284001

STOREYS CREEK TIN MINING COMPANY N.L.

DORSET KAOLIN DIVISION

MT CAMERON KAOLIN DEPOSITS

63_355

AMG REFERENCE POINTS ADDED

284002

STOREYS CREEK TIN MINING COMPNAY N.L.

DORSET KAOLIN DIVISION

Report on the establishment of a plant
at South Mount Cameron, North Eastern
Tasmania, for the production of Kaolin
for fillers in the manufacture of fine papers.

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DORSET KAOLIN DIVISION

SOUTH MOUNT CAMERON

NORTH EASTERN TASMANIA

Prepared by :

H. Keith Turner

April, 1963.

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Acknowledgment is made to the Director and Officers of the Tasmanian Mines Department, to the General Superintendent and Technical Staff of Associated Pulp & Paper Mills Ltd. to Mr. W.H. Cropp and Mr. J. Volker.

Their support and interest has been of immeasurable assistance in bringing the planning of this Project to its present advanced stage.

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DORSET KAOLIN DIVISION

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DORSET KAOLIN DIVISION

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SCHEDULE A. Copy of letter from A.P.P.M.Ltd., Burnie, Tasmania - dated 25th May, 1962.

B. Report by R. Hare & Associates Estimates of reserves of Kaolinitic Clay in the South Mount Cameron Area, Tasmania, 4th July, 1961.

C. Report by the Department of Mines Laboratory of Tasmania Ore dressing investigation R 361, R 364, R 365, R 366 and R 368 20th April, 1961.

D. Report by W.H. Cropp - February, 1963 "The design of Treatment Plant to produce paper quality Kaolin from South Mount Cameron Raw Clay."

E. Flow Sheet and Plant Layout Plan to accompany Schedule C. by W.H.Cropp (2 parts)

F. Chart - Establishment of Operations

G. Plan of Brown's Area, South Mount Cameron.

H. Plan of Scotts Area, South Mount Cameron

I. Plan of South Mount Cameron Area shewing leases etc. - 1" = 10 chains

DORSET KAOLIN DIVISION

Summary & Conclusions.

1. Profit Summary and return on Capital Investment

Assume (a) 15 years life @ 15000 tons
(b) Repayment of Loan Funds £60,000 over 10 years at 5%
(c) Taxation at current rate

Value of Production		£4,612,500
Deduct -	Operating Costs	
	Loans & Interest	
	Income Tax	
	Redemption Working Capital & Preliminary Expenses.	3,909,960
	Nett return	£702,540

p.ton
$ 41

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Annual Return = £46,833
= 49.91% on Equity Capital

2. Provision of Funds and Capital Construction

Cash Capital required	£168,850.
Provision :	
Equity Capital	93,850
Loan from Tasmanian Government	60,000
Loan from Bankers	15,000
	£168,850

3. Raw Clay Reserves

Two areas, Browns and Scotts, have been carefully tested and proved reserves total 190,000 tons of finished clay as paper fillers.
Indicated reserves at the YZ area from a limited drilling and sampling are of the order 200,000 tons of finished paper filling clays.

4. <u>Prices for fillers</u>
A growing market exists with A.P.P.M Ltd. at Burnie and in the near future, requirements of the order 14,000 - 15,000 tons are soundly indicated.
The agreed price of £20.10.0 per ton of filler clays as per specification has been applied in this report.

5. <u>Services and Manpower</u>
The new organization related to the well established Dorset Tin Division in the same area is of importance.

6. <u>Planning</u> for this establishment is well advanced.

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DORSET KAOLIN DIVISION

PREAMBLE

A.P.P.M. Ltd. at Burnie in Northern Tasmania are the largest producers of fine papers in Australia. The industry was established about thirty years ago and draws its chief raw material from the splendid forests in the high country to the south of Burnie.

The present production of fine paper is in excess of 70,000 tons per annum and steadily rising. New paper-making machines have recently been installed and capacity is perhaps as high as 150,000 tons per annum.

For filling purposes, clay is imported from Cornwall in bulk. Coating of these papers is completed at Ballarat in Victoria where A.P.P.M.Ltd. own and operate a clay deposit similar in many respects to the deposits at South Mt.Cameron in Tasmania - those deposits to which this report refers. It is entirely probable that the coating operation of the filled paper will eventually be done at Burnie, and this could be influenced by a supply source of coating grade clay in Tasmania.

A.P.P.M.Ltd., from their Ballarat plant, also supply to A.P.M.Ltd. of Melbourne their principal clay requirements.

The present consumption of filler clays at Burnie, all of which are now imported, is around 15,000 tons per annum and rising with the steadily increasing paper requirements in Australia - filler clay usage is around 15% of the weight of finished paper produced.

A.P.P.M. Ltd., over the past twenty years, investigated several clay sources in Tasmania, but were unsuccessful in locating deposits with suitable characteristics.

The areas referred to in this report in N.E.Tasmania have been discovered in the past ten years, and the extensive research work completed by the Mines Department Laboratories at Launceston has established the treatment required to produce both filler and coating grades of clay.

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MINING TITLES

The following titles are held by Storeys Creek Tin Mining Company N.L. -

1. SCOTTS AREA - Mining Lease 8M/62
   of 40 acres
   at South Mount Cameron.

2. YZ AREA - Mining Lease 9M/62
   of 40 acres
   at Gladstone.

3. BROWNS AREAS - Options over
   (a) Mining Lease 12M/59
   of 5 acres
   in the names of
   G.E. Brown & M.L.Watt.

   (b) Mining Lease 7M/59
   of 10 acres
   in the names of E.E.Brown
   and M.L. Watt.

Total Purchase Price of the above leases in Brown's Areas £7,500.

DORSET KAOLIN DIVISION

Testing of Kaolin Deposits

1. The first testing work completed was on Brown's Areas - M.L. 12M/59, during 1959.
Drilling in these areas was carried out with closed type auger which is used extensively on beach sand deposits. The tool is hand operated and provides an excellent sample from a known horizon without contamination.
Around 25 feet in depth was maximum at this stage the rate of progress was slow and uneconomic.
Samples were cut from 5' drives and sent forward to Burnie - A.P.P.M. Ltd. being the sponsors of this programme.
The laboratory work on these samples consisted of -
    (a) Determination of particle size
    (b) Percentage of minus 30 micron material
    (c) Ignition loss of (b)
    (d) Brightness of (b)
    (e) Yield of suitable clay.

2. Storeys Creek Tin Mining Company N.L. through Dorset Tin Division operating in the area entered the field in 1960 and applied several drilling methods, including a 16" Conrad plant, using casing, to determine variations over a larger cross-section.
In 1961, a special plant was designed and built at Dorset - a mobile unit powered by a Winsconsin petrol motor and using a special earth socket - diameter of 5" - This was the most suitable tool employed.
Samples were taken over 5' drives and in the latter stages of the programme, checks were made over sections of 25' in Scott's areas.

3. A scout boring programme of an area of some 50 square miles has indicated several important targets for future exploration.

DORSET KAOLIN DIVISION

Reserves of Crude Clay in the South Mt. Cameron Area.

Attached hereto is a report (Schedule B), by R. Hare & Associates on the above. Investigation of the areas show considerable potential and refer to filler grade material. The YZ and other areas have been prospected only and are located within a few miles of the proposed treatment plant which will draw supplies initially from Scott's and Brown's areas.

Mining Leases, etc., are held as :-

(a) 40 acres Scott's area.

(b) 40 acres YZ area.

(c) Options over 15 acres leases Brown's area. Prospecting Rights over 100 acres adjacent Brown's areas.

Proved reserves are conservatively stated at 190,000 tons of finished clay.

DORSET KAOLIN DIVISION

Research and Related Aspects.

1. The initial research work on the South Mt. Cameron clays was completed by A.P.P.M. Ltd. in their laboratories at Burnie.
Bleaching aspects were part of this study and paper made by hand using chlorine bleached Kaolin, after several years, has shewn no regression of brightness.

2. Arising from Storey's Creek's interest in the clay areas and their request to the Director of Mines, the Mines Department Laboratories at Launceston, under Mr. Walter Manson, completed a most extensive examination - their report (Schedule C) is filed herewith.

Further studies have covered other technical aspects including economics of drying; which is the most expensive section of production.

Free chlorine as a satisfactory bleach for minor organic matter has been fully established.

3. Drying techniques - The application of spray drying in the Georgia Kaolin areas of the United States was studied by Mr. Walter Manson on a recent visit to those areas, and consideration is at this stage being given to this method in the present flowsheet, replacing steam and conveyor type dryer and limiting the usage of flocculents.

4. With the establishment of the South Mount Cameron operation research laboratories will be set up at the Mine

DORSET KAOLIN DIVISION

MARKETS AND PRICES FOR FILLER AND COATING GRADES OF PAPER CLAYS.

1. Reference is made to the attached copy of letter from A.P.P.M. Ltd., offering £20.10.0 per ton for filler clays as per specification, delivered Burnie in paper bags. (Schedule A). This letter is dated 25th May, 1962, and indicates a firm market for 10,000 - 11,000 tons per annum.

   At the present time, following installation of a new paper making machine, production of fine paper is increasing and of the order 100,000 tons per annum. Filler clay requirements are of the order 15,000 - 16,000 tons per annum, portion of which is met from A.P.P.M.'s clay plant at Ballarat - these fillers being the "seconds" after stripping-out with a centrifuge the coating clays required for their own papers and for the market with A.P.M. Ltd. also in Victoria.

   Recent advices from Mr. Cornelius, General Superintendent of A.P.P.M. Ltd. at Burnie confirm his letter of 25th May, 1962.

2. It is considered that the market for filler clays with A.P.P.M. Ltd. is the first target.

3. Markets for coating clays exist in Japan and New Zealand - the typical price being £30 - £40 per ton F.O.B.

4. Additional plant required for the production of coating grades could cost £50,000 and shew economics well in line with those of filler clays.

5. It is evident that markets within Australia will increase steadily in the future.

DORSET KAOLIN DIVISION

Estimated Capital Cost of Plant

(Full Capacity - 15,000 tons per week - Finished clay)

1.	Site preparation		2,000
2.	Water Supply		1,000
3.	Power Supply		1,000
4.	Roads		1,000
5.	Mining Equipment		
	2 Shovel Loaders } 1 5ton Truck }		14,500
6.	Degritting Section		
	Hopper & Conveyor belt	2,100	
	Clay Mill & agitator	2,400	
	Classifier (ex Broken Hill)	1,500	
	Trommel	600	
	Surge Tank & agitator	1 200	
	3 Hydro separators 20' dia.	12,000	
	4 pumps, motors & piping	1,750	
	1 x 250 mesh screen	2,000	
	Reagent feeders & tanks	1,000	24,550
7.	Bleach Section		
	Storage tank & agitator dia. 23' x 10'	3,500	
	4 tanks 13'6" x 10' with agitators	6,500	
	3 pumps, motors & pipes	1,500	11,500
8.	Filter Section		
	2 Heating tank 16'x 8'x 8' with agitators	3,000	
	Flocculent Mixer	1,000	
	2 Fienc Filters Complete total 600 sq. ft.	22,000	25,000
9.	Drying Section		
	Conveyor type dryer	15,000	
	Oil fired package (Powermaster) Boiler 10,000 lbs/hr	9,000	
	Conveyor - filter to dryer	800	
	Conveyor - dryer to bin	1,000	25,800
10.	Packing Section		
	Bins	2,500	
	Bagging Plant	2,000	
	Dust Collection	1,000	
	Fork lift truck	1,000	6,500

11.	Housing (transfer for Dorset Tin)		9,000
12.	Laboratory & Fittings		5,000
13.	Mine Buildings		
	Stockpile building 4,600 sq.ft.	6,000	
	Mill Housing	6,000	
	Framing & Stairway	2,000	
	Concrete floors etc.	2,000	16,000
14.	Administration & Contingencies		7,500
			£150,350

TOTAL CAPITAL REQUIREMENTS

A.	Mining & Treatment Plant	£150,350
B.	Acquisition of Brown's areas	7,500
C.	Preliminary Expenses	1,000
D.	Working Capital	10,000
		£168,850

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DORSET KAOLIN DIVISION

NOTES ON CAPITAL REQUIREMENTS

1. The estimated capital cost of plant required to provide 15,000 tons of finished clay in paper bags is shown at £146,350.
Reference is made to the Flow Sheet (Schedule E) which provides for production of 40 tons of finished clay per day. Estimates for plant and establishment are based on this layout.

2. A major difference in the Report of the Tasmanian Mines Department Laboratory and Mr. W.H. Cropp's Report is that the experimental work carried out by the Mines Department on degritting was based on the application of miniclones. Mr. Cropp has applied circular hydrosizers, for which £12,000 has been allocated in the capital structure. (Quotations for hydrosizers are from Dorr-Oliver).
Estimated cost of Miniclones and pumps to replace hydrosizers is £2,000.

3. Elenc Filters at £11,000 each installed, area 300 sq.ft., are currently under manufacture by C.P. & E. of Melbourne for titanium white industry being established in Western Australia - landed cost South Mt.Cameron complete with vacuum pumps is estimated at £10,000.

4. Other estimates are from quotation or from knowledge of similar equipment.

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Estimates and Production of Operating Costs
Based on 15,000 tons Finished Clay per Annum.

1.	Mining by excavator and transport to stockpile	1. 0. 0
2.	Degritting Bleaching Filtering Drying	6. 5. 0
3.	Paper Bags & Handling	2. 0. 0
4.	Transport to Railway	10. 0
5.	Rail Freight - Herrick to South Burnie	4. 0. 0
6.	Overheads and Administration	15. 0
	Per ton, delivered in paperbags at South Burnie Railway Station	£14. 10. 0

DORSET KAOLIN DIVISION

NOTES ON PRODUCTION COSTS.

1. Mining Costs are based on figures provided by Municipal Authorities in North Eastern Tasmania - the usage of granitic gravels is general in the area for maintenance and construction of by-ways. Overburden in the clay areas to be operated, is negligible.

2. Treatment Costs - reference is made to W.H.Cropp's attached report - page 11. Total costs of treatment, including amortization is £6.19.0 per ton of finished clay, on the basis of 40 tons per day in storage bins.

3. Transport and Freight Charges - at £4.10.0. is 5% above actual present figures.

4. Overheads etc. at 15/- per ton provides in excess of 10 per cent of other totalled costs.

DORSET KAOLIN DIVISION

ECONOMIC APPRAISAL

Based on proven reserves in excess of 15 years operations at a production rate of 15,000 tons of filler grade per annum.

Life of Proven Reserves -			15 years
Value of recoverable Kaolin 225,000 tons @ £20.10.0. per ton			£4,612,500
Deduct operating costs 225,000 tons @ £14.10.0. per ton			3,262,500
Operating Profit (gross)			£1,350,000
Deduct Interest on borrowed funds (5% on £60,000 repayable over 10 years)			16,500
Operating Profit (nett) before depreciation and Tax			£1,333,500
Less Income Tax			
Profit		£1,333,500	
Less Capital Expenditure		137,850	
Taxable income		£1,195,650	
Tax £ 75,000 x 7/-	= £ 26,250		
£1,120,650 x 8/-	= £448,268		
£1,195,650	£474,518		474,518
			£ 858,982
Deduct Repayment of Borrowings		£60,000	
Redemption of Equity Capital		93,850	153,850
			£ 705,132
Add Recoupment of Worker Capital		£10,000	
Residual Capital Value		20,000	30,000
Return on Investment			£ 735,132

Annual Return after Tax = £49,009

Percentage Annual Return on Equity Capital of £93,850 = 52.2%

DORSET KAOLIN DIVISION

PROGRAMME FOR ESTABLISHMENT OF OPERATIONS

May 1963 - Appointment of Plant Superintendent

June 1963 - Research Work at Mines Department Laboratories.
Preliminary design work.

July 1963 - Continuation of research work
Commencement of drafting at Phoenix Foundry, Launceston under direction of :-
Messrs. W.Manson & W.H.Cropp
N. Gerke, Chief Engineer
Aberfoyle Holdings Ltd.
and Plant Superintendent.

August 1963 -
Completion of design and drafting

September 1963 -
Orders for Plant preparation of tenders.

October 1963) Construction
November 1963) "
December 1963) "
January 1964) "

February 1964 - Production Operation.

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TELEPHONE: BURNIE 450
POST OFFICE BOX NO. 201

TELEGRAPHIC AND CABLE ADDRESS: "ASPULPACO" BURNIE

ASSOCIATED PULP AND PAPER MILLS LIMITED

Registered Office: 360 Collins St., Melbourne

MARINE TERRACE,
BURNIE, TASMANIA.

PLEASE ADDRESS CORRESPONDENCE TO THE COMPANY, NOT TO INDIVIDUALS AND IN REPLY QUOTE JC:MDJ

25th May, 1962.

Mr. K. W. Craig,
Storeys Creek Tin Mining Company N.L.,
100 Collins Street,
MELBOURNE, Victoria.

Dear Sir,

We refer to the discussions we have had with you from time to time about your proposal to set up treatment plant and produce clay for use in papermaking from your clay leases in N.E. Tasmania.

We have provided you with a specification for a suitable clay and you have informed us that you confidently anticipate being able to supply clay of this quality delivered Burnie at a price comparable with the landed cost of similar quality English clay, which is £20/10/- per long ton bone dry.

Our consumption of clay for papermaking is of the order of 14,000 tons per annum and is rising, and some 10/11,000 tons of this we have to import from overseas.

This is a continuing demand which would afford steady outlet for your production and when you have supplies of clay of the required specification available at the landed price of similar overseas clay, we would be happy to place a considerable proportion of our business with you.

We assure you of our wholehearted support because we fully appreciate the advantages of having supplies of suitable clay available locally, and to assist you during the stage of initial production, should your costs of production exceed the landed cost of comparable English clay, on proof of this we are prepared to assist your establishment by paying up to £2/10/- per ton additional for up to 5,000 tons for suitable clay supplied to us during the first year's operation of your plant. Beyond that time we would expect you to be competitive with imports.

Yours faithfully,
ASSOCIATED PULP AND PAPER MILLS LIMITED.

GENERAL SUPERINTENDENT.

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284023
SCHEDULE B

R. HARE AND ASSOCIATES

CONFIDENTIAL REPORT

to

STOREYS CREEK TIN MINING CO. N. L.

ESTIMATE OF RESERVES OF KAOLINITIC CLAY IN THE SOUTH MOUNT CAMERON AREA, TASMANIA

by

R. HARE, B.Sc., M.Aus.I.M.M.

4th July, 1961.

