Slime-cement fill as a stope support for gold mines

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SYNOPSIS

Preliminary tests on cement-consolidated gold-plant slime are outlined, together with preliminary underground tests on the behaviour of two paddocks of the material. Further large-scale testing of the same material pumped to the underground workings from a batching plant on surface is described.

SAMEVATTING

Die voorlopige toetse wat uitgevoer is met uitskotgoudslik wat met sement gekonsolideer is word beskryf, asook die eerste ondergrondse toetse waartydens die gedrag van die materiaal in twee hokke ondersoek is. Verdere grootskaalse toetse met dieselfde materiaal wat van 'n bogrondse mengaanleg na ondergrondse werkplekke gepomp word, word ook beskryf.

Introduction

All the reasons for seeking an effective and economically viable alternative to conventional timber support for use in large tabular underground excavations are too well known to merit reiteration.

The pumping of cement-consolidated slime into stopes forms part of a programme to reduce the labour involved in the handling of support. As the concept is new to South African gold mining, the project directors have proceeded cautiously. The stage has now been reached where a considerable amount of experience has been acquired in the pumping and placing of the slime. Knowledge of the behaviour of the *in situ* material must still be obtained and is the subject of further research.

The availability of an effective operating grout-pumping station, plus the planned extraction of a subvertical shaft pillar on Stilfontein Gold Mine, presented possibilities worth pursuing. The depth of the pillar was such that there would be no necessity for an intermediate underground pumping station, and the ventilation seals together with the required regional support of the inner pillar area would permit reasonably easy placement and would allow the behaviour of the consolidated slime to be observed under stress conditions.

Preliminary Investigation

The concept of pumping support into a stope has many advantages over the conventional support system, especially in times of labour shortage. The idea was first investigated in 1974, when a feasibility study showed that it could be economically viable. However, there were many gaps in the technology, which necessitated many preliminary tests before a full-size test could be launched.

Ideal Stoping Layout

It was necessary, before the feasibility study was done, to establish what stope support system would be used. The layout shown in Fig. 1 was considered to be ideal.

The support in the back area would comprise strike and dip walls with a minimum width-to-height ratio of 3:1. Two possible arrangements of these are shown in Fig. 1. The walls would be reinforced laterally by scraper rope to supply the lateral restraint necessary. The configuration shown in Fig. 1 has about 20 per cent of the stoped area taken up by support. Face support would comprise either mine poles or steel props, which would also support the blast barricade. The support along the gully would comprise timber packs, which would also help to support one side of the cement–slime rib.

This support system would involve considerably less material-handling, with a consequent saving in labour.

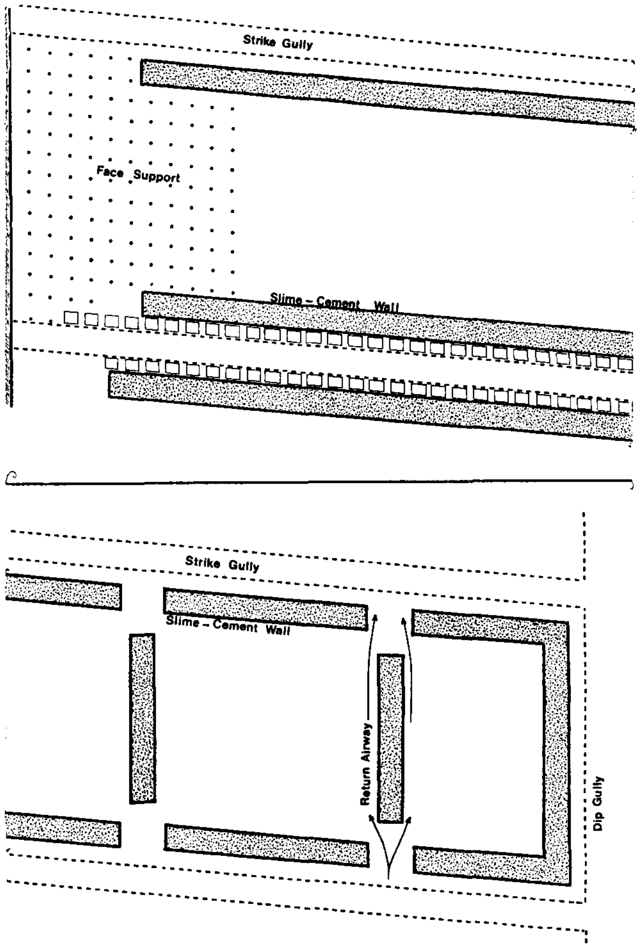

Fig. 1—Possible wall configurations
Upper: Walls designed to seal off back area completely
Lower: Walls designed to leave return airways after stoping has been completed

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*General Mining & Finance Corporation Limited, Johannesburg.

