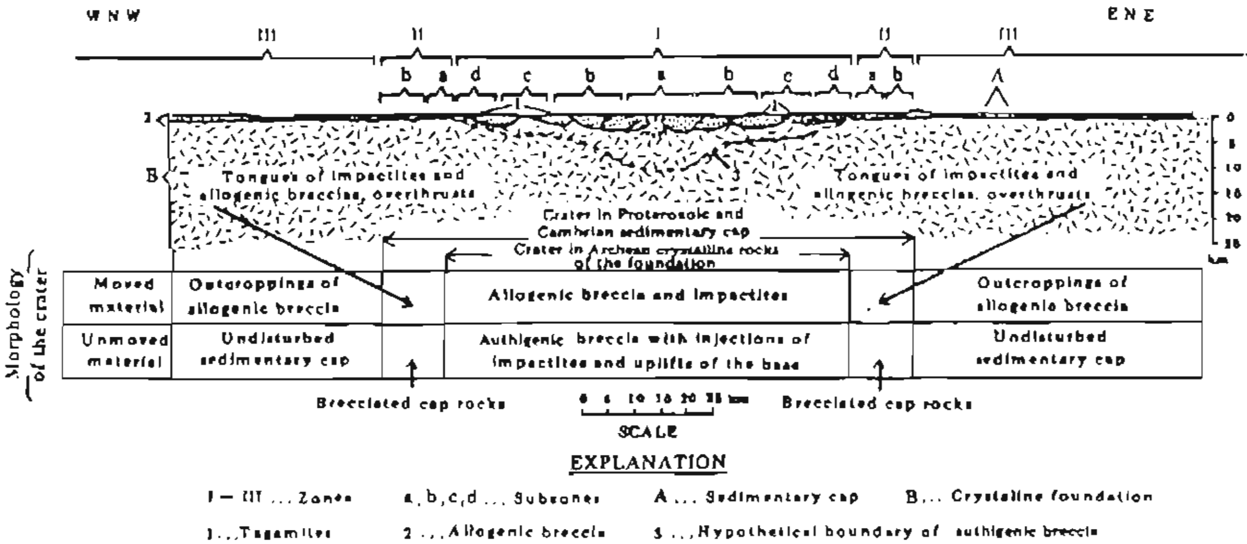


Figure 3. Diagram of the hypothetical Popigay meteorite crater. After Masaytis and others (1976).

AUTHORS POSITION AND RECOMMENDATIONS

The main arguments by proponents of the meteoritic impact theory are not single-valued. Masaytis' argument that shatter cones are proof of impact is inaccurate. Dietz (oral commun., 1984), probably the greatest supporter of astroblemes, has specimens of shatter cones from the Kovdor massif, Kola peninsula, and Trukhalev (oral commun., 1981) found the same structures in pipes located along the eastern boundary of the Anabar Shield. The presence of a dome, as in the central part of the Popigay depression, is typical of resurgent calderas. Even thrusts and overthrusts are usual for volcano-tectonic structures connected with viscous extrusive domes. Such structures, for example, are observed in association with Mt. St. Helens (Lipman and Mullineaux, 1982). The same may be said about minerals that are indicators of high pressure. For instance, pyrope occurs in Quaternary calc-alkaline rocks in Kamchatka; moissanite is found in dacitic pumice on Khangar volcano; and diamonds are found in basalt from Ichinksy volcano (Erlich and Gorshkov, 1979).

In addition, recent work testifies that the presence of diapletic glass in rocks is not unequivocally connected to meteoritic impact. It may also form by shock decompression under pressure of a few kilobars and temperatures above the atmospheric-pressure melting point but below the applied-pressure melting point. Chao and Bell (1969) have shown that glasses formed at pressures of 10 to 50 kbars will densify if quenched within about 30 s to temperatures below 400°C. Such quenching rates could be achieved in fragments or thin selvages by adiabatic cooling.

Moreover, there are several very strong geologic indications developed by Mezhvilk and Trukhalev (National Institute of Oceanic Geology, Leningrad) for a normal volcano-tectonic origin of the structure (Mezhvilk, 1981). They have shown that the Popigay depression is located on the intersection of a series of deep-seated faults, which have been continually reactivated from Proterozoic through Quaternary time. The structure was formed in several stages. Radiogenic data indicate two epochs of volcanism (Masaytis and others, 1976): the first around 29 to 31 m.y. ago (Oligocene) and the second around 41 to 46 m.y. ago (Eocene). In addition, Zhabin (1969) suggested the possibility of a Quaternary stage of volcanism in the depression. It should be noted that the two confirmed periods of volcanism had been documented in other areas of the Soviet Arctic. From data of Trukhalev (oral commun., 1981) the same radiogenic dates were obtained for rocks of the Kara depression in the Polar Urals. This depression is the same size as Popigay and contains similar internal structures and rock compositions (Ryskunov, 1939).