SUMMARY

1. The kaolinised and leached granites of the South Mount Cameron Area of Tasmania are suitable for mining and treatment to produce a high grade filler clay used in the paper manufacturing industry.

2. The Scotts and Browns Areas have been tested by boring, and proved reserve of recoverable finished clay meeting the required specification in the two deposits is estimated as 190,000 long tons.

3. Taking into account the untested YZ area and the widespread occurrence of kaolinised granite, there is no doubt that further reserves of suitable clay will be found by boring.

no record of lease 9/11/62

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Introduction

The South Mount Cameron kaolin clay deposits were inspected by the writer, in company with Mr K. Turner, on 23rd June, 1961. Three deposits were looked at, Browns Area, Scotts Area and the YZ Area. All these deposits lie within a 144 square mile Clay Exploration Area No. El 2/60, held by Storeys Creek Tin Mining Co. N.L. However, some of the area examined is covered by leases. The ownership of these leases has not been checked. Now EL 4/63

The clay area is approximately 50 miles north east of Launceston, 79 miles by the Tasman Highway. The deposits are accessible and easily mined.

The clay deposits have been disclosed by tin sluicing operations. The deposits are areas of highly weathered and leached granite. In this weathering process the felspars have been completely kaolinised. Leaching has almost completely removed iron from the surface zone. The result is a soft, white kaolinitic clay loaded with quartz grains of the original granite.

Extraction of the clay can be accomplished by making a thick pulp of the material, screening to remove coarse quartz and tramp roots, bark, etc., and running the fine fraction through a cyclone. The overflow from the cyclone is bleached if required, then filtered, dried and bagged as a white cake. Experimental work carried out in the Department of Mines Laboratory in Launceston leaves little doubt that the clay can be recovered by this general method in a full-sized plant. The economics of the clay recovery has not been studied, apart from the fact that it is reasonable to assume that ½ ton of finished clay can be recovered from a cubic yard of clay in place.

It is proposed to sell the finished product to Associated Pulp and Paper Mills Ltd., at Burnie for use as a filler clay in paper manufacture. The general specification for the clay provided by the paper company is as follows :

Brightness	(A.S.T.M. Directional Reflectance Method designation E 97-55)
	77
Grit (Quartz)	Chemical method - less than 3%
Sizing	Plus 200 mesh - nil
	Plus 30 microns - less than 2%
	Quantity of ore micron clay generally ranges from 25 to 40%
Ignition loss	Approximately 12%

Determinations on clay from the area, carried out by the Department of Mines, show that clay from Scotts, Browns and the YZ area can be treated to produce a product meeting the above specifications.

Clay Reserves

A. Scott Area.

The deposit in this area has been bored on an irregular 100 foot grid. The holes are to an average depth of around 25 feet. The proved reserve is estimated as follows:

Area of clay (brightness 77)		33,000	square yards
Average depth of clay		7	yards
		231,000	solid yards
Allow 10% for patches of rejected clay,		23,100	solid yards
Reserve	say	200,000	solid yards

Reserve of finished clay at ½ ton/yard 100,000 tons

B. Browns Area.

This area contains clay in three patches, all of which have been bored on an irregular pattern roughly equivalent to a 100 foot grid. The proved reserve is estimated as follows:

Area of clay (brightness 77)		33,000	square yards
Average depth of clay		6	yards
		198,000	solid yards
Allow 10% for patches of rejected clay,	say	20,000	solid yards
Reserve	say	180,000	solid yards

Reserve of finished clay at ½ ton/yards 90,000 tons

C. YZ Area.

No estimate has been made of the reserve in this area due to lack of systematic boring. It is likely that reserves of clay in this deposit meeting the specification will greatly exceed the total reserves in Browns and Scotts areas.

On some of the area of clay, a few feet of overburden will need to be removed. This will present no difficulty.

Potential for further reserves.

There is no doubt that further exploration by boring will greatly increase the reserves of clay in this South Mount Cameron area.

R. Hare
4th July, 1961.

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SCHEDULE "C"

284028

DEPARTMENT OF MINES LABORATORY.

Launceston. 20th April, 1961.

ORE DRESSING INVESTIGATION.

R.361, R.364, R.365, R.366 & R.368.

Clay - South Mount Cameron.

(1). This investigation deals with the treatment of several samples of weathered granite from South Mount Cameron district in North-East Tasmania from leases held by D. Brown and partners for the production of clay suitable for use in the manufacture of paper.

The work consisted of removal of impurities such as coarse quartz and fine grit from the clay by agitation after dispersion with sodium silicate, classification, screening and removal of finest sand by treatment in small diameter hydro-cyclones.

Where necessary, bleaching with sodium hypochlorite or chlorine has been tested to increase brightness of the clay. Degritted clay pulps have been flocculated with aluminium sulphate, and then submitted to filtration to remove surplus water in preparation of final drying of the refined clay. Samples of degritted and dispersed clay have been centrifuged to assess the effect on economics for subsequent flocculation and drying, and also to determine the nature of the clays that can be produced by this means. From results of evaluation of bores and yields of refined clay in experimental work, the yield of clay would range from 30 to 40 percent, and a yield of 33⅓ percent for assessment of economics appears to be reasonable.

(2). Associated Pulp & Paper Mills Ltd., Burnie, are users of considerable quantities of filler clay in paper manufacture, and a specification has been provided by them as a guide to cover clay suitable for their specific uses. This specification is as follows

Brightness.

(A.S.T.M. Directional Reflectance Method designation E97-55) originally 80 and later reduced to 77.

Grit (Quartz).

Chemical Method, less than 3 percent.

Sizing.

Plus 200 mesh-Nil. Plus 30 microns less than 2 percent.

The quantity of one micron clay generally ranges from 25 to 40 percent, and the clay as naturally produced experimentally has been stated to be satisfactory for one micron clay content.

Ignition Loss.

Not specific, but of the order of 12 percent.

Redispersibility.

The dried product to be redispersible. Samples of dried clays were submitted to A.P.P.M., Ltd., and were reported by them to be redispersible.

DEPARTMENT OF MINES LABORATORY

The method used for determination of grit by chemical means is attached to the report.

(3). The treatments given to the bulk samples of clay have resulted in the production of paper clay of a quality which complies with the specification referred to above. Some samples are not bleachable to the desired brightness with hypochlorite which indicated that selective mining is necessary to maintain the high standard of brightness required for the industry. Iron stained clay can be bleached by reduction with zinc hyposulphite, but the previous test work by A.P.P.M. Ltd. reports regression of brightness and the process is not favoured.

(4). The following flowsheet shows the treatment process which has resulted in production of high quality filler clays. Some attention has been given to additional recovery of clay from the primary cyclone spigot, and a possible application is shown in the flowsheet. The bleach process is shown, but this would not be consistently necessary to give the desired brightness. A centrifuge is also included to indicate specifically the position for application.

Crude Clay (Sodium Silicate)

Agitators (Water)

Sands to Waste ← Rake Classifier

Overflow
(Clay with some fine sand)

Oversize to Waste ← Vibrating Screen

Undersize

Cyclone

Spigot Product | Overflow

(Cyclone or Classifier) — Overflow

Centrifuge

Bleach & Flocculation Tank (Chlorine) (Aluminium Sulphate)

Sands to Waste

Filter

Dryer

(5). The results of each treatment process are briefly summarised below.

a. Agitation & Dispersion.

High densities for economic treatment and to provide a scrubbing action to clean the clay from quartz was satisfactory with dispersed pulps containing 50 to 60 percent of solids. Sodium silicate at the rate of 10 lbs. per ton of crude material resulted in acceptable dispersion. No tests were made on "as mined" material to determine the requisite agitation period. However, air-dried material readily broken to half inch size was effectively slacked in less than five minutes.

b. Classification & Screening.

Removal of coarse sand to about 60 mesh was effective in a 6" Akins classifier with a crude clay overflow of over 30 percent solids. The discarded coarse sand contained about 2 percent clay, and this represents an overall loss of about 3 percent of the clay. An electric vibrating screen was effective for removal of root, bark and tramp oversize etc. and plus 72 mesh sand.

c. Cyclone.

Tests have been conducted with 3" and 30 m.m. cyclones at various high pulp densities to remove the finest quartz etc. to 30 micron size, and to produce a clay with a maximum content of 2 percent plus 30 micron material.

The clay is naturally flocculated, and in this condition the pulp is appreciably more viscous than in the dispersed condition, consequently at high densities cycloning results in a more effective separation in the dispersed condition.

The desired sizing separation is readily obtainable in a 30 m.m. cyclone with feed densities of the order of 32 percent solids at 40 p.s.i. A 3" cyclone at pressures up to 75 p.s.i. produced too coarse an overflow at the above high density, and research indicated that the solids content of the feed would have to be reduced to about 20 percent before a primary overflow of 98 percent minus 30 microns was produced. Further work may be desirable to determine the most economic cyclone set up for this separation.

d. Centrifuge.

Application of separation by this means has two features of usefulness.

(1). Separation of the coarser clay as a pulp of about 50 percent solids, and an effluent of low solid content containing the finest clay. Subsequent flocculation and filtration of the effluent only can result in an economy as compared with the addition of flocculant to the total clay pulp. Details of tests are given which show that any preference for use of the centrifuge is dependent upon the relative quantities of clay separated, and the overall effect of filtration capacity and cost.

(2). Use of the effluent as a separate product for marketing as a coating clay.

e. Bleach.

The use of chlorine gas is effective for increase of brightness of clays stained with organic matter, and is appreciably more economic than the solution of sodium hypochlorite. Excess chlorine may be a disability on completion of the bleach process, and although this can be destroyed with sodium bisulphite, tests have shown some reduction of brightness caused by the use of this reagent. It has been suggested that application of bleach as a batch process could have the advantage of mixing bleached clay pulps with clay pulps which do not require bleaching as a means of the destruction of residual chlorine. Investigations have not included this method of operation. Application of bleach to the total clay would be made after degritting the dispersed pulp.

f. Filtration.

Numerous leaf tests were undertaken to test variables for efficient and economic filtration. Generally filter capacity is related to the volume of pulp filtered, and thus it is of special significance that all treatment processes should be performed at highest practical densities to ensure low cost treatment, particularly for filtration. The results of leaf tests were used for selected tests with a 1' x 3' FEinc rotary vacuum filter fitted for string discharge. This unit was made available on loan from the Chemical Plant & Engineering Co. Ltd., Ashly Street, Footscray.

Flocculation was satisfactory with aluminium sulphate in amounts ranging from 15 to 45 lbs. per ton of refined clay. Filtration rates were recorded under varying conditions, and these will vary with the nature of the clay, percent solids and temperature of the pulp, and other conditions effecting flocculation.

Typical results were as follows:

Temperature°C.	Cake Thickness.	Filtration Rate Lbs. of Clay/Sq.Ft./Hour.
11	9-12/64 inch	3-5
58	9-13/64 "	6-12

Discharge of filter cakes of the above thickness takes place by flexure as the cake passes over the roller. Increase in speed produces thinner cakes, and when sufficiently thin, discharge takes place mainly on the alignment comb and filtration rates are doubled by this means. This method of cake removal could be of economic interest, and consideration could be given to this method of operation.

g. Drying.

Drying of the clay has not been included in the investigation with the exdeption of the effects of heating to various temperatures. This work shows the necessity for temperature control in the drying process. Overheating results in reduction of brightness.

Chemical requirements for treatment will vary considerably with variations in the nature of the clay. To indicate the general cost of chemicals used in the test work typical quantities and costs of chemicals are shown.

Reagent	Rate Lbs./Ton of Refined Clay	Cost Ton at Sth.Mt.Cameron	Cost per Ton of Clay
Sodium Silicate	30	£30	£0.4
Aluminium Sulphate	30	£40	£0.6
Tetra Sodium Pyrophosphate	5	£140	£0.32

The above chemicals were used for dispersion and flocculation of the clay, and redispersion before drying.

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Other chemicals and supplies of interest have been costed and are listed below.

Chlorine	£122 per ton at Sth. Mt. Cameron.
Sodium Bisulphite	£88 " " .
" Hypochlorite	7/6 per gallon at " , containing 12 percent chlorine.
Calcium Chloride	£42.6 per ton at Sth. Mt. Cameron.
Sodium Thiosulphate	£75 " "
Oil Fuel, Light & heavy.	£26.5 & £18 at " .
Coal	£6 per ton at " .
Power	Estimated by Mr. H.K. Turner 1.34 pence K.W.H. from costs at Dorset Dredge.

Chemicals for bleach using 6.7 lbs. of chlorine per ton of clay, and sodium bisulphite to destroy excess chlorine show a cost of 10/- per ton of clay.

Our thanks are recorded for assistance given to us in these investigations by the Chief Chemist, Mr. McKercher and Staff of A.P.P.M. Ltd., Burnie, and Mr. G.J. Robertson, Chief Chemist of the Ballarat Clay Co.

The Crude Clay Samples.

A total of 6 crude clay samples were obtained from the Sth. Mt. Cameron deposit. These were as follows:-

Sample No.	Weight of Sample	Location of Sample
R. 361.	2 Tons	Near Bore Hole 3, North block.
R. 364.	1 Cwt.	Bore Hole No. 7, to 16 feet.
R. 365 A.	1 "	" " 8.
R. 365 B.	1 "	" " 8.
R. 366.	1 "	" " 16, to 22 feet.
R. 368.	1 Ton	Near bore hole 1 and 2, North block.

Samples R. 361 and R. 368 were obtained as bulk samples of average quality crude clay for pilot plant tests, involving beneficiation by degritting and sizing in classifiers and cyclones, and dewatering by filtration.

Samples R. 364, R.365 and R. 366 were obtained specifically as sources of organically stained crude clay for bleaching tests.

A sizing of R. 361 is more or less typical of the crude clays tested.

Size Fraction B.S. Screen	Percent Weight
+ 44 mesh	39
- 44 + 60 mesh	5
- 60 + 100 mesh	3
- 100 + 200 mesh	3
- 200 + 350 mesh	3
- 350 + 30 microns	5
- 30 + 20 microns	5
- 20 microns	37
	100

In all of the crude clays handled only a very small amount of the quartz is coarser than $\frac{3}{8}$ inch size.

The clay contents of each of the 6 bulk samples in terms of minus 30 micron material were-

R.361	42 percent
R.364	41 "
R.365 A	44 "
R.365 B	34 "
R.366	48 "
R.368	50 "

Bouyoucos hydrometer sizings of the clays prepared by the "minus 30 micron separation" method (see appendix for details) were-

Size Fraction	CLAY R.361	R.364	R.365 A	R.365 B	R.366	R.368
+ 10 micron	14	14	17	18	7	12
- 10 + 5 micron	17	21	16	17	12	23
- 5 + 2 micron	24	24	29	28	25	31
- 2 + 1 micron	23	21	13	14	16	12
- 1 micron	22	20	25	23	40	22

Clays obtained by the minus 20 micron "Evaluation of Weathered Granite" method (see appendix for details) had brightnesses as follows:-

	CLAY					
	R.361	R.364	R.365 A	R.365 B	R.366	R.368
Unbleached brightness	76	71	74	60	69½	79
Maximum brightness attained with excess chlorine	77	83½	81	76½	82½	84

Dispersion of the Crude Clay.

Sodium silicate was used throughout the laboratory and pilot plant work to disperse the crude clay prior to de-gritting. Sodium silicate gave satisfactory dispersion, and the dispersed clay was later readily flocculated with aluminium sulphate and other flocculants.

Other dispersants were not tested.

Quantity of Sodium Silicate required for dispersion.

There is no absolute laboratory method of measuring the degree of dispersion of a clay pulp. It is thus difficult to determine the quantity of sodium silicate required to disperse crude clay.

Laboratory bench scale tests showed that clays prepared by either of the standard decantation methods (see appendix for details) produced clays meeting the tentative specifications of A.P.P.M. Ltd. when the crude clay was dispersed with 9-10 pounds per ton of sodium silicate. These tentative specifications are discussed later.

Approximately 10 lbs. of sodium silicate per ton of crude clay was used throughout pilot plant test work. This usage agrees closely with 11 pounds per ton of crude clay used by Ballarat Clay Co. Pty. Ltd., Ballarat.

Grades of Sodium Silicate.

Several grades of sodium silicate are available from manufacturers. We have no data relating to the relative economic effectiveness of these different grades in dispersing crude clay. Rubanit Roofing and Paper Products Pty. Ltd. have suggested either N84 or A140 grades for clay dispersion.

Silicate & Dolomite Sales (N.S.W) Pty. Ltd. have suggested N84 grade.

Ballarat Clay Co. Pty. Ltd., and A.P.P.M. Ltd. both use N84 for dispersing clay.

Several grades of sodium silicate were used during test work and were all effective in dispersing the crude clay.

Cost of Sodium Silicate for Dispersion of Crude Clay.

Rubanit Roofing & Paper Products Pty. Ltd. have quoted N84 Sodium Silicate at £18: 5: - per ton F.O.B. Melbourne, in minimum lots of 10 x 44 gallons. Price includes non-returnable drums.

Cost at Sth. Mt. Cameron has been estimated at £30 per ton.

At 10 lbs. per ton of crude clay, cost of sodium silicate will be £0.13 per ton of crude clay.

Assuming an average yield of 33 percent of paper clay from the crude clay the cost of sodium silicate will be £0.39 per ton of paper clay.

Summary: Dispersion.

1. Crude clay can be satisfactorily dispersed with approximately 10 lb./ton of sodium silicate.

2. Several grades of sodium silicate are available, but N84 grade seems to be favoured for clay dispersion generally.

3. Cost of sodium silicate for dispersion is estimated at £0.39 per ton of paper clay.

Agitation for Dispersion of Crude Clay.

Dispersion of the crude clay with sodium silicate at high pulp densities requires some agitation. It is difficult to translate laboratory and pilot plant scale agitation on air-dried clays to commercial practice using large lumps of wet clay.

Test work involved comparatively dry clay with maximum sizes of about $\frac{1}{4}$ inch for laboratory work, and air-dried clays up to about 1 inch size for pilot plant work. Under these conditions, experience has shown that agitation of a few minutes only is required. We have no data relating to agitation required to disperse large lumps of wet clay, say 1-2 feet diameter.

The pilot plant agitators used were No. 1 Denver conditioner-super agitators. Dispersed mobile pulps up to 70 percent solids could be obtained, but it was impractible to operate at this high pulp density. Good operation was experienced at pulp densities of 50-60 percent solids, and pilot plant dispersion was carried out in this range.