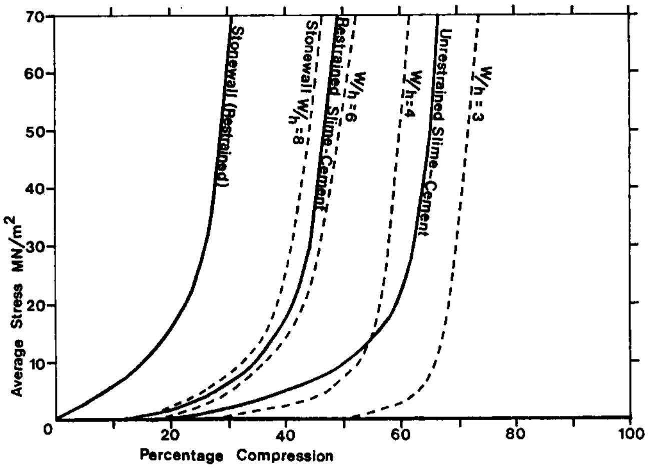

Fig. 2—Comparison of slime–cement and stonewall models

Other advantages are the effective sealing off of back areas to minimize heat pick-up and a considerable reduction in the fire hazard.

Fig. 2 shows graphs of the average stress versus the percentage compression for restrained and unrestrained slime–cement walls[1]; these are superimposed on the curves for stone walls presented by Cook *et al*[2]. The two groups of curves are very similar in shape, and it is therefore probable that slime–cement walls and stone walls act in a very similar way.

However, there is a significant difference in the width-to-height ratio. In tests on slime–cement models, there was no significant difference in the stress generated between specimens with a width-to-height ratio of 3:1 and those with a ratio of 6:1. Wide stone walls are significantly stronger than narrow stone walls.

Slime–cement walls can safely be assumed to provide an effective stoping width of approximately 40 per cent of the original stoping width, which has the effect of reducing the energy release rate by a corresponding amount. The slime–cement walls are visualized as occupying 20 per cent of the stoped area. The benefit derived from this was shown[2] to be 95 per cent of the benefit that would be derived if 100 per cent of the area were filled. The system is therefore ideally suited to mechanized stoping at depth in narrow tabular ore-bodies.

Pumping Tests

Preliminary pumping tests conducted by the contractor who was supplying the slime–cement underground indicated that a relative density of 1,6 was about correct. Beyond this, the friction head increased considerably in relation to the relative density.

Laboratory Tests

A series of tests was conducted by the Faculty of Civil Engineering of the University of the Witwatersrand on both fresh and weathered slime with a variety of cementing agents and varying percentages of water. The main points arising from the tests[1] are as follows.

(a) As coarse a material as possible should be used because it requires less water to form a pumpable slurry and, when cemented, has superior strength characteristics.

(b) A variety of cementing agents such as flyash, gypsum, and lime give results equal or superior to those produced by more expensive Portland cement.

(c) If sufficient cementing agent is used, a pumped cemented-slurry fill can be more effective than a broken-rock fill.

(d) Lateral restraint of a fill considerably improves its percentage compression characteristic. Two-way reinforcement of fills using wire-rope scrap appears to offer a feasible way of providing lateral restraint in practice.

(e) If liquefaction of a cemented fill is likely to cause problems, use should be made of coarser materials, which have been shown to be less susceptible to liquefaction than are fine materials.

(f) There is no difference between the stress generated by a test specimen with a width-to-height ratio of 3:1 and that with a ratio of 6:1.

Underground Tests

A currently working stope 1900 m below surface was used for the initial tests. Two paddocks were constructed and filled when the working face was 4 m away. (The face was not being advanced at the time.) Run-of-mine tailings were transported underground to a mixing station, and 10 per cent by mass of cement was added together with sufficient water to provide a pumpable mixture. The slurry was then pumped into the paddocks until they were full, the temporary pumping station being situated in a cross-cut some 500 m away.