Several other geologic features suggest a volcano-tectonic origin. The first is a transverse fault, approximately east-west in strike. As is well known, such faults are typical of normal calderas and volcano-depressions (Smith and Bailey, 1968; Erlich and Gorshkov, 1979). They cannot logically be connected with meteoritic structures. The second feature is two concentric structures. Masaytis suggested that they were superimposed meteoritic craters, but the probability of this is extremely low. Furthermore, geophysical and drilling data of recent years show no evidence, down to 3,000 ft, of a lower boundary to the depression (Trukhalev, oral commun., 1981). Lastly, if the meteoritic impact theory is accepted, the absence of iron-nickel matter, typical of meteorites, must be explained.

All the above data support the theory that the Popigay structure is a specific type of volcano-tectonic depression. The high pressure associated with the formation of the structure is probably connected with the amount of volatiles in the magma during the structure's formation. By the same magmatic processes, a resurgent dome in the inner part of the structure could be formed. The Popigay depression is probably one of the greatest structures of this type on Earth and possibly the only one on an ancient platform.

This leads us to an important recommendation. While it is possible that the Popigay depression is the only volcano-tectonic depression in the world that contains lonsdaleite, such an assumption is highly improbable. There are, no doubt, more to be found. The appropriate literature should be examined for the presence of this hitherto unknown, but important, type of diamond.

ACKNOWLEDGMENTS

The authors appreciate the editorial assistance provided by P.A. Baily (Fulcrum Management, Inc.), G.H. Brodie (Diversified Resources Limited), P.W. Lipman (U.S. Geological Survey), and K.M. Goff (DGGS).

REFERENCES

Bundy, F.P., and Kasper, J.S., 1967, Hexagonal diamond, a new form of carbon: Journal Chemical Physics, v. 46, no. 9, p. 3437-3446.

Chao, E.T., and Bell, P.M., 1969, Annealing character of dense feldspar glass: Carnegie Institution Yearbook, v. 67, p. 126-130.
Erlich, E.N., and Gorshkov, G.S., eds., 1979, Quaternary volcanism and tectonics in Kamchatka (special volume): Bulletin Volcanologique, v. 42, no. 1-4, 298 p.
Fleischer, M., 1967, New mineral names: American Mineralogist, v. 52, p. 1379.
Frondel, C., and Marvin, R.B., 1967, Lonsdaleite, a hexagonal polymorph of diamond: Nature, v. 214, p. 587-589.
Golovanya, S.V., Kovostova, V.P., and Makarov, E.S., 1978, Hexagonal modification of diamond (lonsdaleite) in the eclogites of metamorphic complexes: Geochemistry International, no. 5, p. 82-84.
Hanneman, R.E., Strong, H.M., and Bundy, F.P., 1967, Hexagonal diamonds in meteorites: implications: Science, v. 155, p. 995-997.
Kaminsky, F.V., Blinova, G.K., Galimov, E.M., Gurkina, G.A., Kliuev, Y.A., Koptil' V.I., Krivonos, V.F., Frolova, L.N., and Khrenov, A.Y., 1985, Polycrystalline aggregates of diamond with lonsdaleite from placers in Yauktia: Mineralogical Journal, v. 7, no. 1, p. 27-36 (in Russian).
Lipman, P.W., and Mullineaux, D.R., eds., 1982, The 1980 eruptions of Mt. St. Helens, Washington: U.S. Geological Survey Professional Paper 1250, 844 p.
Lonsdale, Kathleen, 1971, Formation of lonsdaleite from single-crystal graphite: American Mineralogist, v. 56, p. 333-336.
Masaytis, V.L., Mikhaylov, M.V., and Selivanovskaya, T.V., 1976, The Popigay meteorite crater: NASA-TT-F-16900, Science Translation Service, 171 p.
Mezhvilk, A.A., 1981, Popigay volcano-tectonic structure: U.S.S.R., Izvestya Academy of Science, Geology Series, no. 6, p. 19-24.
Miliuvina, V.A., 1977, Lonsdaleite as an indicator of conditions of growth of natural diamonds, in Petrologiya i geohiiya kimberlitovikh rasplavov na rannikh etapakh evolutsii, Yakutsk, p. 125-147 (in Russian).
Polyakov, M.M., and Trukhalev, A.I., 1975, Popigay volcano-tectonic ring structure: International Geology Review, v. 17, no. 7, p. 1027-1034.
Roberts, W.L., Rapp, G.R., and Weber, J., 1975, Encyclopedia of minerals: Van Nostrand Reinhold, p. 363.
Ryskunov, I.L., 1939, Young volcanogenic formations of Pay Khoye: Problemy Arktiki, no. 9, p. unknown (in Russian).
Smith, R.L., and Bailey, R.A., 1968, Resurgent cauldrons: Geological Society of America Memoir 116, p. 613-662.
Sokhor, M.I., Polkanov, Y.A., and Yeremenko, G.K., 1973, A find of the hexagonal polymorph of diamond (lonsdaleite) in placers: U.S.S.R, Doklady Academy of Science, Earth Science Section, v. 209, p. 118-121.
Sozin, Y.I., Beliankina, A.V., and Svirid, A.A., 1983, Substructure of diamond and lonsdaleite in diamonds formed by dynamic synthesis: Svrkhtverdiye Materialy, no. 1, p. 11-21 (in Russian).
Vdovykin, G.P., 1970, Presence de diamants dans les meteorites de North Haig et Dingo Pup Donga, Australie Occidentale [Presence of diamonds in meteorites in North Haig and Dingo Pup Donga, eastern Australia]: Paris, C.R. Academy of Science, ser. D, v. 271, p. 2225-2227.
Zhabin, V.V., 1969, A finding of Cenozoic effusives on the north slope of the Anabar Massiff: Bulletin of Scientific Technical Information VIEMS, Series of Regional Geology and Methods of Geologic Mapping, no. 2, p. 22-23 (in Russian).