Agitation times of approximately 5 minutes were adequate to wash the quartz clean and to disperse the clay. It is possible that, in practice, no particularly prolonged agitation of the crude clay pulp will be required, and that normal plant handling will be sufficient.

There is some evidence that excessive agitation degrades the clay subsequently removed, possibly by breaking up the quartz. In one test (detailed later) the following results were obtained from duplicate samples.

Time of agitation	Yield on decantation	Brightness of Clay
1 minute	41.5	73.0
1 "	41.5	72.8
30 minutes	44.9	71.8
30 "	44.2	72.2

This table indicates that the increased agitation increased the yield by about 3 percent, and degraded the brightness of the clay by almost one unit.

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Summary:. Agitation.

1. The air-dried clays were readily dispersed with little agitation in pilot plant work.

2. Excessive agitation appears to increase the yield of clay, but the clay is degraded in quality.

Removal of Quartz Gravel & Sand from Dispersed Crude Clay.

Pilot Plant Operation.

The minus 1 inch air-dried crude clay was fed to a No. 1 Denver conditioner-super agitator by means of a vibrating feeder. Regulated quantities of make-up water and sodium silicate were added continuously.

Addition of sodium silicate. 10 pounds per ton of crude clay.

Retention time in agitator. 5 minutes(average).

Pulp density in agitator. 50-60 percent solids.

The dispersed pulp was fed to a 6 inch Akins spiral classifier to remove gravel and coarse sand. Entrained clay was washed out of the classifier sands by spray water.

Classifier overflow was pumped to a Hummer electric vibrating screen, fitted with a 72 mesh stainless steel cloth. The screen scalped out the small quantity of wood and fibrous vegetable material present in the crude clay, plus a small quantity of tramp oversize in the classifier overflow. Screen oversize returned to the classifier. The screen undersize gravitated to a second Denver agitator for storage prior to cycloning.

Notes on Pilot Plant Operation.

1. Densities of Pulps.

For later filtration the density of the pulp should be maintained as high as possible. Cyclone tests(detailed later) indicate that clay of acceptable quality can be obtained by treating pulps in a 30 m.m. cyclone at 34-35 percent solids to give cyclone overflow pulp densities of about 30 percent solids. With more dense pulps the clay in cyclone overflow fails to meet the specifications for grit and sizing This factor then limits the density of the screen undersize to 34-35 percent solids.

The scalping screen removes very little solids, and the density of the feed to the screen, i.e. classifier overflow is thus limited to about 35-36 percent solids.

The primary agitation and dispersion can be carried out at densities up to 60 percent solids. Removal of the classifier sands from a feed of this density leaves a pulp of up to about 40 percent solids. The density of this pulp as classifier overflow is reduced to the required 35-36 percent solids by the application of spray water to the classifier sands to give maximum recovery of entrained clay. A small quantity of spray water is also applied to the scalping screen.

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2. Loss of Entrained Clay in Classifier Sands.

Dispersion at high densities allows maximum use of spray water to recover clay from the classifier sands. Tests showed less than 2 percent of minus 30 micron material in the classifier sands. This loss of clay in the classifier sands amounts to about 2 or 3 percent of the recoverable clay in the original crude clay.

3. Use of Scalping Screen.

The crude clay contains a small qualtity of wood and fibrous vegetable matter. This, together with a small amount of tramp oversize,was scalped out by a 72 mesh screen. Without this scalping screen ahead of the cyclone, considerable difficulty was encountered with blockages of the very small underflow orifice of the 30 m.m. cyclone.

As the clay in the pulp is dispersed when fed to the scalping screen, the fine clay and water readily pass through the screen. The quantity of tramp oversize is small (apart from the particles of vegetable matter), but would quickly block the underflow orifice if not removed prior to cycloning.

4. Storage in Second Denver Agitator..

Production rate of the cyclone feed pulp was less than the capacity of the cyclone-pump combination. Use was made of this agitator to store the pulp and to ensure a uniform feed to the cyclone.

Summary: Pilot Plant Operation.

1. A dispersed clay pulp (suitable as feed to a 30 m.m. cyclone for final degritting) was prepared in the pilot plant involving

a. agitation and dispersion with sodium silicate in a No. 1 Denver conditioner-super agitator,

b. removal of coarse gravel and sands in an Akins 6 inch spiral classifier,

c. removal of vegetable trash and some tramp oversize by a 72 mesh Hummer screen.

2. Initial agitation and dispersion of the crude clay should be at high densities, to allow maximum use of wash water to remove entrained clay in the classifier sands.

3. Loss of clay in the classifier sands amounts to about 2 to 3 percent of the recoverable clay in the crude feed.

4. A scalping screen ahead of the 30 m.m. cyclone is essential to minimize blockages of the cyclone orifice.

Production of Paper Clay in a 30 m.m. Cyclone.

The 30 m.m. Cyclone&Pump.

Production of paper clay was obtained by treating the classifier overflow in a 30 m.m. hydrocyclone, supplied by Liquid-Solids Separations, London.

The cyclone is rubber lined, and has a fixed vortex finder $\frac{3}{8}$ inch outside diameter and $\frac{1}{4}$ inch inside diameter. The diameter of the orifice can be varied by removing or adding rubber spacers containing graded orifices.

Consideration of the problem of sizing the clay in a 30 m.m. cyclone indicated an operating pressure of not less than 40 p.s.i., and as the desired results were obtained at this pressure from pulps of up to 35 percent solids, tests were not conducted at lower pressures

The cyclone was fed by a $\frac{3}{4}$"/1" Kelly & Lewis water pump. The pump-cyclone unit handled approximately 3 gallons per minute of feed at a pressure of 40 p.s.i.

The cyclone was mounted over the conical pump sump. Flexible lines taking both cyclone overflow and underflow allowed the products to be returned to the pump sump, or to be removed as desired.

Specifications Relating to Grit & Sizing.

A tentative specification, drawn up by A.P.P.M. for our guidance, includes the following specifications:

> "Material coarser than 200 mesh B.S.S. nil, coarser than 30 microns not more than 2 percent."
>
> "Grit by the sulphuric acid digestion method, reference Analytical Chemistry, Treadwell and Hall, ninth English edition, volume 2, page 422, not to exceed 3 percent."

Cyclone Tests.

The object of treatment in the 30 m.m. cyclone was to produce a clay containing not more than 2 percent of plus 30 micron material. A typical sizing of cyclone feed is shown below.

Fraction	Percent Weight
- 60 + 100 mesh	5.0
-100 + 200 "	5.2
-200 + 350 "	5.6
-350 mesh + 30 micron	8.9
- 30 micron + 20 micron	9.0
- 20 micron	66.3

The clay was produced by cycloning the dispersed pulp. The finest sized fractions in the samples tested are clay, and are free of quartz and mica. The quartz from the weathered granite is comparatively coarse, and the bulk of this quartz is removed by classification. Almost all of the plus 30 micron material in the feed to the cyclone is removed in the cyclone underflow.

With the clays tested, R.361 and R.368, and operation of the cyclone to give a product to meet the specification of not more than 2 percent coarser than 30 microns, the clays readily meet the specification relating to grit.

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Other clays in the deposit could contain more very fine quartz, and separated clays from such may not meet the grit specification, although passing the sizing specification.

Evaluation of the cyclone overflows with reference to the 30 micron sizing was by the A.P.P.M. method "Evaluation of paper clays for 30 micron separation" (see appendix).

More detailed sizings of some typical products were made with a Bouyoucos type soil hydrometer, No. 152 H, following the A.S.T.M. tentative standard-designation D 422-54 T for grain size analysis of soils.

30 m.m. Cyclone Operation.

Effect of Pulp Density.

For best economic filtration, the density of the pulp should be maintained as high as possible. Little difficulty was experienced in producing clays to meet the 30 micron sizing specification by cycloning pulps at low densities. Test work was directed towards obtaining a high yield of acceptable quality clay in an overflow of relatively high pulp density.

The following series of tests show the inter-relation between grit and sizing of the cyclone overflow with pulp density on clay R.361.

	Cyclone Feed	Primary Cyclone Overflow				Cyclone U/flow.
Test No.	Pulp Density % Solids	Yield % from Crude Clay	Pulp Density % Solids	Grit %	Plus 30 Microns %	Pulp Density % Solids
7.	32	28.0	27	1.7	1.3	46
6.	34	29.0	30	2.0	1.9	45
5.	37	29.0	34	2.4	3.9	46
4.	41	29.0	37	2.9	4.9	46

This series of tests indicate

1. Yield of clay in the cyclone overflow is reasonably constant, and seems independent of pulp density of the feed.

2. The grit content of the overflow increases with increase in pulp density, but the highest pulp density tested (41 percent solids in feed: 37 percent solids in overflow) the grit content of the overflow on the samples tested still meets the tentative specification.

3. The plus 30 micron content of the overflow increases with increase in pulp density. Overflows with pulp densities of 31 and 34 percent solids met the tentative specification. Overflows with pulp densities of 37 and 41 percent solids did not meet the specification.

4. Pulp density of the underflow is reasonably constant, and is independent of pulp density of the feed in the range tested.

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Clays R.361 and R.368 were produced by cycloning for other test work, such as filtration and centrifuging. These clays were generally evaluated for grit and for plus 30 micron content. Yield in terms of crude clay was determined in some cases.

Typical 30 m.m. Cyclone Test Results.

Test No.	Clay	Cyclone Feed Pulp Density % Solids	Primary Cyclone Overflow Yield from Crude Clay%	Primary Cyclone Overflow Pulp Density % Solids	Primary Cyclone Overflow Grit %	Primary Cyclone Overflow Plus 30 Microns %	Cyclone U/flow Pulp Density %Solids	Cyclone U/flow % -30 Micron Material
1.	R.361	26	32	23	2.0	1.0	44	69
2.	R.361	25	34	24	2.1	1.4	47	69
10.	R.361	33	-	29	2.2	1.0	44	-
13.	R.368	34	39	30	1.0	1.4	45	52
14.	R.368	35	37	31	1.2	1.5	46	53

These tests indicate that clay produced by cycloning samples R.361 and R.368 at feed densities of up to 33-35 percent solids will give clay overflows of up to 30 percent solids which meet the tentative specifications relating to plus 30 micron sizing and grit.

Further recovery of clay is possible by retreatment of the primary cyclone underflow by further cycloning, or by classification, as shown in the following example.

Thus in test R.361/1 results of primary cycloning was:-

Product	Weight % of Crude Clay	Pulp Density % Solids	Grit %	Plus 30 Microns %
Primary O/flow	32	23	2.0	1.0
" U/flow	12	44	-	31.0
Feed to Cyclone	44	26	-	-

The primary cyclone underflow was then diluted with water and again cycloned.

Product	Weight % of Crude Clay	Pulp Density % Solids	Grit %	Plus 30 Microns %
Secondary O/flow	4	5	-	1.0
" U/flow	8	46	-	44.0
Primary U/flow	12	-	-	31.0

This method allows the clay yield to be increased, but the clay pulp is more dilute. Normally such dilute pulp should not be added to the primary cyclone overflow, but could readily be circulated back to the original dispersing agitator without decreasing the plant pulp density, or alternatively join the feed to the primary cyclone.

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Retreatment of cyclone underflow for recovery of additional clay has also been tested by classification by decantation. The results obtained by both methods are shown below, and indicates a high quality product by classification.

	Feed % Solids	Weight % of Crude Clay	Overflow % + 30 Microns	Overflow % Grit
Classification	20	2.6	6.7	1.6
Recyclone	15	2.5	2.8	3.5

A series of tests was carried out on clay R.361 in the naturally flocculated state. The object of these tests was to examine indicated costs of degritting and subsequent flocculation with clay dispersed with sodium silicate.

The cyclone feed pulp was obtained by agitating the crude clay with water only, and screening on a 72 mesh screen.

Test No.	Cyclone Feed Pulp Density % Solids	Cyclone Overflow: Yield from Crude Clay %	Cyclone Overflow: Pulp Density % Solids	Cyclone Overflow: Plus 30 Microns %	Cyclone Overflow: Grit %
11 a.	31	27	27	5.7	3.2
11 b.	23	24	20	4.2	2.3
11 c.	18	21	12	1.0	1.5
11 d.	12	21	10	1.0	1.3

The results show that acceptable quality clay can be obtained only at comparatively low pulp densities, and that overall clay yield is much less than that obtained from dispersed pulps.

As a result of the unfavourable comparison flocculation and filtration tests were not proceeded with.

Tentative A.P.P.M. Specification: 200 Mesh.

The specification requires "material coarser than 200 mesh nil". Unless a 200 mesh screen is incorporated in the circuit, it is unlikely that this specification will be met. The proportion of plus 200 mesh material in various products is listed below

Product	Sizing: % + 200 Mesh
Cyclone Overflow R.361, Test 4	0.23
" " " , " 10	0.20
" " " , FEinc Filter Feed	0.12
Test 13, R.368	Trace
Ballarat Clay, ex A.P.P.M.	0.10
English Clay, ex A.P.P.M.	0.05

Sizing of Cyclone Overflows.

Cyclone overflows from clay R.361 were sized by the Bouyoucos hydrometer method with the following results.

Size Product	% Weight Sample "A"	% Weight Sample "B"
+ 30 Microns	3.5	2.0
- 30 + 20 Microns	2.0	1.5
- 20 + 10 "	8.0	8.5
- 10 + 5 "	19.5	18.0
- 5 + 1 Micron	42.0	44.0
- 1 Micron	25.0	26.0

Sample "A" was produced in a pulp of 33 percent solids.

" "B" " " 21 " " .

A sample of R.361 cyclone overflow was sized by sedimentation, and the grit content of each fraction was determined.

Size Fraction	% Weight	% Grit	% Distribution of Grit
- 30 + 18 Microns	6.2	12.0	37
- 18 + 8 "	12.8	4.5	28
- 8 Microns	81.0	0.9	35
Composite Clay	100.0	2.0	100

The sizing shows clearly the predominance of grit in the coarser sizes.

Preparation of Paper Clay in a 3 inch cyclone.

A series of tests was carried out on clay R.361 to determine the possibility of using a 3 inch cyclone in place of the 30 m.m. cyclone previously used in paper clay production. The 30 m.m. cyclone has a very small orifice, and unless considerable care is taken in preparation of the cyclone feed, this orifice is easily blocked. A 3 inch cyclone has a much larger orifice, and hence is much more attractive from the aspect of orifice blockage.

The cyclone used was a 3 inch rubber lined unit made by Warman Equipment (W.A.) Pty. Ltd. The cyclone was fed by a Warman split case pump, driven by a variable speed motor. Portion of the pump discharge could be by passed back to the pump sump.

The pulp was prepared by the normal pilot plant procedure.

Variables tested in the series included cyclone inlet pressure, density of feed pulp, and underflow orifice diameter.

In general, at similar pulp densities, the 3 inch cyclone gave more plus 30 micron material in the overflow than the 30 m.m. cyclone under similar conditions. Progressive dilution of the feed decreased the quantity of plus 30 micron material in the overflow. An overflow product meeting A.P.P.M.'s tentative specification of not more than 2 percent plus 30 microns was obtained with a feed of 21 percent solids and an inlet pressure of 75 p.s.i. giving a cyclone overflow density of 19 percent solids. Other variables tested had little effect upon results. Volume of cyclone feed ranged from 9 to 12 g.p.m.

Test Results. 3" Cyclone.

		Cyclone Feed	Cyclone Overflow		
Test	Pressure p.s.i.	Pulp Density % Solids	Pulp Density % Solids	% + 30 Microns	% Grit
A.	40	33	32	6.3	-
D.	60	33	30	5.1	-
F.	70	30	27	4.2	-
G.	75	28	24	2.9	-
H.	75	21	19	1.9	2.0

The trend towards better quality clay with dilution of feed pulp is clearly indicated in the above table of test results.

Bleaching with Sodium Hypochlorite.

Clays from the Gladstone area are usually coloured to some extent with organic matter. It is possible to improve the colour (brightness) of many of these clays by bleaching with chlorine. Initial bleaching tests were carried out using sodium hypochlorite, supplied from Launceston manufactured stock by Imperial Chemical Industries of Australia and New Zealand Limited. The sodium hypochlorite solution, as supplied, contains 12.5-12.8 grams of available chlorine per 100 ml. of solution.

Some obvious factors influencing bleaching are

1. quantity of available chlorine used,

2. time of contacts with the chlorine,

3. temperature,

4. removal of bleach liquors and washing the clay free from liquors.

Work was not carried out to determine the effect of temperature on bleaching. It was considered that bleaching would probably be carried out in unheated pulps.

Work was not carried out to determine the effect of washing the clay free of bleach liquor after bleaching. It would be unpracticable to do this and still retain the comparatively high pulp densities desired during filtration.

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Initial bleach work was attempted on the bulk sample R.361. The separated clay from R.361 had a brightness of about 76, and it proved impossible to increase the brightness of this clay by more than one unit. Four additional bulk samples of clay were then obtained from areas known to give clays of low brightness to determine the effect of bleaching-these were samples R.364, R.365A, R.365B and R.366. Later bulk sample R.368 was also used in bleach test work.

Bleaching Tests.

Various clays were prepared by decantation after normal dispersion with sodium silicate. The prepared clay pulp was then divided into a number of similar samples in sealed plastic bottles. Varying quantities of sodium hypochlorite were added to the plastic bottles which were agitated at Irregular intervals. Portion of the pulp in each bottle was removed at predetermined intervals, flocculated with aluminium sulphate, filtered, dried, and the brightness of the clay determined.

Clay R.364.

Quantity of Chlorine added: % of Clay	Brightness. After 5 hours contact	After 24 hours contact	After 48 hours contact
Nil	70.6	71.8	71.0
0.09	75.0	75.8	75.0
0.19	77.4	78.0	78.6
0.94	81.6	81.8	82.0
1.89	82.4	82.6	83.0
2.83	83.2	83.0	83.4
3.77	83.0	83.2	83.0
4.72	83.6	83.0	83.0

ClayR365 B.

Quantity of Chlorine added: % of Clay	Brightness. After 5 hours contact	After 8 hours contact	After 72 hours contact
Nil	60.0	60.4	60.4
0.17	62.6	63.2	63.6
0.35	67.4	67.2	67.4
0.87	71.8	72.0	73.8
1.74	75.0	74.4	75.4
3.47	76.4	75.8	76.4

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Sheet No. 18.