After six weeks, by which time the face was 8 m away, inspection of the paddocks revealed no damage from the blast, and a 10 per cent convergence had taken place. Three months after filling, a 35 per cent convergence had occurred, and, although there was considerable bulging at the sides, the retaining mesh of the paddock was still unbroken (Fig. 3). At that stage the face was 25 m from the paddocks, and there was considerable deterioration of the hangingwall around the walls, which appeared to indicate that the walls were setting up considerable distortion in the hanging around their edges.

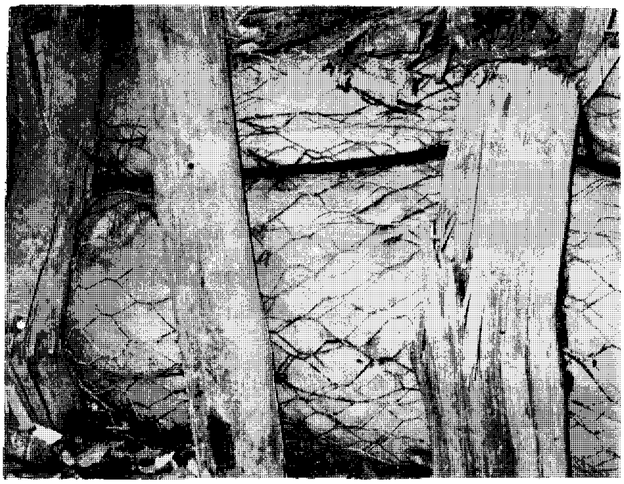

Fig. 3—Test paddock after 35 per cent compression

The discontinuation of work in this area prevented further observations being made.

Large-scale Tests

Choice of Site

The results of the preliminary tests were sufficiently promising to warrant a large-scale test. The availability of a stope in a shaft pillar, which it was planned to extract, presented an ideal site for such a test. The ventilation seals and the required waste walls could be replaced by the slime-cement walls, and a saving could be effected at the same time. Supervision and communication would be good and access for measurement ideal. The availability of a grout-pumping station on surface meant that the tests could be done with a minimum capital outlay and with considerable 'know-how' already available.

After an initial period of four months, when minor plant modifications were done, filling commenced in September 1977.

General Layout of Shaft Pillar

The shaft pillar consists of an area of approximately 63 000 ca, of which the area comprising the inner pillar is roughly 8500 ca. It is this latter area which surrounds the shaft itself, and the rock pass systems that involve the ventilation seals and the initial regional support. The general layout of the shaft-pillar area is shown in Fig. 4.

The reef body, dipping at about 10° from north-east to south-west, intersects the shaft 6 m below the 19 level station elevation, and at this point is 200 m below the sub-shaft and 1445 m below surface. Sub-development on the reef was done from an existing raise connection to the east of the block in order to hole into, and stope around, the shaft and rock passes. This permitted the sealing and initial support work to be done.

Pumping Layout

Experimentation was done in an attempt to obtain a coarser product by the use of cyclones, but it was apparent that the existing columns would have to be replaced by high-pressure piping. Not only did this involve considerable expense, but the time factor was also critical.

The layout consists basically of pumping run-of-mill tailings from the gold plant to the pumping-station storage tank. In the tank, a flocculant is added and the slime dewatered. Messrs Rodio S A Pty Ltd, to whom the pumping station belongs, kindly undertook to test flocculants in order to find a suitable one.

After dewatering and secondary mixing, during which 10 per cent by mass of cement is added, the resultant slurry with a relative density of 1,57 is pumped by means of a plunger pump through the 3500 m long steam pipe (25 mm diameter) to the paddocks. Fig. 5 shows the schematic layout of the pumping arrangements.

A standby pump is positioned on 19 level station. If it becomes necessary to flush the column in a breakdown, this pump can be brought into operation rapidly. Telephonic communication is provided from the placing site to the surface pumping station.

As this was a pilot scheme, it was decided not to enlarge the existing tank-storage capacity, but to make do with the minimum of renovations and alterations to the existing plant.