Chao, E.T., and Bell, P.M., 1969, Annealing character of dense feldspar glass: Carnegie Institution Yearbook, v. 67, p. 126-130.
Erlich, E.N., and Gorshkov, G.S., eds., 1979, Quaternary volcanism and tectonics in Kamchatka (special volume): Bulletin Volcanologique, v. 42, no. 1-4, 298 p.
Fleischer, M., 1967, New mineral names: American Mineralogist, v. 52, p. 1379.
Frondel, C., and Marvin, R.B., 1967, Lonsdaleite, a hexagonal polymorph of diamond: Nature, v. 214, p. 587-589.
Golovanya, S.V., Kovostova, V.P., and Makarov, E.S., 1978, Hexagonal modification of diamond (lonsdaleite) in the eclogites of metamorphic complexes: Geochemistry International, no. 5, p. 82-84.
Hanneman, R.E., Strong, H.M., and Bundy, F.P., 1967, Hexagonal diamonds in meteorites: implications: Science, v. 155, p. 995-997.
Kaminsky, F.V., Blinova, G.K., Galimov, E.M., Gurkina, G.A., Kliuev, Y.A., Koptil' V.I., Krivonos, V.F., Frolova, L.N., and Khrenov, A.Y., 1985, Polycrystalline aggregates of diamond with lonsdaleite from placers in Yauktia: Mineralogical Journal, v. 7, no. 1, p. 27-36 (in Russian).
Lipman, P.W., and Mullineaux, D.R., eds., 1982, The 1980 eruptions of Mt. St. Helens, Washington: U.S. Geological Survey Professional Paper 1250, 844 p.
Lonsdale, Kathleen, 1971, Formation of lonsdaleite from single-crystal graphite: American Mineralogist, v. 56, p. 333-336.
Masaytis, V.L., Mikhaylov, M.V., and Selivanovskaya, T.V., 1976, The Popigay meteorite crater: NASA-TT-F-16900, Science Translation Service, 171 p.
Mezhvilk, A.A., 1981, Popigay volcano-tectonic structure: U.S.S.R., Izvestya Academy of Science, Geology Series, no. 6, p. 19-24.
Miliuvina, V.A., 1977, Lonsdaleite as an indicator of conditions of growth of natural diamonds, in Petrologiya i geohiiya kimberlitovikh rasplavov na rannikh etapakh evolutsii, Yakutsk, p. 125-147 (in Russian).
Polyakov, M.M., and Trukhalev, A.I., 1975, Popigay volcano-tectonic ring structure: International Geology Review, v. 17, no. 7, p. 1027-1034.
Roberts, W.L., Rapp, G.R., and Weber, J., 1975, Encyclopedia of minerals: Van Nostrand Reinhold, p. 363.
Ryskunov, I.L., 1939, Young volcanogenic formations of Pay Khoye: Problemy Arktiki, no. 9, p. unknown (in Russian).
Smith, R.L., and Bailey, R.A., 1968, Resurgent cauldrons: Geological Society of America Memoir 116, p. 613-662.
Sokhor, M.I., Polkanov, Y.A., and Yeremenko, G.K., 1973, A find of the hexagonal polymorph of diamond (lonsdaleite) in placers: U.S.S.R, Doklady Academy of Science, Earth Science Section, v. 209, p. 118-121.
Sozin, Y.I., Beliankina, A.V., and Svirid, A.A., 1983, Substructure of diamond and lonsdaleite in diamonds formed by dynamic synthesis: Svrkhtverdiye Materialy, no. 1, p. 11-21 (in Russian).
Vdovykin, G.P., 1970, Presence de diamants dans les meteorites de North Haig et Dingo Pup Donga, Australie Occidentale [Presence of diamonds in meteorites in North Haig and Dingo Pup Donga, eastern Australia]: Paris, C.R. Academy of Science, ser. D, v. 271, p. 2225-2227.
Zhabin, V.V., 1969, A finding of Cenozoic effusives on the north slope of the Anabar Massiff: Bulletin of Scientific Technical Information VIEMS, Series of Regional Geology and Methods of Geologic Mapping, no. 2, p. 22-23 (in Russian).