Clay R.365 A.

Quantity of chlorine added: % of Clay	Brightness After 28 hours contact
Nil	74.2
0.11	77.0
0.22	78.6
0.55	80.2
1.10	81.0
1.64	81.0
2.19	81.2

Clay R.366.

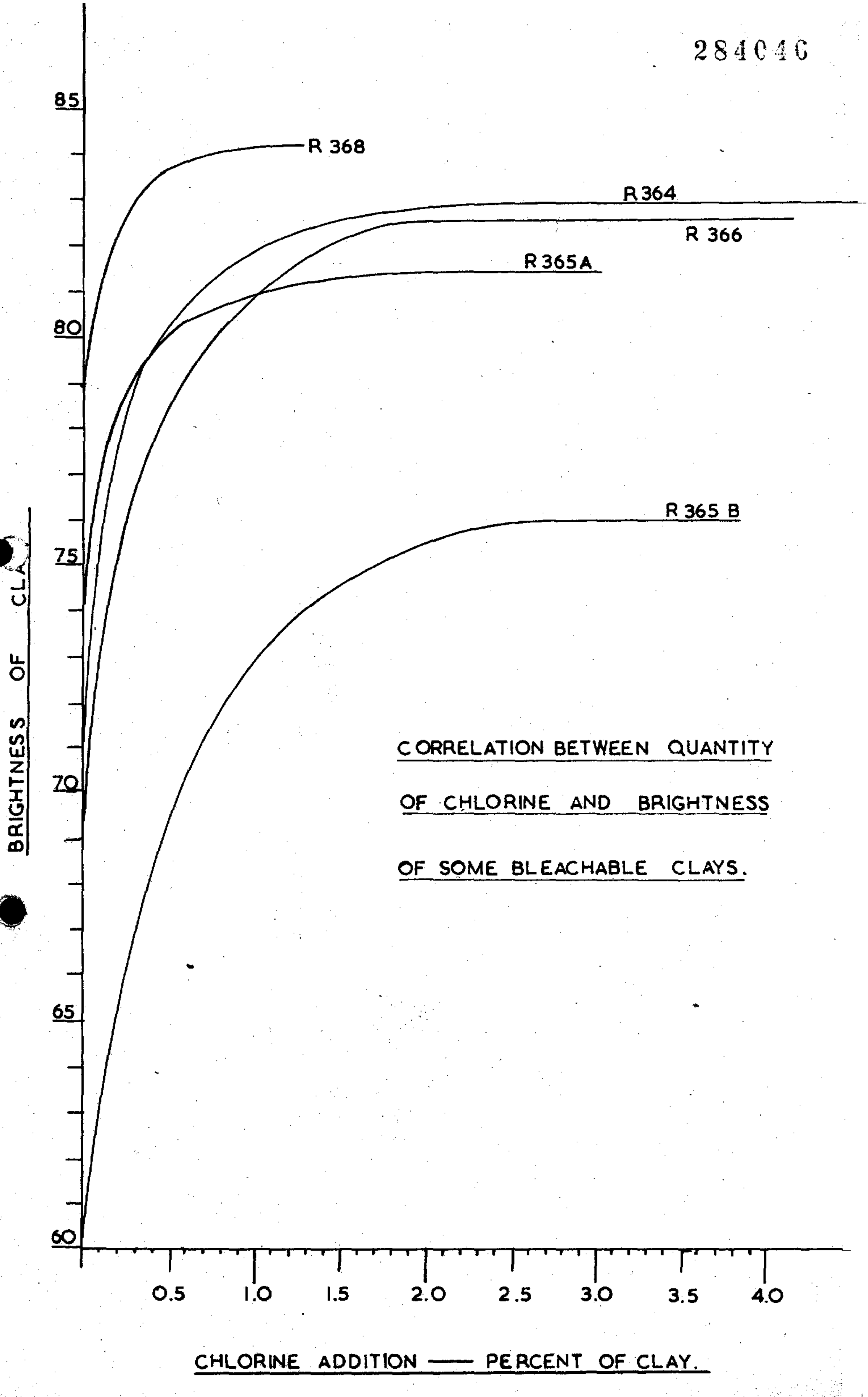
Quantity of chlorine added: % of clay	Brightness After 26 hours contact
Nil	69.4
0.14	73.8
0.27	77.2
0.68	79.4
1.36	82.0
2.04	82.8
2.71	82.6

Clay R.368.

Quantity of chlorine added: % of clay	Brightness After 18 hours contact
Nil (triplicate)	78.8, 78.6, 79.2
0.45 (duplicate)	83.4. 84.0
0.46	83.8
0.93	84.2

The tests on clays R.364 and R.365 B above indicate that there is little, if any, gain in brightness due to contact times in excess of 5 hours. Differences obtained in these series appear to be due to random experimental error. Minimum contact times for bleaching were not determined.

The above tests show that the brightness increases with chlorine additions up to about 2 percent chlorine, but the maximum gain in brightness is obtained from the initial 0.1 to 0.5 percent chlorine addition.


85
80
75
70
65
60
R 368
R 364
R 366
R 365 A
R 365 B
BRIGHTNESS OF CLAY
CORRELATION BETWEEN QUANTITY OF CHLORINE AND BRIGHTNESS OF SOME BLEACHABLE CLAYS.
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
CHLORINE ADDITION —— PERCENT OF CLAY.

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From the attached graph an approximation can be obtained of the gain in brightness due to successive additions of 0.1 percent chlorine.

Addition of 0.1 % chlorine in range	Gain in Brightness Units R.364	R.365 A	R.365 B	R.366
Nil to 0.1	3.9	2.8	2.3	3.0
0.1 to 0.2	3.0	1.6	2.2	3.0
0.2 to 0.3	1.2	0.8	1.6	2.1
0.3 to 0.4	0.7	0.3	1.4	0.5
0.4 to 0.5	0.5	0.4	0.9	0.6
0.5 to 0.6	0.4	0.2	0.9	0.4
0.6 to 0.7	0.4	0.2	0.7	0.4
0.7 to 0.8	0.3	0.2	0.7	0.5
0.8 t6 0.9	0.2	0.1	0.7	0.6
0.9 to 1.0	0.2	0.1	0.5	0.4

The above table clearly shows how the initial addition of 0.1 percent chlorine gives a substantial increase in brightness of the clays. Subsequent additions give further increases, but the gain per unit chlorine addition rapidly decreases. In commercial practice, it is probable that maximum economic gain in brightness would be obtained with an addition of 0.2 or 0.3 percent chlorine, taking int6 account the fact that some of the clays do not require bleaching to meet the brightness standard.

Bleaching tests, quoted above, on samples R.364-R.366 may give the impression that the brightness of all clays from the area are bleachable with chlorine. This is not so. Some clays of low colour are stained with iron minerals or chlorite, and are virtually unbleachable with chlorine. The following series of clays were prepared by decantation. Half of the separated clay was bleached with 1.0 percent chlorine (as hypochlorite) with 18 hours contact.

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Sheet No...20.....

Sample No.	Brightness of original clay	Brightness of bleached clay	Increase in brightness due to bleaching
1219.	71½	78½	7
1220.	71	79	8
1221.	80	80½	½
1222.	70½	83	12½
1223.	71½	79½	8
1224.	74	80½	6½
1225.	73	79	6
1226.	71½	72	½
1227.	61	70	9
1228.	61½	66	4½
1229.	71	79	8
1230.	71	77	6
1231.	69½	76½	7
1233.	76	80½	4½
1234.	75	79½	4½
1235.	74½	77	2½

In this series the increase in brightness due to bleaching varied from a ½ unit to 12½ units, with 12 of the 16 clays showing increases of between 4½ and 9 units. The two samples showing a gain of only ½ unit can be regarded as unbleachable. Although it must be remembered that one of these samples had a high initial brightness, and it is perhaps unreasonable to expect any increase.

Cost of Bleaching, using Launceston Manufactured Sodium Hypochlorite.

Cost of sodium hypochlorite has been quoted by I.C.I.A.N.Z as 6/6 per gallon F.O.R. Launceston. Freight and handling to South Mount Cameron, plus return to Launceston of the empty container, substantially increases the cost of hypochlorite at South Mount Cameron.

The cost of bleaching (using hypochlorite) at South Mount Cameron has been estimated (for chemicals only) per ton of clay, as

£1.3 using 0.2 percent available chlorine,

£2.0 " 0.3 " " " ,

£3.3 " 0.5 " " " .

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Cost of Bleaching using South Mount Cameron Manufactured Sodium Hypochlorite.

An estimate can be made of the cost of manufacture of sodium hypochlorite at South Mount Cameron. However, the proposed industry is not likely to employ more than one or two technical officers, and it may be a burden for a small industry to manufacture its own sodium hypochlorite.

I.C.I.A.N.Z. have quoted as follows:-

Solid Caustic Soda: 868 lb. drums. £68 per ton ex wharf, Launceston.

Chlorine.

Chlorine has been priced at £88:15/- per ton, F.O.B., Melbourne, and with freight and handling charges, including return empties to Melbourne, the cost at South Mount Cameron amounted to £122 per ton.

Without taking into consideration capital cost of the hypochlorite plant, or labour and power etc. required, the cost of bleaching has been estimated for chemicals only per ton of clay as-

£0.4, using 0.2 percent available chlorine

£0.6, " 0.3 " " "

£1.0, " 0.5 " " "

Bleaching with Sodium Hypochlorite, Summary.

1. Sodium hypochlorite is an effective bleach for some coloured clays from South Mount Cameron area. Other coloured clays are not bleached by chlorine.

2. There is no advantage in using bleach contact times in excess of 5 hours. Minimum contact times were not investigated.

3. There is gradual increase in the brightness of bleachable coloured clays with chlorine additions up to about 2 percent.

4. Maximum gain in brightness per unit of chlorine occurs with the initial addition. Subsequent additions give further increases in brightness, but the gain per unit chlorine addition rapidly decreases.

5. Cost of sodium hypochlorite has been estimated per ton of clay as-

Usage of chlorine	Launceston manufactured	Sth. Mt. Cameron manufactured
0.2 %	£1.3	£0.4
0.3 %	£1.9	£0.6
0.5 %	£3.2	£1.0

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Destroying Excess Hypochlorite after Bleaching.

Presence of excess available chlorine in the pulp after bleaching could result in severe corrosion of the equipment used in later dewatering and drying operations.

Sodium bisulphite, or sodium thiosulphate can be used to destroy excess chlorine.

After addition of excess sodium hypochlorite to a clay pulp, with irregular agitation and overnight contact, the odor of chlorine is noted in the pulp. We have no data concerning the ability of different clays to consume chlorine. It is reasonable to assume that this ability will vary widely, and it may depend upon the organic content of the clay.

Residual Chlorine Related to Initial Chlorine Addition.

A series of filtration tests carried out on clay R.361 to determine the effect of sodium hypochlorite as a flocculant proved abortive. The filtrates gave some interesting data relating to chlorine consumption after overnight contact.

Quantity of chlorine added: % of clay	Available Chlorine in Filtrate as % of clay in original pulp	Available Chlorine in Filtrate as % of initial addition	pH Value of filtrate
Nil	Nil	-	6.8
0.07	0.01	14	-
0.14	0.07	50	7.9
0.28	0.18	64	8.0
0.43	0.29	67	8.0
0.57	0.40	70	8.2
0.85	0.61	72	8.5
1.14	0.95	83	8.7

It will be noted that the quantity of residual chlorine in the filtrate increases rapidly with increased initial hypochlorite addition.

Some data are also available from pilot plant filtration work on bleached pulps. Full details of this work is given later. The following extract relates to the residual chlorine content of the filtrate.

1. Filtration test on clay R.368, first run. Chlorine added to pulp 0.50 percent of clay. Available chlorine in solution after overnight contact and filtration, 38 percent of initial addition.

2. Filtration test on clay R.368, second run. Chlorine added to pulp 0.50 percent of clay. Available chlorine in solution after overnight contact and filtration, 38 percent of initial addition.

Consumption of chlorine in these two pilot plant tests is much higher than that obtained in laboratory tests. This difference may be due to use of commercial equipment in the pilot plant as against glass and plastic equipment used in the laboratory tests.

Cost of Destroying Excess Chlorine.

I.C.I. A.N.Z. have quoted sodium bisulphite at £76 per ton ex wharf, Bell Bay, and sodium thiosulphate at £70 per ton ex wharf, Bell Bay.

On the basis of these costs, sodium bisulphite will destroy excess chlorine at about one quarter of the cost of an equivalent quantity of sodium thiosulphate.

The cost of destroying each 0.1 percent available chlorine with sodium bisulphite at South Mount Cameron has been estimated at £0.13 per ton of clay.

Purging Excess Chlorine with Air or Vacuum.

An attempt was made to purge the excess chlorine from a pulp bleached with sodium hypochlorite, by blowing air through the pulp for 16 hours. Negligible drop in the available chlorine content of the pulp was obtained. Similar tests with pulp maintained under vacuum for some 18 hours showed negligible drop in available chlorine content.

Destruction of Excess Chlorine with Unbleached Clays.

Laboratory work has indicated that some of the clays at South Mount Cameron do not require bleaching. It may be possible to destroy a slight excess of chlorine by blending bleached pulps with pulp not requiring bleaching. No work was carried out along these lines.

Effect on Brightness when destroying excess chlorine.

A sample of R.368 clay was prepared by decantation and divided into a number of similar samples. Identical quantities of sodium hypochlorite were added to two pairs of samples. After irregular agitation, and overnight contact, one set of samples was flocculated with aluminium sulphate and filtered. The chlorine content of the filtrate was then determined, and the equivalent quantity of sodium thiosulphate to reduce the excess chlorine was calculated. An excess of sodium thiosulphate was then added to destroy the excess chlorine in the duplicate samples. The brightness of each filter cake was determined after drying etc.

Quantity of Cl added as % of clay	Cl. in filtrate as % of clay	$Na_2S_2O_3$ (Calc.) to destroy excess Cl. as % of clay	Actual $Na_2S_2O_3$ added to destroy Cl. as % of clay	Cl. in filtrate solution	Brightness of clay	Decrease in Brightness due to $Na_2S_2O_3$ (units)
Nil	Nil				79.2	
0.46	0.32				83.8	
0.46		2.2	2.8	Nil	82.0	1.8
0.93	0.67				84.2	
0.93		4.8	5.6	Nil	82.8	1.4

The decrease in brightness due to the destruction of the excess chlorine by thiosulphate is about 1½ units. It is not known whether this degradation by brightness can be expected with other clays under similar conditions.

The proportion of residual chlorine in this test is of the same order as that obtained during earlier laboratory tests from similar initial additions of hypochlorite.

If the proposed industry expects to use chlorine bleach and later destroy excess chlorine chemically, more work on the above lines should be carried out.

Destroying Excess Chlorine, Summary.

1. After bleaching with hypochlorite there remains some residual available chlorine in the pulp solution.

2. The quantity and proportion of residual chlorine in the pulp solution increases with increase in initial hypochlorite addition.

3. Residual chlorine from pilot plant scale tests was considerably less than that obtained from laboratory tests using similar initial hypochlorite addition.

4. Nought point one percent of excess chlorine per ton of clay can be destroyed with sodium bisulphite for £0.13.

5. Chemical destruction of excess chlorine may cause some degradation in brightness. Further consideration of this aspect is warranted if the proposed industry expects to use chlorine for bleaching.

6. It may be possible to destroy excess chlorine by blending bleached and unbleached clays.

Bleaching with Chlorine Gas.

Bleaching with gaseous chlorine, as opposed to bleaching with sodium hypochlorite, offers some obvious advantages in cost of chemicals. It is to be expected that use of gaseous chlorine will be more difficult technically.

Several series of tests were carried out to determine the relative merits of the two methods of bleaching.

Samples of clays R.365 A, R.365 B and R. 366 were prepared by decantation after agitation with sodium silicate. The prepared clays were divided into a number of similar samples in sealed plastic bottles. Various pre-determined quantities of sodium hypochlorite solution or chlorine gas were then added to each sample as required. The samples were agitated at irregular intervals and allowed to stand overnight. The samples were then filtered etc. and the brightness of each sample determined.

In the tables following, the quantity of chlorine added refers to the actual addition. No account was taken of residual chlorine.

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Sheet No. 25.

Clay R.365 A.

Chlorine added: % of clay	Brightness of Clay Chlorine Gas	Hypochlorite
Nil	74½	74½
0.11	77	78½
0.22	79	78
0.32	78½	79½
0.43	80	78½
0.54	80	79
0.65	80½	79½
0.75	81	79
0.86	80½	80
0.98	81	-

Clay R.365 B.

Nil	60	60
0.11	62½	63
0.22	64½	64½
0.34	68	65½
0.45	69	65
0.56	71	63½
0.67	71½	67½
0.78	72½	69
0.89	73½	69

Clay R.366.

Nil	68	68
0.10	73	72
0.20	75½	75
0.30	76½	78
0.40	76½	75
0.50	77½	77
0.60	76½	79
0.70	77½	79
0.80	77½	80
0.90	78	-

On the basis of these tests, it seems that gaseous chlorine is as effective as chlorine from sodium hypochlorite in bleaching the South Mount Cameron clays.

The actual addition of gaseous chlorine to the clay pulp may present handling and corrosion problems. I.C.I.A.N.Z. have indicated that they will offer technical advice on the handling of chlorine gas for bleaching.

Cost of Bleaching with Gaseous Chlorine.

As previously stated the total cost of chlorine at South Mount Cameron is approximately £122 per ton of chlorine.

Cost of bleaching (chemicals only) per ton of clay is thus-

£0.24 using 0.2 percent chlorine

£0.37 " 0.3 " "

£0.61 " 0.5 " "

This cost is considerably less than that derived using sodium hypochlorite.

Flocculation by Gaseous Chlorine.

Addition of gaseous chlorine will flocculate clay dispersed with sodium silicate. The degree of flocculation increases (within limits) with increase in chlorine added. The flocculation is accompanied by a decrease in pH value.

Similar samples of clay R.366 were obtained by decantation after dispersion with sodium silicate.

Varying quantities of chlorine were then bubbled through the pulp in measuring cylinders. A high proportion of the chlorine passed through the pulp and was not absorbed.

Chlorine added to pulp % of clay	pH Value of pulp
Nil	7.3
0.76	4.6
1.40	3.1
2.16	2.8
4.32	2.4

Bleaching with Chlorine Gas, Summary.