Filling Operations

A 24-hour cycle is being operated as follows for the filling of the paddocks: 5 hours are required to fill the storage tank, total capacity 64,2 m^3, of which the bottom 0,6 m involving 11,31 m^3 is 'dead', being below the discharge point; 1 hour is required for the adding and mixing of the flocculant, 10 hours for settling of the slurry, 1 hour for dewatering, 1 hour for re-mixing, and

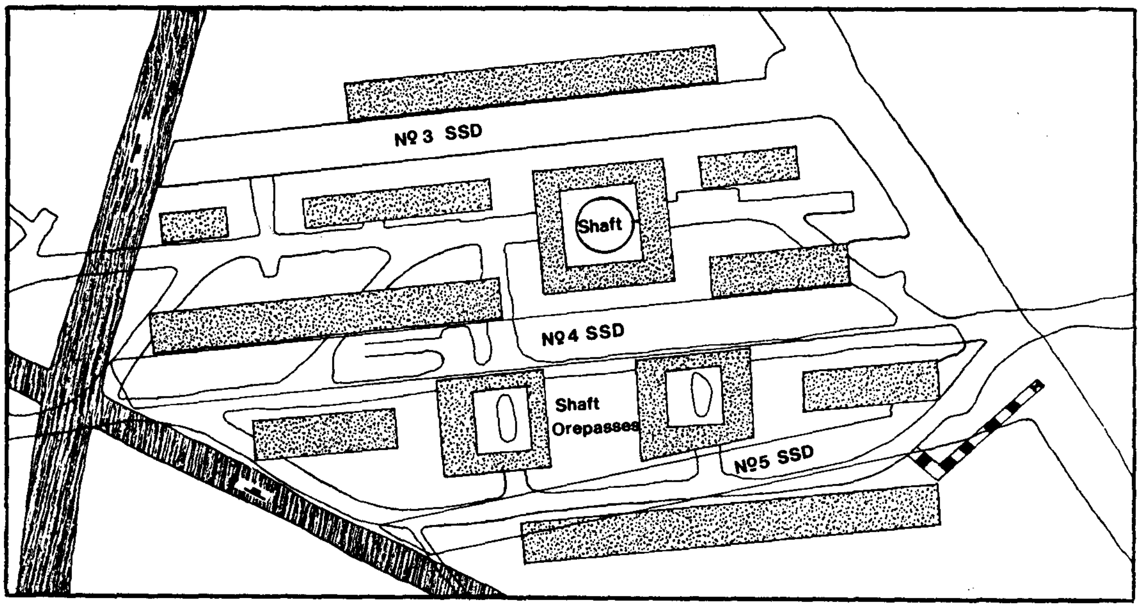

Fig. 4—Plan of proposed slime-cement walls in the vicinity of the shaft

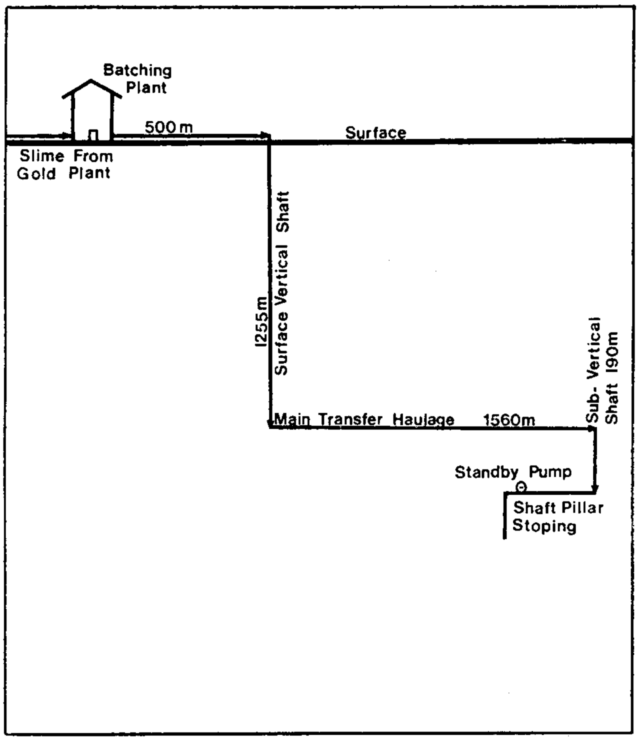

Fig. 5—Schematic layout of the pumping system

the remaining 6 hours for pumping at a delivery rate of about 4 m^3/h.

This cycle is based on the received slime from the gold plant having a relative density of 1,3. If the relative density is higher, the settling time is reduced and the quantity of slurry available for pumping is higher. A volume of 32,4 m^3 has been achieved during one day. The paddocks are kept under close surveillance during the filling operations, and a bypass column is available for use should the necessity arise. The feed column is tested by the pumping of water through it prior to each filling operation, and is completely flushed out at the end of the slurry pumping. These two operations require about 35 minutes each and are dovetailed into the cycle.