1. Chlorine gas is as effective as sodium hypochlorite for bleaching stained clays.

2. Chlorine gas readily flocculates clay dispersed with sodium silicate.

3. Cost of bleaching by gaseous chlorine is appreciably less than bleaching with sodium hypochlorite, considering chemicals only. The following table shows the comparative chemical cost of chlorine gas and sodium hypochlorite manufactured at Launceston and South Mount Cameron.

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Usage of chlorine % of clay	Gaseous chlorine	Launceston manufactured	Sth. Mt. Cameron manufactured
0.2	£0.24	£1.3	£0.40
0.3	£0.37	£1.9	£0.6
0.5	£0.61	£3.2	£1.0

Flocculation and Filtration.

The naturally occuring clay has a pH value of about 4 and is in a flocculated condition, but was dispersed with sodium silicate to allow best conditions for degritting in dense pulps. The clay, dispersed with 10 lb./ton of sodium silicate, will on standing gradually re-flocculate. In addition, many other chemicals will readily flocculate the clay, but insufficiently to allow good subsequent filtration rates. These chemicals include sulphuric acid, sodium hypochlorite and chlorine. Of various flocculants tested to date, aluminium sulphate has been found the most satisfactory.

The high molecular weight water soluble synthetic polymers are extremely effective flocculants. However, as the filter cake must be readily redispersible, the use of these reagents appears unlikely.

Filtration Tests.

Considerable preliminary work on filtration was carried out with Buchner funnel filtration tests. This work gave much information which was incorporated in later filter leaf tests and pilot plant scale filtration tests. In view of the data obtained from pilot plant scale filtration tests, it is proposed to merely summarize filter leaf tests, and to mention only such Buchner funnel tests as are relevant.

Laboratory Test Leaf Procedure.

A laboratory test leaf filter was supplied by Chemical Plant & Engineering Co. Pty. Ltd., Melbourne. The laboratory test leaf filter has an effective filter area of 6" x 4". The filter is fitted with interchangeable cloths, strings for cake discharge etc. A tabulation, supplied with the filter, quotes times for filtration and dewatering corresponding to various filter speeds at different drum submergences. As the pilot plant filter had a drum submergence of 37½ percent, all laboratory test leaf filtration tests corresponded to 37½ percent submergence.

When discussing laboratory leaf filter test results, the time cycle of the test is summarised as "filter speed,r.p.m.", and the time cycles employed correspond to this filter speed, in r.p.m., at 37½ percent submergence.

A vacuum of 15-16 inches of mercury was used throughout filter leaf tests.

Pilot Plant Filter.

The pilot plant was a 3 feet diameter by 1 foot face width, FEinc string discharge rotary vacuum filter. The filter was made available by Chemical Plant & Engineering Co. Pty. Ltd. of Melbourne.

The filter bath has two overflow ports , corresponding to 25 and 37½ percent submergence. All tests were carried out at 37½ percent submergence. With the tank supplied, the submergence could not be increased above 37½ percent.

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Filter cloth used for the bulk of the tests was No. 416 Terylene cloth, obtained through Australian Titan Products Pty. Ltd., Tasmania. String used was "Emu" Brand No. 5, Bricklayers Macrame.

The filter was fitted with provision for rollers, but these were not used during plant tests. Vacuum used on both filtration and dewatering cycles was 15-16 inches of mercury, with a little fluctuation due to the cyclic nature of the valves etc.

Filter speed is variable, and is controlled by a variable pitch Reeves pulley. Motors of 960 and 1440 r.p.m. were used, and filter speeds in the range of one revolution in 20 minutes to one revolution in 2¾ minutes were possible.

Probably due to the lack of elevation between the filtrate receivers and the position of the filtrate pump, the filtrate pump was not effective. No filtrate pump was used during the tests, as the filtrate receivers proved more than adequate to store the filtrate during the period of the tests. No problems were encountered during filtration test runs.

Quantity of Aluminium Sulphate for Flocculation.

Buchner funnel tests showed that an increase in the quantity of aluminium sulphate increased the filtration rate of the clay within limits. The requirements of various clays varied somewhat, but generally it was found that additions of between 15 and 45 pounds of aluminium sulphate per ton of clay gave reasonable filtration rates. Beyond a certain point additional aluminium sulphate gives no further benefit, and may result in slightly decreased filtration rates. In general, a series of Buchner funnel tests was carried out to obtain an approximation of the optimum addition of aluminium sulphate before filter leaf tests.

Cost of Aluminium Sulphate for Flocculation.

On the basis of Launceston prices, the cost of aluminium sulphate at South Mount Cameron has been estimated at £45 per ton.

The consumption of aluminium sulphate appears to vary from clay to clay, and we have used between 15 and 45 pounds per ton, with a general average of about 25 or 30 pounds per ton.

Usage pounds per ton	Cost of Aluminium Sulphate, per ton of clay
15	£0.3
30	£0.6
45	£0.9

Filter Leaf Tests.

Filter leaf tests were carried out on the bulk samples R.361, R.364, R.365 A, R.365 B, R.366 and R.368. A number of variables were investigated, including filter speed, increase in temperature and effect of rolling on moisture content of the filter cake.

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1. Preparation of Clay Pulps.

The clays were prepared by decantation after ¼ hour agitation with 14 pounds of sodium silicate per ton of crude clay. Sodium hypochlorite was added to give the equivalent of 0.5 percent available chlorine. Aluminium sulphate was added at a quantity predetermined by Buchner funnel tests. The quantities used are given below.

All pulps had a specific gravity of 1.180-1.190, equivalent to 25-26 percent solids, except clay R.361, which had a specific gravity of 1.24, equivalent to 31 percent solids. Temperatures of the pulps are stated below.

Filter cloth Terylene 416 was used on all the tests detailed below.

2. Filtration Rates at Various Filter Speeds.

Clay R.368.

Temperature: atmospheric = 9°C.

Aluminium Sulphate: 45 lb./ton.

Filter Speed r.p.m.	Filtration Rate: lb./sq.ft./hr. of dry clay	Moisture in Filter Cake %
1/16	3.7	41.7
1/14	4.2	42.0
1/12	4.2	43.0
1/10	4.7	42.3
1/8	5.3	43.1

Clay R.361.

Temperature: atmospheric = 8° to 10°C.

Aluminium Sulphate: 25 or 26 lb./ton.

Filter Speed r.p.m.	lb./sq.ft./hr. of dry clay	Moisture Content of Filter Cake *	Thickness of Filter Cake inches *
1/16	4.9	-	-
1/16	5.3	46.1	13-14/64
1/16	5.1	48.5	14-16/64
1/16	5.2	48.0	13-14/64
1/14	-	47.1	13/64
1/12	5.4	48.1	11-12/64
1/10	5.9	48.2	9-10/64
1/8	7.2	-	-
1/8	7.0	48.0	10/64

* Some cakes were rolled for reduction of moisture tests: other cakes were rolled over half the area. Figures quoted for moisture content and cake thickness refer to unrolled cakes.

The filtration rate increases significantly with increase in filter speed. The actual thickness of filter cake decreases with increase in filter speed-this aspect is discussed later.

3. Filtration Rates at Elevated Temperatures.

Buchner filtration tests established that increased pulp temperature markedly increased filtration rates. The effect of temperature upon filtration rate when all other conditions remain constant is shown in the series below using clay R.368 and aluminium sulphate 45 lb./ton.

Temp. °C	Filter Speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry cake	Moisture Content of cake %
9	1/8	5.3	43.1
32	1/8	7.7	41.3
43	1/8	7.5	40.4
52	1/8	8.2	40.8
61	1/8	9.0	40.2

Because the filter speed has remained constant, and the filtration rate has increased, the thickness of filter cake has naturally increased considerably.

A better camparison of filtration rates should be possible when the cake thickness is the same in both cases. This is readily attained by using increased filter speeds at the elevated temperatures.

Clay	Temp. °C	Aluminium Sulphate lb./ton of clay	Filter speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry cake	Moisture content of filter cake %	Filter Cake thickness (approx.) inches
R.361.	8	26	1/16	5.3	46.1	13-14/64
	40	26	1/10	9.7	46.6	16-20/64
	57	26	1/8	9.5	46.9	18-19/64
R.364.	11	44	1/16	4.0	47.2	12/64
	58	44	1/8	8.0	47.6	13/64
R.365 A.	11	28	1/16	5.2	46.8	12/64
	56	28	1/8	11.6	43.4	13/64
R.365 B.	9	46	1/16	3.9	44.0	10-11/64
	62	46	1/8	9.3	44.0	13-14/64
R.366.	10	30	1/16	3.1	47.8	8-9/64
	59	30	1/8	6.5	47.2	9-10/64
R.368.	9	45	1/16	3.7	41.7	-
	61	45	1/8	9.0	40.2	-

The tabulation on the previous page shows clearly how the filtration rate is approximately doubled by heating the pulp from about 8-10°C to about 60°C.

Buchner funnel tests indicated further advantages by boiling or almost boiling the pulp. Reliable test leaf filtration data on boiling pulps was not obtained due to the difficulties in maintaining other conditions constant, and to experimental manipulation difficulties, with boiling viscous pulps.

Correlation between Moisture Content of Filter Cake and Fineness of Clay.

The moisture content of the filter cakes of the various clays tested vary in the range 40-48 percent moisture, and is related to the fineness of the clay. This is illustrated in the following table, which compares the moisture contents of the filter cakes obtained from filter leaf tests at atmospheric temperatures and equivalent 1/16 r.p.m., with the quantity of minus 2 micron material in the minus 30 micron separated clays.

Clay	% Moisture in Filter Cake	% Minus 2 Micron Material in minus 30 Micron Clay
R.366.	48	56
R.361.	46	45
R.364.	47	41
R.365 A.	47	38
R.365 B.	44	37
R.368.	42	34

On the very fine clays, such as centrifuge overflows, containing between 60 and 70 percent minus one micron material, the moisture content of the filter cake is approximately 60 percent.

Effect of Pressure Rolling on Moisture Content of Filter Cake.

FEinc string discharge filters are equipped so that the filter cake can be pressure rolled to reduce the moisture content of the cake, if so desired. A series of leaf tests were carried out on clays R.361 and R.368 to determine the effect of rolling on the moisture content of the filter cakes. In some cases, half of the cake on the filter leaf was rolled, and half remained unrolled. In other cases the test was duplicated, and the whole cake rolled or not rolled respectively.

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Clay R.368.

Temp. °C	Filter Speed r.p.m.	% Moisture In Unrolled Cake	% Moisture In Rolled Cake	Decrease in moisture content of Filter Cake %
9	1/16	46.6	41.7	4.9
9	1/14	46.0	42.0	4.0
9	1/12	46.7	43.0	3.7
9	1/10	46.5	42.3	4.2
9	1/8	47.0	43.1	3.9
32	1/8	46.6	41.3	5.3
43	1/8	45.7	40.4	5.3
52	1/8	44.9	40.8	4.1
61	1/8	46.0	40.2	5.8

Average decrease in moisture content of filter cakes due to rolling 4.6 percent.

Clay R.361.

Temperature: 8 to 10°C

Filter Speed r.p.m.	% Moisture in unrolled cake	% Moisture in rolled cake	Decrease in moisture content of filter cake %
1/16	48.5	45.1	3.4
1/16	48.0	46.2	1.8
1/12	48.1	45.3	2.8
1/10	48.2	46.0	2.2
1/8	48.0	45.3	2.7

Average decrease in moisture content of filter cake due to rolling 2.6 percent.

Laboratory Filtration Tests, Summary.

1. Clay dispersed with sodium silicate can readily be flocculated by various chemicals. Aluminium sulphate was the best flocculant tested.

2. Quantity of aluminium sulphate for optimum filtration rates varies with different clays, and usage was in the limits of 15 to 45 pounds per ton of clay.

3. Cost of aluminium sulphate is:

Usage: pounds per ton	Cost, per ton of clay
15	£0.30
30	£0.60
45	£0.90

4. Moisture content of filter cakes varied with different clays in the range 40 to 48 percent moisture, and depends upon the fineness of the clay.

5. Increased filter speeds, resulting in thinner filter cakes, result in significantly increased filtration rates.

6. Elevated pulp temperatures significantly increase filtration rates of the clays: an increase of about 50 centigrade degrees approximately doubles filtration rates.

7. Pressure rolling of filter cakes reduces the moisture content of the cake on an average of 4.6 percent for clay R.368, and 2.6 percent for clay R.361.

Pilot Plant Filtration Tests.

1. Preparation of Pulps.

Pulps for filtration were prepared in the pilot plant by cycloning in a 30 m.m. cyclone after dispersion with sodium silicate, as discussed earlier in the report.

The cyclone overflow was collected in 10 gallon storage drums. Sodium hypochlorite, equivalent to 0.50 percent of available chlorine, was added to the pulp, and after vigorous agitation for a few minutes the pulp was allowed to stand overnight.

The pulp was flocculated next morning with aluminium sulphate, which was added just prior to filtration. The addition of the aluminium sulphate resulted in a very viscous pulp, which made thorough mixing in of the last of the flocculant rather doubtful. The pulp was added to the filter tank as required.

The filter tank is provided with an agitator, and it is assumed that this agitator gave a uniform feed in the filter tank.

Quantities of aluminium sulphate added were predetermined by Buchner funnel tests.

Clay R.361.

Pulp density: 31 percent solids: specific gravity of pulp 1.24.

Temperature: atmospheric = 9°C.

Aluminium Sulphate: 23 pounds per ton.

Duration of test: minutes	Filter speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry cake	Moisture content of cake %	Cake thickness inches
30	1/20	3.4	47.2	14/64
30	1/15	4.2	47.0	12/64
30	1/12	4.5	46.3	10-11/64
20	1/9½	5.0	46.3	9-10/64

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Clay R.368.
First Run

Pulp density: 30 percent solids: specific gravity of pulp 1.23.

Temperature: atmospheric = 8°C.

Aluminium Sulphate: 45 pounds per ton.

Duration of test: minutes	Filter speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry cake.	Moisture content of cake %
30	1/17	4.1	46.8
30	1/15	4.1	46.6
20	1/9½	5.2	46.0
15	1/6½	6.5	46.0
8	1/3¾	8.5	45.9

Quantity of available chlorine added for bleaching = 0.50 percent of clay.

Quantity of available chlorine in filtrate liquors after filtration = 0.19 percent of clay.

Consumption of chlorine was 62 percent of that added. This proportion is considerably higher than that obtained under laboratory tests (see earlier discussion).

Clay R.368.
Second Run

Pulp density: 27 percent solids.

Temperature: atmospheric = 9°C.

Aluminium Sulphate: 45 pounds per ton.

Duration of test: minutes	Filter speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry cake	Moisture content of cake %
10	1/6⅓	5.7	46.6
10	1/3 2/3	7.0	46.6
10	1/2 9/10	8.3	46.5
10	1/2¾	9.1	45.6

Quantity of available chlorine added for bleaching = 0.50 percent of clay.

Quantity of available chlorine in filtrate liquors after filtration = 0.19 percent of clay.

Consumption of chlorine was 62 percent of initial addition. This consumption is identical with that obtained from the first run with clay R.368.

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This second run on clay R.368 was carried out to determine the effect of comparatively high filter speeds on filtration. Most of the clay was discharged at the alignment comb, rather than from the discharge roll.

Thickness of Filter Cake.

The FEinc string discharge filter is designed so that the filter cake is lifted from the drums by the series of strings provided. As the strings change direction over the discharge roll, the cake falls from the strings. As the strings return to the drum, they pass through the alignment comb. This comb quickly builds up with pieces of filter cake, and eventually the returning strings are wiped clean of adhering lumps of filter cake by the accumulation of clay.

With clay filter cakes thicker than about ⅛ inch, good discharge from the strings occurs at the discharge roll, but as mentioned above, any clay adhering to the strings is wiped off at the comb, and eventually there is a small but regular discharge of clay at this point.

With cakes less than about ⅛ inch, discharge from the roll decreases and most of the clay discharges from the alignment comb.

Carried still further, with very thin cakes such as produced with centrifuge effluents (see later) and relatively high filter speeds, no cake is discharged at the rolls, and all is discharged at the alignment comb.

We can see little objection to this type of discharge. The clay builds up on the combs irrespective of the cake thickness, and the strings are then pulled through an accumulation of clay on the combs. It does not seem to matter whether only a small part or whether the majority of the clay is discharged at the comb, as far as wear on the strings or comb is concerned. This argument applies only to very fine clay, with virtually no quartz or other abrasive material. It is suggested that this method of operation be referred to manufacturers for their opinion.

Increasing filter speed significantly increases the capacity of a filter. For example, the following figures are taken from the two runs with clay R.368.

Filter Speed r.p.m.	Filter Capacity lb./sq.ft./hr. of dry cake
1/17	4.1
1/15	4.1
1/9½	5.2
1/6½	6.5
1/3¾	8.5

It appears from these, and similar, figures that if thin filter cakes, i.e. comb discharge can be accepted, then a significant increase in filtrate rate can be obtained. From the above table, it appears that the filtration rate can be doubled by increasing the filter speed four times.

Now, previously, it was found that increasing the temperature of the pulp prior to filtration to about 60°C roughly doubled the filtration rate.

If comb discharge can be combined with filtration of heated pulp, it appears that filtration rates about four times greater than those obtained at atmospheric temperatures may be attained. As previously mentioned all tests with the pilot plant filter were at atmospheric temperature.

Pilot Plant Filtration, Summary.

1. Pilot plant filtration work confirmed previous filter leaf test results, regarding filtration rates, moisture content of cakes, and increase in filtration rates with increase in filter speeds, resulting in the formation of thinner filter cake.

2. A marked increase in filtration rates is possible if thin cakes can be discharged at the alignment comb. Manufacturers advice regarding this proposal should be sought.

Some Economic Consideration Regarding Filtration.

Filter Area Required.

At atmospheric temperatures, the following filtration rates were obtained by leaf and pilot plant tests on the various clays.

Nature of test	Clay	Filter Speed r.p.m.	Filtration Rate lb./sq.ft./hr. of dry clay
Pilot Plant	R.361	1/15	4.2
Filter Leaf	R.364	1/16	4.0
" "	R.365 A	1/16	5.2
" "	R.365 B	1/16	3.9
" "	R.366	1/16	3.1
Pilot Plant	R.368	1/15	4.1

Average of the six clays is 4.1 lb./sq.ft./hour.

Assuming-

(1) an annual rate of production of 10,000 tons per year of 50 weeks,

(2) a five day week, 24 hour operation,

(3) filter working time: 85 percent of possible,

then filter area required = 1100 sq. ft.

On the same basis, but assuming a seven day week, and working time of 85 percent of possible, the filter area required is 770 sq. ft.

Heating the filter feed pulp to about 60°C reduces these areas to about 560 sq.ft. on a five day basis, and 390 sq.ft. on a seven day basis.

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Cost of Heating Pulp Prior to Filtration.

Two fuels can be considered-

(a) furnace oil at £17.9 per ton at South Mount Cameron, calorific value 18,500 B.Th.U./lb.

(b) Tasmanian coal at £6 per ton at South Mount Cameron, calorific value 10,000 B.Th.U./lb.

Assuming a clay pulp at 30 percent solids, the heat required to raise the temperature 90 Fahrenheit degrees (i.e. 140°F) per 100 tons of pulp is 15,500,000 B.Th.Units.

Assuming an overall efficiency of 70 percent then the quantity of coal required would be one ton. At the cost of £6 per ton for coal, the cost per ton of clay would be £0.2.

On the same basis, the cost of furnace oil to heat the pulp would be £0.32 per ton of clay.

On the basis of fuel cost only, Tasmanian coal is considerably cheaper as a source of fuel.

Economics of Heating Prior to Filtration.

The relative economic advantage of heating pulps will depend upon a number of inter-related factors.

1. The increase in filtration rate, listed above, due to heating can be summarized:-

	Atmospheric			Elevated			Increase in	
Clay	Temp. °C	Filtration Rate lb/sqft/hr.	Filter Speed r.p.m.	Temp. °C	Filtration Rate lb/sq.ft./hr.	Filter Speed r.p.m.	Filtration Rate %	pulp temp. °C
R.361.	8	5.3	1/16	57	9.5	1/8	79	49
R.364.	11	4.0	1/16	58	8.0	1/8	100	47
R.365A.	11	5.2	1/16	56	11.6	1/8	123	45
R.365B.	9	3.9	1/16	62	9.3	1/8	138	53
R.366.	10	3.1	1/16	59	6.5	1/8	110	49
R.368.	9	3.7	1/16	61	9.0	1/8	143	52
				AVERAGE OF SIX CLAYS			115	49

On the above clays, an increase in temperature of about 50 centrigrade degrees will roughly double the filtration rate.

2. Cost of fuel to heat the pulp through 50 centrigrade degrees was shown to be £0.20.

3. The cost of a drum filter 8 ft. x 10 ft., with a filter area of 250 sq. ft., covered against corrosion, has been estimated at between £8000 and £9000.

4. For present purposes we can assume that the annual cost of filtration due to capital charges, amortization and maintenance is 30 percent of the capital cost of the filter.

5. Filter area required under varying conditions was previously assessed as-

Pulp Temperature	Filter Area Required 5 Day Week	7 Day Week
Atmospheric	1100	770
Heated to 60°C	560	390

6. On the basis of £8500 per 250 sq. ft. of filter area, and assuming pro rata capital cost, then capital cost of filters will be.

Pulp Temperature	Estimated Capital Cost 5 Day Week	7 Day Week
Atmospheric	£37,400	£26,200
Heated to 60°C	£19,000	£13,300

7. Thirty percent of capital charges, over 10,000 tons per year will be-

Pulp Temperature	Estimated Cost/Ton Clay 5 Day Week	7 Day Week
Atmospheric	£1.12	£0.79
Heated to 60°C	£0.57	£0.40

8. Labour to operate the filters is assumed to be one man per shift, irrespective of the number of filters involved. Weekly wage of £15 per week of five eight hour shifts is assumed. Cost of labour, based on 10,000 tons of clay per year is-

£0.23 per ton of clay on a five day week.

£0.32 per ton of clay on a seven day week.

A summary of the various factors follows.

Operating Conditions Temperature of pulp	Operating week	Fuel Cost per ton of clay	Capital Charges per ton of clay	Labour	Total Cost
Atmospheric	5 day	-	£1.12	£0.23	£1.35
"	7 "	-	£0.79	£0.32	£1.11
Heated to 60°C	5 "	£0.20	£0.57	£0.23	£1.00
Heated to 60°C	7 "	£0.20	£0.40	£0.32	£0.92

Economic Conditions, Summary.

These preliminary estimates indicate that there is an economic advantage in heating the pulp prior to filtration. They also indicate that operation on a seven day week is more economic than five day operation.

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Drying.

No experimental work has been carried out to determine the relative advantages or economics of various methods of drying the finished clay.

Effect of Increased Temperatures on Brightness of Various Clays.

Some methods of drying involve contact of the clay with gas inlet temperatures of the order of 1000°F. In such cases there could be a short period of overheating. To determine the possible effect of excessive heat on various clays, a series of tests was carried out on natural and bleached clays. The clays tested all had a low initial brightness due to organic staining. The clays were maintained at the temperatures stated for half an hour.

(a) Tests on Bleached Clays.

	Brightness after Heat Treatment					
Clay	Heated to 105°C.	Heated to 122°C.	Heated to 190°C.	Heated to 250°C.	Heated to 350°C.	Heated to 450°C.
R.364	83	82½	82	76½	74½	73
R.365A	80½	80½	79	73½	70½	68
R.365B	74½	74½	71½	66½	65	64½
R.366	81½	81½	79	77	71½	69½

The brightness of the clays is unchanged at 122°C, and has dropped only slightly at 190°C. At temperatures of 250°C and higher, there is appreciable degradation in brightness.

(b) Tests on Unbleached Clays.

	Brightness after Heat Treatment					
Clay	Heated to 105°C.	Heated to 122°C.	Heated to 190°C.	Heated to 250°C.	Heated to 350°C.	Heated to 450°C.
R.364	71	71	66	62½	68	69
R.365A	74	74½	71½	66½	65	64½
R.365B	60	59½	56	51	59½	62
R.366	69½	69	65	66	69	69

The brightness of these unbleached clays gradually decreased with increase in temperature to 250°C. At higher temperatures the brightness increased again, and at 450°C, the brightness of three of the four clays approximated the initial brightness.

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Cost of Drying.

The moisture content of the filter cake varies somewhat with the different clays.

Clay	Average moisture content from filter tests
R.361	46½%
R.364	47½%
R.365A	46½%
R.365B	44 %
R.366	47½%
R.368	46½%
Average 6 clays	46½%

Pressure rolling may reduce this moisture by 2 - 4 percent, and redispersion with phosphates (see later) may increase it by a similar amount. These possibilities are neglected for present purposes.

Assuming

(i) a weekly production of 200 tons of clay

(ii) initial temperature of filter cake 10°C

(iii) use of Tasmanian coal of 10,000 B.T.U./lb

(iv) cost of coal at South Mount Cameron £6: -: - per ton

then the quantity of coal required, and the cost per ton of clay will depend on the overall thermal efficiency (of fuel combustion and heat transfer) as follows

Overall thermal efficiency	Tons of coal per week	Cost of coal per ton of clay
25	80	£2.4
30	66	£2.0
40	50	£1.5
50	40	£1.2

These calculations assume the final moisture content of the finished product to be reduced to zero. In practice, it is possible that the moisture content of the clay would be reduced to somewhere between 3 and 10 percent, dependent upon the method of drying. Costs above relate merely to the cost of the fuel, and does not allow for capital costs, operation maintenance or depreciation etc.

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Redispersion of Filter Cake.

The filter cake, flocculated with aluminium sulphate and containing about 47 percent moisture, can be liquified and the clay redispersed by the addition of small quantities of tetra sodium pyro phosphate.

No quantitative tests have been carried out to determine the minimum quantity of pyro phosphate required for redispersion. Several qualitative tests indicated that clay R.368 was readily liquified by agitation with about 9 lb. of pyro phosphate per ton of clay.

Under similar conditions Ballarat Clay Company Pty. Ltd. redisperse filter cake with 5 lb. per ton of tetra sodium pyro phosphate, and there seems little reason why this quantity would not suffice on South Mount Cameron clays.

A.P.P.M. carried out tests on a sample of clay R.361 prepared in the pilot plant in the usual manner. The sample was divided into two parts.

Part "A" was evaporated to dryness in an air oven in the dispersed state.

Part "B" was flocculated with 30 lb. per ton of aluminium sulphate and then dried.

In a letter dated 22nd August, 1960, A.P.P.M. made the following comment regarding these samples.

"The Mines Department sample A was readily dispersible with a small quantity of sodium silicate, ¼% N84 grade. Sample B was also dispersible with silicate but the requirement was higher around 1½%. However, filtration before drying may reduce the demand. The A.P.P.M. bleached sample was dispersible with 1¼% silicate. As a point of interest sodium hypochlorite was added to sample A dispersed with 1% silicate and flocculation did not occur until 3% of available chlorine had been added.

The following methods were used to measure the degree of dispersion:-

(1) Time for a given volume to pass through an orifice as a measure of viscosity.

(2) Viscosity as measured with a wire type torsion viscometer.

(3) Sensitivity of a hydrometer placed in the slurry.

(4) Visual appearance.

Viscosities of the Mines Department samples slurried with

(a) No dispersant addition,

(b) 1% Calgon,

(c) 1½% N84 grade sodium silicate,

were compared with viscosities of slurries of E.T.M., Ballarat, English, Rodda and Huber clays.

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If viscosity can be taken as a reliable measure of dispersion then the Gladstone clay can be satisfactorily dispersed at a concentration of 3 lb. solids per gallon with sodium silicate. Of the clays listed above, Ballarat clay is the only one which disperses satisfactorily without addition of chemical. Indications are that calgon is no more effective a dispersent for the alum flocculated clay than silicate under the conditions of test. It may well be that at high solids concentration the silica gel may absorb so much water that viscosity is increased merely by lack of water and not by failure to disperse.

Dried lumps from samples A and B disintegrated or "slaked" readily in solutions of sodium silicate and indications are that our mixing plant could handle this material in either dispersed or flocculated form.

Ballarat clay is dried in a dispersed state, and this probably accounts for the ease of dispersion of the clay at A.P.P.M., as mentioned above.

If the South Mount Cameron clay is dried in the dispersed condition, it is assumed that it will behave in a similar manner to Ballarat clay.

If the South Mount Cameron clay is dried in the flocculated state, the tests carried out by A.P.P.M. indicate that it will be readily dispersible to A.P.P.M. requirements.

I.C.I.A.N.Z. have quoted Tetra sodium pyro phosphate at £135 per ton (in 56 lb. paper bags) at Launceston. Cost at South Mount Cameron has been estimated at £140. Cost of 5 lb. per ton of clay would be £0.32.

There are a number of different methods by which the clay filter cake can be dried. Unless it is necessary to liquify the cake before drying by such methods as drum driers (as at Ballarat) or spray driers, there is no point in redispersing the filter cake with tetra-sodium pyro phosphate.

We have no knowledge of the method of drying that will be used in the proposed commercial plant.

Use of Centrifuge for dewatering clay.

There are two possible applications for a centrifuge in the preparation of the clay

(1) as a method of dewatering a large proportion of the clay prior to drying, leaving a relatively dilute pulp to be filtered.

(2) as a method of making coating grade clays by stripping portion of the finest sizes into a second product, to be filtered and dried separately,

This second application was not investigated by us and for this reason only a limited number of sizings were determined.

The Experimental Centrifuge.

The Burton 7" centrifuge was hired from Roy Burton and Co. Pty. Ltd., Melbourne.

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The unit is fitted with an agitated feed tank, gear pump for feed to the centrifuge, and a by pass valve to control the feed rate. Centrifuge speed can be varied by changing drive pulleys. All centrifuging tests were carried out at speeds of 2450 r.p.m., producing a centrifugal force of 600 times gravity.

Initial Tests.

These tests were carried out at comparatively low densities and at variable feed rates. The pump installed in the unit did not allow the centrifuge to be fully loaded under conditions of test. These initial tests are of limited value as thickening tests as the feed rates were well below the capacity of the centrifuge, and the pulp densities are considerably less than likely to be used in practice. Tests were carried out on a sample of R.361, prepared by normal pilot plant procedure.

Test No.	Product	Pulp density % solids	Pulp rate gals/hr	Pulp rate lbs/hr of solids	% recovery of solids
1	Discharge	53	1.0	7.9	76
	Effluent	4	5.8	2.5	24
	Feed	15	6.8	10.4	100
2	Discharge	53	2.0	16.2	60
	Effluent	7	14.3	10.8	40
	Feed	15	16.3	27.0	100
3	Discharge	52	1.4	10.4	77
	Effluent	7	4.8	3.1	23
	Feed	20	6.2	13.5	100
4	Discharge	54	3.1	24.8	63
	Effluent	9	14.1	14.6	37
	Feed	20	17.2	39.4	100
5	Discharge	54	2.6	20.6	72
	Effluent	11	6.8	8.0	28
	Feed	26	9.4	28.6	100
6	Discharge	55	4.0	33.3	61
	Effluent	14	13.4	20.7	39
	Feed	26	17.4	54.0	100

These tests show that when the centrifuge is handling feed rates well below its capacity

(1) the pulp density of the centrifuge underflow is relatively constant at 53 - 55 percent solids, irrespective of feed rates.

(2) an increase in the feed rate increases the proportion of solids reporting in the effluent

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(3) an increase in density, while maintaining the same feed rate in terms of pounds of solids per hour, increases the proportion of solids reporting in the discharge.

The centrifuge effluent from test No. 6 was sized and gave the following results.

Size product	Percent weight
+6 microns	0.2
-6+1 microns	31.9
-1 micron	67.9
	100.0

The discharge from test No. 6 was sized with a Bouyoucos hydrometer and gave the following results.

Size product	Percent weight
+30 microns	4.0
-30+20 "	1.5
-20+10 "	18.0
-10+ 5 "	23.0
-5+ 1 "	44.5
-1 micron	9.0
	100.0

These sizings indicate that about 70 percent of the minus ore micron material reported in the centrifuge effluent under the conditions of test.

Capacity of Centrifuge and Effect of Feed Rate.

A series of tests was carried out on clay R.361 prepared by the standard pilot plant method to determine the capacity of the centrifuge. The pulp density was 31 percent solids. To carry out this series of tests, the pump supplied with the centrifuge unit was replaced by a larger gear pump.

One centrifuge test was also carried out on a sample of clay R.368 prepared by normal pilot plant methods.

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Sheet No....45:.....

Clay	Test No.	Product	Pulp density % solids	Pulp rate gals/hr	Pulp rate lb/hr of solids	% recovery of solids
R.361	7	Discharge	54	4.6	37.1	45
		Effluent	20	19.8	45.2	55
		Feed	31	24.4	82.3	100
R.361	8	Discharge	54	7.3	60.0	38
		Effluent	22	37.5	98.3	62
		Feed	31	44.8	158.3	100
R.361	9	Discharge	55	8.0	66.8	39
		Effluent	22	39.9	104.3	61
		Feed	31	47.9	171.1	100
R.361	10	Discharge	55	9.8	82.5	28
		Effluent	25	70.2	207.8	72
		Feed	32	80.0	290.3	100
R.361	11	Discharge	55	9.9	82.5	24
		Effluent	27	81.1	263.2	76
		Feed	31	91.0	345.7	100
R.368	12	Discharge	55	7.5	62.7	75
		Effluent	13	14.8	21.0	25
		Feed	31	22.3	83.7	100

The tests on clay R.361 show

(1) the pulp density of the discharge is constant at 54 - 55 percent solids, confirming initial observations.

(2) the experimental centrifuge has a maximum discharge rate of about 9.8 - 9.9 gallons per hour under the conditions of the test. This occurs at feed rates of about 80 gallons per hour, under the conditions of the test.

(3) an increase in the feed rate increases the proportion of solids reporting in the effluent.

A comparison between test No. 7 with R.361 and test No. 12 with R.368 is interesting. Under very similar conditions of feed rate and pulp density, there is a marked difference in the proportion of solids reporting in the respective products. Thus with clay R.361, 55 percent of the feed reports in the effluent. With clay R.368, only 25 percent of the clay reports in the effluent.

A tentative conclusion from these results was that clay R.368 contains considerably less very fine material (say minus 1 or minus 2 micron). This conclusion is supported to some extent by Bouyoucos hydrometer sizings on samples of clay prepared from clays R.361 and R.368 by decantation under similar conditions.

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Sheet No. 46.

Size product	Percent weight R.361	R.368
+10 microns	14.0	11.5
-10+ 5 microns	17.0	23.5
- 5+ 2 microns	24.0	30.5
- 2 microns	45.0	34.5

Thickening Dispersed Pulp by Settlement.

The application of a centrifuge to thicken clay pulps has been detailed. Pulps can also be thickened by settlement and tests were undertaken to show this application. The process can also be applied to produce coating clays and is practised in various commercial plants.

A series of settlement tests were performed on dispersed clay R.361, prepared by normal pilot plant methods. The tests were carried out in glass measuring cylinders with a settling column of 15 inches. The pulp density of the feed was 31 percent solids.

Test No.	Settlement time hours	Thickened product % solids	% weight
1	1½	54	22
2	4	55	33
3	5	55	38
4	6	56	39
5	30	56	61
6	92	57	80

The density of the thickened pulp was 54 - 57 percent solids, which is very similar to that attained with the centrifuge. Increase in settlement time naturally increases the proportion of solids in the thickened product.

Similar results can be obtained with the centrifuge by varying the feed rate, and it appears probable that any thickening made by the centrifuge could be approximated by settlement over prolonged periods.

Filtration of Centrifuge Overflow.

Use of the centrifuge will decrease the volume of clay pulp to be filtered, and will greatly decrease the solid content of the same pulp. This can reduce the cost of flocculant and redispersant. However the pulp to be filtered contains the finest fractions of the clay and this has a much lower filtration rate from dilute pulps than the normal clay.

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Filtration of the whole clay was successful on No. 416 Terylene cloth. No. 416 cloth was unsuitable for filtration of the centrifuge overflow at atmospheric temperatures, as the bulk of the pulp passed through the filter cloth. Two multi-filament nylon cloths, No.'s 206 and 212, were received from Chemical Plant and Engineering Co. Pty. Ltd., Melbourne, for filtration of these fines. Although much more closely woven, these cloths still allow passage of some fines during filtration at atmospheric temperatures. At temperatures of 50 - 60°C, little or no clay passes through the cloths, due doubtless to better flocculation at these higher temperatures.