In this area, with a minimum lapse of 18 hours between placings, all the superficial moisture evaporates and the mixture sets to a consistency that will support the weight of a man with the minimum of indentations.

Paddock Construction

The laid-down support pattern of this area consists of regularly spaced pipe-sticks interspersed with 80 cm solid wattle packs and a stipulated maximum stoping width of 1,2 m. Faulting coupled with friable hangingwall conditions makes the provision of stoping width difficult to maintain. Crush pillars have also been left on the perimeters of the ventilation seal areas.

The paddocks are constructed so as to fit in with the support pattern. Their retaining walls are made up of unlaminated polypropylene cloth glued to the hanging and footwall, backed by a sturdy wire mesh that is nailed to the packs and regularly spaced vertical wooden poles between the packs. Split lagging between the poles and packs permits stapling of both wire mesh and cloth, and thereby obviates excessive bulging or tearing of the latter during filling operations. Figs. 6 and 7 show the construction of a paddock.

Lateral restraint is supplied by a lattice of tensioned, discarded wire rope between opposite poles. Paddocks are sub-divided so that possible air spaces produced by irregularities in the hangingwall can be eliminated. Fractures developing by sagging hangingwall, which are liable to result in leakages of slurry, are sealed with cement. The nature of the ground makes it imperative that the building and filling of paddocks should be concurrent with the stoping operations.

Instrumentation

The following instruments were installed within the cement–slime seal surrounding the shaft itself:

(1) convergence meters,
(2) lateral-displacement meters comprising wires contained within telescopic pipes,
(3) piezometers, and
(4) sets of three flat jacks each, with two installed vertically on dip and strike to measure the horizontal pressure and a third placed horizontally to measure the vertical pressure.

In addition, three convergence-ride meters were installed around the circumference of the shaft to measure the ride and convergence outside the fill.

To date 10 per cent convergence has occurred in the vicinity of the shaft. The flat jacks are registering a ratio of vertical pressure to horizontal pressure of 3:1, and the

Fig. 6—Paddock construction showing the retaining mesh and split lagging supported on mine poles

Fig. 7—The inside of a paddock showing polypropylene filter cloth and wire-rope reinforcing

piezometers have registered no fluid pressure. There is very little bulging on the slime–cement walls.

Costs

Current Operation

The seals around the shaft and the two orepasses required a total of 900 m^3 or 1404 t of slurry, of which 800 t were solid tailings and 80 t cement.

The breakdown of these costs is as follows:

	R/t placed	R/m^3 filled
(a) Paddock construction . .	2,70	2,65
(b) Unskilled labour	5,60	5,51
(c) Supervision	8,91	8,76
(d) Flocculant	0,37	0,36
(e) Cement	2,38	2,34
(f) Pumping-plant amortization and contract payments (includes power, water, and transportation)	8,17	8,04
TOTAL . . .	28,13	27,66

The amortization of the pump column is estimated at 5c per ton placed.

A Large-scale Operation

The effect of the scale of operations on the cost structure is shown by the following breakdown, which is an estimate for a plant supplying 7000 m^3 per month.

	R/m^3 filled
(a) Paddock construction	2,65
(b) Unskilled labour	2,47
(c) Supervision	1,00
(d) Flocculant	0,40
(e) Cement	2,34
(f) Pumping-plant amortization and contract payments	6,00
TOTAL . . .	14,86

These estimates are based on the following:

(i) a surface pumping plant at R150 000,
(ii) an underground relay station at R50 000,
(iii) pumping and placing on two 6-hour shifts for 24 hours,
(iv) 20 effective pumping days per month,
(v) two 40 mm-diameter high-pressure seamless delivery columns from surface to the relay station,
(vi) eight placing columns to sites and steam piping with a delivery rate of about 4 m^3/h.

Conclusions

(1) It is technically feasible to pump a slime–cement mixture underground and to confine it in such a way that it can replace waste walls in stopes.
(2) The behaviour of the slime–cement material in a paddock appears to be superior to the behaviour of test specimens in the laboratory. A detailed study of the *in situ* behaviour of the slime–cement will have to be made.
(3) The cost of the operation is considerably higher than the feasibility study showed.
(4) Liquefaction does not appear to be a problem. The fractured hangingwall and footwall probably provide additional drainage.
(5) Slime–cement walls provide an excellent ventilation seal and are much cheaper than vermiculite seals.
(6) Woven unlaminated polypropylene provides an ideal filter material for the cement–slime fill.
(7) That each daily placement of slime in the paddock areas produces a relatively slow increase in the height of the slime has a twofold effect: firstly, drainage through the polypropylene is facilitated and, secondly, any possibility that the material will tear is obviated.