Clay R.361.

Filter leaf tests were carried out on a sample of centrifuge overflow from test No. 6, outlined previously. Data from this test were:-

Test No.	Product	Pulp density % solids	Pulp rate gals/hr	Pulp rate lb/hour of solids	% recovery of solids
6	Discharge	55	4.0	33.3	61
	Effluent	14	13.4	20.7	39
	Feed	26	17.4	54.0	100

Data relating to filtration tests on the effluent were:-

Pulp density 14% solids.
Bleaching with 0.69% chlorine as hypochlorite.
Aluminium sulphate 30 lb. per ton.
Temperature 8°C.
Filter cloth multi-filament nylon No. 212.
Submergence 37½%.
Vacuum 15 inches mercury.

Filter speed r.p.m.	Filtration rate lb/sq.ft/hr of dry clay	Moisture content of filter cake	Thickness of filter cake, inches
1/16	2.3	57.7	7/64
1/12	2.6	58.6	6-7/64
⅛	3.2	59.5	5-6/64
1/6	3.7	62.7	4-5/64
average		59.6	

The filtrate contained a small quantity of very fine clay.

Filter cakes at the higher filter speeds were very thin, and although they lifted off the filter cloth freely, they did not fall away from the strings (see previous discussion on "Thickness of filter cake").

Various filter leaf tests at atmospheric temperatures concerning clays equivalent to the centrifuge feed have been detailed before. They can be briefly summarized as follows:-

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Filter speed r.p.m.	Filtration rate lb/sq.ft/hr of dry clay	Moisture content of filter cake	Thickness of filter cake, inches	Aluminium sulphate lb/ton
1/16	5.3	46.1	13-14/64	26
1/16	4.9	-	-	25
1/16	5.1	48.5	14-16/64	25
1/16	5.2	48.0	13-14/64	25
1/14	-	47.1	13/64	25
1/12	5.4	48.1	11-12/64	25
1/10	5.9	48.2	9-10/64	25
1/8	7.2	-	-	25
1/8	7.0	48.0	10/64	25
average		47.7		

Due to the inter relation of filter speed, cake thickness and filtration rates, it is difficult to directly compare filtration rates obtained for centrifuge feed and centrifuge overflow. The practicability or otherwise of comb discharge of filter cake is also most important.

An approximate comparison can be made from the available data, if it is assumed that average filtration rates are as follows:-

Filter speed r.p.m.	Filtration rate lb/sq.ft/hr of dry clay — Centrifuge feed	Centrifuge effluent
1/16	5.1 (average of 4)	2.3
1/12	5.4	2.6
1/8	7.1 (average of 2)	3.2
Average of 3 speeds	5.9	2.7

This comparison is faulty to the extent that filter cakes from the centrifuge feed are much thicker than those obtained from the centrifuge effluent. If the thickness of the cakes from the centrifuge feed was reduced to that from the centrifuge effluent, then the filtration rates from the centrifuge feed would average considerably above the figure of 5.9 lb/sq.ft/hr used in the comparison.

Then using from the above data:-

(1) centrifuge overflow contains 39 percent of the solids originally contained in the centrifuge feed.

(2) filtration rate of centrifuge feed is 5.9 lb/sq.ft/hr
filtration rate of centrifuge overflow is 2.7 lb/sq.ft/hr.

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then the time to filter the overflow using the same filter conditions is 85 percent of that required to filter the whole feed.

If similar cake thickness for the filter cake from filtration of both centrifuge feed and overflow are assumed, it is probable that the time to filter the whole of the feed would be less than that required to filter the centrifuge overflow under the conditions of the tests.

There is a considerable saving in consumption of aluminium sulphate and tetrasodium pyrophosphate. The centrifuge overflow requires 30 lb/ton of aluminium sulphate, which is equivalent to 11.7 lb/ton for the whole of the clay. This compares with 25 lb/ton used for centrifuge feed, and represents a saving of about 54 percent of the aluminium sulphate.

On a similar basis, because the centrifuge overflow contains 39 percent of the feed, then consumption of tetrasodium pyrophosphate for redispersion will be only 39 percent of that required for the whole of the feed.

The saving in cost of chemicals by use of the centrifuge appears to be.

Filter feed	Aluminium sulphate per ton clay £	Tetra-sodium hyrophosphate per ton clay £	Total chemicals per ton clay £
Whole of pulp	0.52	0.32	0.84
Centrifuge overflow	0.24	0.12	0.36
Saving due to centrifuge	0.28	0.20	0.48

The use of tetrasodium pyrophosate is applicable only for the drying of a dispersed pulp such as required as feed to a steam or spray dryer. If drying of the flocculated filter cake is to be considered the costs of the dispersant would not apply.

From the filter tests quoted above, the average moisture of a filter cake from the whole of the pulp is 47.7 percent moisture, and that from the centrifuge overflow is 59.6 percent moisture.

The average moisture content of the combined centrifuge products prior to drying is then

Product	Proportion of solids in original pulp	% moisture
Centrifuge discharge	61	45
Centrifuge overflow	39	59.6
Feed to drier	100	50.7

Use of a centrifuge therefore increases the moisture content of the clay to be dried from 47.7 percent to 50.7 percent.

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The cost of coal to dry the respective products can be estimated from data previously given.

Feed to drier	Cost per ton of clay (£) Overall fuel efficiency 25%	30%	40%	50%
Whole of clay, ex centrifuge	2.8	2.4	1.8	1.5
Whole of clay, not centrifuged	2.5	2.1	1.6	1.3
Extra cost due to centrifuge	0.3	0.3	0.2	0.2

Assume an average of these figures, viz. £0.25 per ton of clay as the extra cost of drying.

Cost of a centrifuge is perhaps £10,000. Assuming capital charges, amortization operation and maintenance to amount to 30 percent p.a. over production of 10,000 tons of clay per year, then the economics of a centrifuge for partially dewatering clay R.361 become

	Cost per ton of clay
Cost of centrifuge	£0.30
Plus extra cost of drying	£0.25
Less saving in chemicals	£0.48

then extra cost of using centrifuge is £0.07 per ton of clay.

It would seem that use of a centrifuge on clays similar to R.361 is marginally uneconomic as a dewatering means. If at the same time, it is desired to produce a coating clay, then this can be readily done with the centrifuge at more or less nominal cost.

Clay R.368.

Pilot plant filter tests were carried out on whole feed and centrifuge overflow pulps of clay R.368, produced by normal pilot plant methods.

Centrifuge performance was

Test No.	Product	Pulp density % solids	Pulp rate gals/hr	Pulp rate lb/hour of solids	% recovery of solids
12	Discharge	55	7.5	62.7	75
	Effluent	13	14.8	21.0	25
	Feed	31	22.3	83.7	100

Data relating to pilot plant filtration of the effluent are:-

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Pulp density: 13 percent solids.
Bleaching with 0.5 percent chlorine as hypochlorite.
Aluminium sulphate: 50 lb per ton.
Temperatures: 10°C.
Filter cloth: multi filament nylon No. 206.
Submergences: 37½ percent.
Vacuum: 15 inches mercury.

Filter speed r.p.m.	Filtration rate lb/sq.ft/hr of dry clay	Moisture content of filter cake	Remarks
1/7⅚	3.0	60	average of 2 tests.
1/2¾	5.8	-	all comb discharge: very thin cake.

Filter cloth 206 allowed some clay to pass through, and a total of 13 percent of the clay originally in the feed was recovered in the filtrate. This is similar to, but more pronounced, than batch tests using cloth 212. The 212 cloth was not available in sufficient quantity for use on the 3' x 1' filter.

As before, it is difficult to arrive at a correct basis of comparison of filtrate rates. An approximate comparison can be made from the data extracted from earlier tests, using similar filter speeds.

Pilot plant tests on R.368 whole feed gave filtration rates of 5.7 and 6.5 lb/sq.ft/hour which were obtained at filter speeds of 1/6½ and 1/6⅓ r.p.m. respectively. Average rate 6.1 lb/sq.ft/hr. On centrifuge overflow of R.368, a filtration rate of 3.0 lb/sq.ft/hr was obtained at a filter speed of 1/7⅚ r.p.m.

At higher filter speeds, using comb discharge of filter cake, respective rates were

(a) whole feed, rates of 8.3, 8.5 and 9.1 lb/sq.ft/hour for speeds between 1/3¾ and 1/2¾ r.p.m. Average rate 8.6 lb/sq.ft/hr.

(b) centrifuge overflow 5.8 lb/sq.ft/hr for filter speed of 1/2¾ r.p.m.

An average of the two sets of rates gives

Filter speeds	Filtration rates lb/sq.ft/hr Whole feed	Centrifuge overflow
medium speeds 1/6 - ⅛ r.p.m.	6.1	3.0
high speeds 1/2¾-1/3¾ r.p.m.	8.6	5.8
Average	7.4	4.4

As before, with this comparison the filter cakes from the centrifuge feed are much thicker than those obtained from the centrifuge effluent. If similar cake thicknesses were compared, then the rates for filtration of the feed would be appreciably higher than the average figure of 7.4 lb/sq.ft/hr used in the comparison.

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Using the data above viz.

(1) centrifuge effluent contains 25 percent of the solids contained in the original feed

(2) filtration rate of whole feed is 7.4 lb/sq.ft/hr
filtration rate of overflow is 4.4 lb/sq.ft/hr.

then the time to filter overflow using the same conditions is about 42 percent of that required to filter the whole feed. (Aluminium sulphate for whole feed 45 lb/ton solids Aluminium sulphate for centrifuge overflow 50 lb/ton solids Tetra sodium pyrophosphate 5 lb/ton solids).

As in the case of R.361 above, it is possible to estimate the savings in chemicals due to the centrifuge for clay R.368 as

Filter feed	Aluminium sulphate £ per ton of clay	Tetra sodium pyrophosphate £ per ton of clay	Total chemicals £ per ton of clay
whole of pulp	0.90	0.32	1.22
centrifuge overflow	0.26	0.08	0.34
saving due to centrifuge	0.64	0.24	0.88

The average moisture content of the combined centrifuge products prior to drying is then

Product	Proportion of solids in original pulp	% moisture
Centrifuge discharge	75	45
Centrifuge overflow (Filter Cake)	25	60
Feed to drier	100	48.8

Average moisture content of whole of filter cake is 46.3 percent from the two pilot plant tests on R.368.

The centrifuge thus increases the moisture content of the feed to the drier from 46.3 to 48.8 percent.

The cost of coal to dry the respective products can be estimated as:-

	Cost per ton of clay £			
	Overall fuel efficiency			
Feed to drier	25%	30%	40%	50%
whole of clay, ex centrifuge	2.6	2.2	1.7	1.3
whole of clay, not centrifuged	2.4	2.0	1.5	1.2
Extra cost due to centrifuge	0.2	0.2	0.2	0.1

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Assume an average of these figures, viz. £0.2 per ton of clay as the extra cost of drying.

As before, cost of the capital, amortization and maintenance of the centrifuge per ton of clay can be estimated at £0.30.

Now filter area required to filter centrifuge overflow as shown above, is only 42 percent of that required to filter the whole of the pulp. The capital charges of filter plant to treat the whole feed was previously estimated at

Pulp temperature	Estimated cost/ton clay. 5 day week	7 day week
Atmospheric	£ 1.12	£ 0.79
Heated to 60°	£ 0.57	£ 0.40

A saving of 58 percent of this cost amounts to an amount between £0.23 minimum and £0.65 maximum. Assume an average of £0.44.

Then the economics of installing a centrifuge become

	Cost per ton of clay.
Cost of centrifuge	0.30
plus extra cost of drying	0.20
total extra cost	0.50
less saving in chemicals	0.88
less saving in filter costs	0.44

overall saving due to centrifuge is £0.82 per ton of clay.

In this case the saving due to a centrifuge is substantial.

On the basis of these preliminary figures on clay samples R.361 and R.368, it appears that use of a centrifuge will show a small loss in cases similar to that of R.361 examined, and will show a substantial gain on clays similar to the case of R.368 examined. Maximum economic gain would appear to be attained when the centrifuge is used to produce the maximum proportion of solids in the discharge.

Apart from the above considerations, use of a centrifuge allows the separate production of a coating grade clay. If there is any substantial market for coating grade clays, then installation of a centrifuge becomes attractive.

The preliminary nature of the above discussion relative to the merits of the application of a centrifuge should be emphasised.

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Appendix. Sheet No. 54.

Evaluation of clays from the South Mount Cameron deposit.

Evaluation of weathered granite in terms of paper clay.

The method originally used by A.P.P.M. was briefly:-

(1) Weigh 250 grams of the previously dried but uncrushed sample into a tall beaker.

(2) Cover with 6 inches of water, containing 10 g. of sodium silicate.

(3) Allow to stand overnight, if possible.

(4) Agitate violently for one minute with a stirring rod and let stand for 5 minutes.

(5) Decant the top 3 inches of pulp.

(6) Rebulk to 6 inch mark, stand for 5 minutes, and again decant.

(7) Repeat until a total of 5 decantations have been made.

(8) Flocculate pulp with one gram of alum, and let stand.

(9) Decant clear liquor, filter settled solids, and dry at low temperature.

(10) Dry and weigh residue.

(11) Evaluate the separated clay for yield, brightness, ignition loss, grit etc.

This method of evaluation, with settling times of 5 minutes for the depth of 3 inches has been found to give a size separation slightly in excess of 20 microns. The clay produced has a maximum particle size comparable with high quality, commercially prepared paper filler clay.

The method can be modified in some respects without affecting the evaluation.

(A) It is usually more convenient to use semi-wet as-received crude clay for evaluation. The yield and quality of the clay is unaltered.

For example:-

Sample No.	Condition of crude clay as tested	Separated clay % Yield	Brightness	Ignition loss
R.361/4	Wet, as received.	35.5	78.0	13.2
R.361/5	Wet, as received.	35.6	77.6	13.1
R.361/6	Dry.	36.8	77.8	13.1

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(B) Quantity of sodium silicate used for dispersion can be reduced to 1.0 grams without alteration to quality of clay separated.

Bore hole No.	Footage ft.	grams sodium silicate	Yield of clay	Brightness of clay	Ignition loss of clay
3	4-8'9"	1	44.1	84.8	13.0
		10	43.4	84.2	13.4
3	15-20	1	40.1	84.6	13.1
		10	40.9	84.8	13.1
19	surface	1	31.2	83.0	12.7
		10	31.0	83.0	12.8
19	7-12	1	39.8	83.0	13.5
		10	39.4	82.2	13.5
26	5-10	1	54.9	79.2	12.8
		10	53.4	79.0	12.7

The above method has been used in evaluating all weathered granite from Brown's area in terms of paper clays by the Mines Department Laboratory, using semi-wet crude clays and 1.0 gram of sodium silicate for dispersion.

Evaluation of Paper Clays for 30 micron Separation.

The method presently used by A.P.P.M. Ltd. is as follows:-

" 30 Micron Separation.

Weigh 100 grams of the O.D. sample, (if in lump form DO NOT GRIND but break lumps with fingers), into a milk shake mixer container and add 125 mls of calgon solution (40 g/litre) + 375 mls water and disperse on the mixer for about half an hour. Transfer to a tall 1 litre beaker and dilute to a mark about half an inch from the top. Stir and allow to settle for 5 mins. Syphon the top 3 inches into a Buchner flask. Refil beaker to mark, repeat the stirring and syphoning process twice more at 5 mins. settling and four times at 2 mins. settling time. Combine all decantations add about 10 mls. 10 percent alum and allow to settle. Decant clear liquor dry residue and weigh as minus 30 micron. The residue remaining in the beaker is allowed to settle and dried and weighed as plus 30 micron."

Evaluation of South Mount Cameron Clays by A.P.P.M.

The deposit at South Mount Cameron was tested by a series of boreholes during 1959. Laboratory test work was carried out by A.P.P.M. to evaluate the deposit as a source of paper filler clay.

Clay separations were made by the 30 micron separation method given above.

The results of this test work are fully recorded in the report of A.P.P.M. dated 21st September, 1959.

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Evaluation of some Borehole Samples by the Mines Department Laboratory.

Duplicate samples of some of the borehole samples tested by A.P.P.M. were evaluated in the Mines Department Laboratories by the "Evaluation of weathered granite in terms of paper clay," with the modifications mentioned previously.

Comparison of Evaluations by A.P.P.M. and Mines Department Laboratories.

There were some marked differences in yield and quality of the separated clays obtained by the two methods of evaluation. These may be summarized as follows:-

Bore hole No.	Footage	A. P. P. M. Yield of -30 micron clay	A. P. P. M. Ignition loss %	A. P. P. M. Brightness	Department of Mines Yield of -20 micron clay	Department of Mines Ignition loss %	Department of Mines Brightness
3*	4-8'9	51.7	13.0	79	44.1	13.0	84.8
	9-15	55.5	13.0	78	42.3	13.1	83.8
*	15-20	52.5	12.9	78.5	40.1	13.1	84.6
*	23-25	45.7	12.8	77	34.9	13.0	82.8
6*	10-20	46.5	12.9	79.5	32.7	12.9	82.8
*	20-25	43.5	12.5	76	29.2	12.2	81.2
11*	1- 6	48.9	13.5	73 **	35.7	13.5	78.2
*	6-10	52.6	12.7	77	37.7	13.3	81.4
*	10-15	49.5	13.2	75.5	40.4	13.4	83.4
*	15-20	50.4	12.7	76.5	38.1	13.5	83.8
*	20-23	49.8	12.8	77	38.9	13.3	83.8
17*	6-10	52.7	13.2	67 **	42.9	13.7	75.4
	10-15	50.5	12.2	67.5**	37.2	13.7	74.2
*	15-20	49.1	12.8	69.5**	40.3	13.3	75.6
19	Surface	46.6	12.5	78.5	31.2	12.7	83.0
*	2- 7	49.1	12.7	75.5**	37.4	13.9	80.4
	7-12	51.3	12.5	76.5	39.8	13.5	83.0
*	12-17	50.3	12.7	77.5	40.4	13.3	82.8
26*	5-10	71.5	13.0	76.5	54.9	12.8	79.2
	10-15	47.0	11.4	76	30.0	11.3	78.6
*	15-18	49.3	12.7	79	30.5	12.3	79.0
	22-25	45.0	11.6	76.5	32.4	12.1	81.2
Average of 22 clays		50.4	12.7	75.8	37.8	13.1	81.2

Samples marked with asterisk were reported by A.P.P.M. as satisfactory for grit contents. Samples marked ** were reported as bleachable.