References

1. Blight, G. E., More O'Ferrall, R. C., and Avalle, D. L. Properties of high water content cemented tailings used as fill in mining excavations.
2. Cook, N., *et al.* Practical rock mechanics for gold mining.

Discussion of the previous paper

A. WHILLIER*

An additional benefit that is often overlooked when methods for the backfilling of stopes in deep gold mines are being considered is the saving in refrigeration. A significant fraction of the total amount of heat that enters deep mines arises from the exposed rock in back areas; this heat flow can be prevented only by ensuring that there is zero air flow through such zones. The magnitude of the benefit is best illustrated by means of an example.

In a shaft breaking 8000 m^2 of reef per month in rock at 55 °C, the total heat load will be about 10 MW, and practically all of this will have to be removed by refrigeration. Three-quarters of this heat arises from the rock. If backfilling can be introduced into, say, half of the back areas (involving the filling each month of about 4000 m^2 of stoped-out area, i.e. about 8000 t per month), the reduction in heat load would be at least 1200 kW. The annual financial saving because of the reduced requirement for refrigeration would be about $1200 \times 160 =$ R192 000, which is equivalent to

192 000/(4000 × 12) = R4 per m^2 that is backfilled,
or R2 per m^2 of reef that is mined,
or 50 cents per ton of reef mined.

My message to the mining community, therefore, is to get on with it: find practical methods for backfilling, and the reduced cooling costs will cover a significant portion of the cost of backfilling.

There is a further, and perhaps even more cogent, reason why backfilling is a vital necessity for mining in rock that is hotter than about 55 °C. At these high rock temperatures, the amount of refrigeration that must be distributed into stopes will be so great that it is almost impossible to achieve this distribution effectively. The result is that it is going to be extremely difficult to keep wet-bulb temperatures below about 31 °C everywhere in these hot mines, unless backfilling is done as a routine. Furthermore, there is no chance that wet-bulb temperatures can be kept below 28 °C unless backfilling is adopted in such mines. The problem is so serious that we are even contemplating backfilling with a lightweight foam, although a backfill that has structural integrity is obviously much preferred.

A further benefit when stopes are backfilled is that no additional ventilation control walls are needed, and, except for occasional brattices, no particular care is necessary in the control of ventilation within the stope itself. The saving in labour from this score can be significant, yielding a useful increase in the tonnage of rock broken per man.

With stopes that are backfilled, there is considerable reduction in the fire hazard. The only way in which fires could spread would be along the timber support that lines the dip gullies. Fortunately, there are effective methods for preventing the rapid spread of fires in gully support timber.

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Deep-ocean mining

The theme of the 1978 Underwater Mining Institute, which is to be held at San Diego (California) on 19th and 20th October, 1978, is 'Deep-ocean Mining'. There will be a heavy emphasis on nodules, including resource assessment, composition, ore-grade assessment, and processing technology. Other topics will include phosphorites and bulk mineral resources, and a keynote address on the future of deep-ocean mining. For information, write to either of the following:

Gregory D. Hedden, Institute Coordinator, University of Wisconsin, Sea Grant Advisory Services, 1815 University Avenue, Madison, Wisconsin 53706, U.S.A.

J. Robert Moore, Program Chairman, University of Alaska, Institute of Marine Science, Fairbanks, Alaska 99701, U.S.A.

Offshore minerals

An international seminar on offshore mineral resources is to be held in Orleans (France) from 23rd to 27th October, 1978. The topics to be discussed include aggregates, placer deposits, phosphorites, metalliferous brines, and manganese modules. The various aspects of each subject — prospecting, mining, economic, technological, and legal problems — will be considered and discussed by professionals.

The seminar is being organized by GERMINAL (Groupe d'Etude et de Recherche de Minéralisations au Large), which was created in July 1973 and includes twelve European members (civil engineering, mining, dredging and shipbuilding companies, banking groups, state organizations). The objective of GERMINAL is the study and development of offshore mineral resources.

Further information is obtainable from Mr L. Galtier, c/o GERMINAL, B.P. 6009, 45018 Orleans Cedex, France.