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Comparison of the Two Evaluations.

A comparison of the 22 samples shows marked variations in yield and quality. Average figures relating to the samples from 6 bores, more or less taken at random, show

Evaluation by	Separated clay Yield	Ignition loss %	Brightness
A.P.P.M.	50.4	12.7	75.8
Mines Department	37.8	13.1	81.2

It will be seen that the procedure adopted by A.P.P.M. resulted in a greater yield of an inferior clay. There are several reasons for this.

(1) The A.P.P.M. method gives basically a minus 30 micron separation, and that used by the Mines Department gives a 20 micron separation.

(2) The A.P.P.M. method requires violent agitation for 30 minutes, as against 1 minute for Mines Department method.

(3) The A.P.P.M. method requires 7 decantations, as against 5 decantations as used by the Mines Department.

There are some data to indicate that the extra agitation degrades the clay slightly, and that the coarser fractions recovered by the 30 micron separation are of relatively low grade material.

Two lots of duplicate samples were prepared by decantation after agitation with sodium silicate. Test conditions were identical except that one pair of duplicates was agitated for 30 minutes, and the other pair for 1 minute in the milk shake mixer. Clay was recovered from 3 decantations, allowing settling times of 5 minutes each.

Agitation	Separated clay Yield %	Brightness
1 minute	41.5	73.0
1 minute	41.5	72.8
Average	41.5	72.9
30 minutes	44.9	71.8
30 minutes	44.2	72.2
Average	44.5	72.0

The above table indicates an increase in yield of 3 percent, and a drop of 1 unit in brightness.

The residues from the above tests were then decanted further, following the two evaluation methods outlined earlier. For the material agitated for 1 minute, 2 further decantations, each of 5 minutes settling time were made. For the material agitated initially for 30 minutes, 4 decantations, each of 2 minutes settling time were made.

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Agitation	Final decantation	Clay from final decantations Yield %	Brightness
1 minute	2 at 5 mins	2.4	70.8
1 minute	2 at 5 mins	2.2	69.6
Average		2.3	70.2
30 minutes	4 at 2 mins	5.2	64.8
30 minutes	4 at 2 mins	5.3	63.4
Average		5.3	64.1

The greatly increased yield of inferior clay was obtained by the A.P.P.M. method.

Total yield of clay by the A.P.P.M. method was (44.5 + 5.3) = 49.8 percent.

Total yield of clay by the Mines Department was (41.5 + 2.3) = 43.8 percent.

It is considered that the differences in evaluation of the clays as determined by the 2 methods are due to results similar to above.

Commercial practice will probably involve agitation with sodium silicate in thick pulps. It is unlikely that intense agitation will take place in the plant. Consequently it is felt that the evaluation involving comparatively mild agitation with sodium silicate will approach plant conditions, and it is felt that evaluation of the deposit by this method gives a more correct assessment of the raw material for production of high grade filler clay.

Clay

Siliceous Grit Determination.

The method consists of digestion of the clay sample with sulphuric acid to decompose the clay and on dilution and boiling with 5 percent HCl the siliceous grit and silica from the clay are separated from soluble salts by filtration and washing. The silica from the clay is dissolved in hot 5 percent sodium carbonate solution and the remaining insoluble siliceous grit is obtained by filtration washing and ignition of the residue.

References of the method are:

Treadwell & Hall, pages 421 & 422, 29th Ed.

Associated Pulp & Paper Mills Ltd., Burnie, private communication, 1960.

The method was examined in the laboratory and slight amendments made as a result of the investigation. Adequate digestion and final free fuming with sulphuric acid is essential for concordant results.

1. To 2 grams of clay in a 400 ml beaker add 50 ml of water, and carefully add 50 ml of concentrated sulphuric acid. If organic matter is evident, add 2 ml of concentrated nitric acid.

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2. Cover and place on a hot plate with an approximate temperature of 130°C. Stir frequently, and slowly increase temperature to about 200°C in 2½ hours.

3. Remove cover and gradually increase hot plate temperature to approximately 250°C, and continue heating for approximately ¾ of an hour during which period dense fumes are evolved. Cool, add 150 ml of water and 20 ml of concentrated HCl.

4. Cover and heat carefully for ½ an hour with a hot plate temperature of approximately 130°C, stirring frequently.

5. Dilute to 350 ml with hot water, and filter on a Whatman No. 44 filter paper, washing with hot 2 percent HCl.

6. Transfer filter paper and contents to original beaker, add 200 ml of water and 10 grams of Na_2CO_3. Heat for 30-45 minutes at a low hot plate temperature of about 140°C, stirring frequently. Filter through a No. 44 Whatman filter and wash with very hot water until the washings do not give a colour with phenolphthalin.

7. Dry and ignite the filter, and weigh the residue as grit.

The Associated Pulp and Paper Mills Pty. Ltd. submitted a sample of prepared clay from South Mount Cameron to the Australian Mineral Development Laboratories and a copy of this report is shown below.

"MINERALOGY & PETROLOGY SECTION 23rd September, 1960.

REPORT No. 3.0.0/1184.

MATERIAL:	Clay.
SUBMITTED BY:	Australian Pulp and Paper Mills Ltd.
DATE RECEIVED:	22nd September, 1960.
MARKS or NOS:	Order No. F/5166 Reference KAM:JS, Gladstone Refined Clay.
INFORMATION REQUIRED:	X-ray diffraction examination.
METHODS OF EXAMINATION:	X-ray diffraction.

RESULTS OF EXAMINATION:-

The only minerals identified from the X-ray diffractorgraph of the above sample were kaolin, illite and quartz. The kaolin mineral lies between kaolinite and fireclay in the kaolinite-fireclay-halloysite series. Illite is present in small amounts (only a few percentage), and quartz represents approximately 2 percent of the sample.

Neither chlorite nor any titanium minerals were detected from the diffractograph.

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Under the microscope the sample is very fine grained and only lightly iron stained. Only occasional grains of opaque minerals and rutile were seen but chlorite was not detected. Fine grained aggregates of amorphous titanium oxide were seen and this, together with the trace of rutile, accounts for the presence of approximately 1.0 percent of TiO_2 as measured on the X-ray spectrograph.

Signed - A.E. Tynan.
MINERALOGIST."

Research Officer.

Research Metallurgist.

Chief Chemist & Metallurgist.

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Schedule D

028

REPORT

on

THE DESIGN OF A TREATMENT PLANT

to

PRODUCE PAPER QUALITY KAOLIN FROM SOUTH MOUNT CAMERON RAW CLAY

and

AMPLIFY THE ACCOMPANYING LAY-OUT PLAN FOR THE PLANT.

by

W. H. CROPP.

February, 1963.

SUMMARY.

This report describes the treatment plant recommended for an annual production of 10,000 tons of paper quality kaolin from South Mount Cameron raw clay. It is necessarily lengthy since it has to serve as a source book for information on the numerous details needed for preparation of construction drawings, for ordering equipment, and for operation of the plant. A paragraph index is provided to assist in locating information. The salient features of Parts I and II are summarised below. Part III is not amenable to condensation.

The report recommends procedures and equipment similar to the practices well proven at Ballarat for stockpiling, slurrying and degritting the raw clay for heating the degritted kaolin slip. It departs from Ballarat practice in the omission of the centrifuge which there separates the kaolin into coating and filler grades, in the inclusion of chlorine bleaching, and in the variations in filter and dryer equipment which seem certain to follow analysis of information now being gathered.

Of particular moment is the recommendation that, as at Ballarat, the critical final degritting be done with hydroseparators, instead of with the 30 m.m. cyclones used in the Department of Mines Research Investigation on the S.M.C. raw clay. No commercial plant using miniature cyclones has been found, and so despite their virtues, this report cannot shoulder the responsibility of committing the S.M.C. project to the pioneering of cyclones to commercial success, at grave risk to the quality of its sale product. For the sale kaolin has neither assured market nor local value apart from its guaranteed quality attribute.

The attached flowsheet shows the sequence of operations and explains the function of each item of equipment. Whilst the extraction of the kaolin from the raw clay is straight mineral dressing procedure, the processing into sale form requires in the main the techniques of industrial chemistry.

The attached layout plan shows the recommended arrangement of the equipment items. It is the key plan for the preparation of the plant construction drawings by expanding it and substituting equipment manufacturers' dimensions and details. It provides vacant space alongside all the Finishing Section units which would require duplication should future demand call for production in excess of the nominal 14,000 ton full time annual capacity of the plant. The relevant ground areas would need to be left vacant, or used for non-permanent structures only.

The raw materials and services required are:-

(I) Raw Clay: 30,000 cubic yards broken volume annually of which 18,000 cubic yards will need to be stockpiled at the plant during summer for supply over the seven winter months.

(II) Water: 100,000 gallons each week day (if the residue be disposed of by pumping), or 250,000 gallons (if disposal be by gravity launder). The effect of the local river water on the kaolin brightness is unknown and needs investigation.

(III) Power: Motors totalling about 175 name plate H.P., with a quarterly power charge, including lighting, of about £565. A closer estimate will be obtainable from the finished construction plans.

(IV) Labor: Probably 14 operating and 2 maintenance men.

Summary (continued)

Treatment cost commencing with reclamation of raw clay from the plant stockpile and ending with delivery of the dry sale kaolin into the plant bulk storage bin ready for bagging, is estimated at £7 per ton of kaolin. This figure is bare - devoid of any allowance for contingencies.

This report makes no attempt to estimate the capital cost of the plant. No reliable estimate will be practicable until quotations have been obtained for all equipment units, and construction drawings completed to permit the cost of buildings, concrete structures, foundations and services inatallations to be calculated.

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PART I

ACKNOWLEDGEMENT AND INTRODUCTION

1. The foundation on which this design for the South Mount Cameron treatment plant rests is the extensive research on South Mount Cameron raw kaolin samples by the Launceston Research Laboratories of the Tasmanian Department of Mines, the results of which work are collected in the Department's "Ore Dressing Investigation R.361, R.364, R.365, R.366, R.368, dated 21/4/61. This is referred to below as O.D.I. This in turn underlies the same laboratory's "Flowsheet Design" dated 25/7/61, referred to below as F.D. South Mount Cameron is likewise abbreviated to S.M.C.

2. Elaboration of the figures of various test sequences reported as O.D.I. and background information on the behaviour of the raw material during testing was gained from discussions with the officers of the laboratory. Substantial information on raw kaolin processing and the fruit of long experience was gathered from the Ballarat Clay Company's plant through a visit arranged by courtesy of A.P.P.M. Ltd. at the request of Mr. Turner of S.M.C. Both Mr. Davis, General Manager, and Mr. Robertson, Chief Chemist, freely gave me the benefit of their experience in kaolin processing and impressions gained from visits to U.S.A. kaolin plants. Local information was provided by Mr. Turner. Acknowledgment is also made to Mr. W. St.C. Manson, Chief Chemist & Metallurgist, of the Department of Mines, for many helpful discussions and for information based on his 1961 visit to Southern U.S.A. kaolin plants, and to Mr. Kuzniarski of the Launceston Technical College Staff for his assistance with steam heating problems.

3. In filling out and mechanizing the laboratory F.D., I have drawn on a lifetime of experience in the design and operation of commercial mineral dressing plants, large and small. The translation of laboratory research into commercial achievement is not a simple matter. Quirks in raw material behaviour that escaped notice during research can show up and become magnified into problems in the commercial equivalent of research operations. The S.M.C. plant will meet its share of these problems and have to overcome them. But an endeavour has been made to limit them by heeding the experience of the Ballarat Clay Company, and by using only methods and equipment proven successful elsewhere. The Ballarat Co., operating on a deposit of similar raw material, has tried and replaced several plant units installed initially because they looked right for the job and, by modifying others, has hammered out a reliable treatment plant. So it is with any commercial mineral dressing plant seen to be operating efficiently - it is the end product of evolution. After the S.M.C. plant has gone into production it will also start to evolve as operations meet and defeat problems arising from deficiencies in methods and variations in raw material behaviour. So much for the straight mineral dressing section that extracts the kaolin from the raw feed.

4. Subsequent operations in the plant pass from mineral dressing into the sphere of industrial and physical chemistry, notably for the techniques of bleaching, flocculation, filtration, and drying of the extracted kaolin. The special used experience of I.C.I.A.N.Z. Ltd. will need to be sought for the design, installation, operation, and control of the chlorine bleach. Research will be needed on flocculation to lighten the onerous burden of filtration. Aid will have to be obtained

from experienced sources for the best solution to the costly critical drying problem. And since value resides not in the kaolin mineral itself, but in high standards of purity, particle size, and brightness of the sale product, rigid maintenance of these standards will demand considerable technical knowledge and supervision.

5. Finally, detailed specification of all the plant equipment is not complete - it stops after the degritted kaolin slip has been prepared ready for elimination of about half its water content by filtration. More information on U.S. kaolin filtration equipment, capacity figures, and experience, is needed before the type and size of filter can be decided with confidence. A satisfactory method of transferring the wet sticky filter cake to the dryer has to be uncovered, also a suitable cheaper alternative to the natural gas fired spray dryers used throughout U.S. plants. Ways of handling, binning and bagging the hot dry kaolin have to be examined and worked out in detail. A decision has to be made on the most suitable equipment and fuel for producing some 70 tons of steam daily at S.M.C. This report can only outline the general requirements of the equipment for each of these operations, and indicate suitable positions for each unit on the layout plan. One thing is prominent - paper kaolin production on this small scale will be a continuing struggle against excessive production costs.

General Description of the Flowsheet and Treatment Plant

6. Production of paper quality kaolin from the raw material involves two main operations:

(a) Extraction of the wanted kaolin fraction from the crude kaolin raw material.

(b) Processing the extracted kaolin to the form required by the consumer.

The treatment flowsheet and plant accordingly consists of two sections.

(A) The Degritting Section.

(B) The Finishing Section.

(A) The Degritting Section

7. This will take in each day shift 120 tons of raw material from an adjacent stockpile, slurry it with water, extract 33 per cent or 40 tons as paper quality kaolin in the form of a kaolin-water mixture designated "Degritted Kaolin Slip," and deliver this slip to a storage tank in the Finishing Section for processing to marketable form. The balance of 67 per cent or 80 tons will be a quartz grit residue requiring delivery to an adjacent disposal area. The section will operate on day shift, five days a week, and annually process 30,000 dry tons or cubic yards stockpile volume of raw material into 10,000 tons sale kaolin, and 20,000 tons grit residue. The natural location for this section is at S.M.C. adjacent to raw material, water supply, and residue dumping area.

8. The two main operations in the section are:

(i) Reduction of the damp acid raw material to a thick dispersed alkaline slurry, with a limited amount of water containing sodium silicate dispersant.

(ii) Degritting the slurry.

This report recommends slurrying be done in propeller type agitator tanks which are used at Ballarat on very similar raw material. Their main attributes are simplicity and proven reliability. Their chief disabilities are high power consumption, considerable propeller wear, and the necessity for scuttling after a power failure and periodically to get rid of accumulations of oversize grit.

The alternative would be slurrying in a low power revolving drum washer equipped with lifters and discharge trommel for continuous discard of oversize stones. This has attractive features, but would require a pilot scale test to investigate its slurrying efficiency. The slurry passes through a standard rake classifier which, in one simple operation, eliminates some 40 per cent of the raw material as grit coarser than 60 mesh, and overflows the remainder as a thin slurry ready for final degritting.

9. For this final degritting operation which eliminates all remaining grit coarser than sale product specification limit, and separates the wanted kaolin fraction for delivery into storage in the Finishing Section, the Mines Department research program exhaustively investigated and successfully developed the use of miniature cyclones 30 millimeters (1.18 inches) internal diameter. The laboratory's F.D. based on this research work suggests the use of these cyclones in the proposed S.M.C. treatment plant, but indicates hydroseparators as an alternative means.

This report prefers and recommends the use of hydroseparators. Whilst it is a function of a research laboratory to test new types of equipment designed to achieve a desired objective, it is quite another matter for a newborn company to risk its existence and production pioneering that new equipment application through to commercial success. This is a job for an established producer who would be risking only the minor capital cost and finished product of an experimental pilot cyclone plant. A suggestion has been made by the Mines Department Chief Metallurgist that the final degritting be done by a 30 m.m. cyclone assembly installed in a plant layout which provides for replacement by hydroseparators if the cyclones prove unsatisfactory. The following presentation of the pro's and con's of hydroseparators versus 30 m.m. cyclones is given to help clarify the situation and show what is at stake.

Hydroseparators

10. These separate out the fine grit by settlement in water. Whilst there is no material difference in specific gravity between kaolin and quartz grit to promote a differential settling rate, separation by water settlement is facilitated by the difference in shape - the kaolin particles are lamellar and hence slower settling than the granular grit.

This simple operating principle plus their simple mechanical construction make for reliability of operation, freedom from blockage upsets, minimum maintenance, negligible (and that unskilled) attention and low power consumption. Moreover, a hydroseparator is a flexible device able to absorb into its large body of pulp considerable variations in feed volume and composition, with no immediate effect on the quality of its degritted kaolin overflow product, leaving appreciable time for corrective action to be taken before vitiation of the stored sale kaolin slip can occur. All of this makes them particularly suited to the isolation of S.M.C., and the limitation the small size of the plant places on the size of its technical Staff.

Ballarat Clay Co. switched to hydroseparators after failure to obtain trouble free and satisfactory degritting with 3 inch cyclones. (The Mines Department was also unsuccessful with a 3 inch cyclone). The Ballarat separators give little operating trouble, and are consistently making specification quality degritted kaolin slip appreciably lower in water content than that produced by the laboratory's 30 m.m. test cyclone-point of some importance in its bearing on finishing section costs.

11. The price to be paid for this reliability of operation and security of quality of the sale kaolin is the much higher capital cost of hydroseparator equipment as compared with a cyclone unit. Offsetting this is their elimination of the operator which the cyclones would require - saving of at least £18 weekly, or £936 annually. This saving has a substantial capital equivalent. Invested annually in a sinking fund earning 5 per cent the £936 grows to £30,950 over a 20 year plant life, or several times the added cost of the hydroseparator installation or, put another way, the impact on production cost is the same whether a 2/- charge arise from the £936 annual wage of one man, or the amortization and dividend charge on £7,500 of additional capital. The important thing is the impact on the quality of the sale kaolin. The wage charge is certain, immediate and continuing; the capital charge is more flexible for the dividend element in it is not obligatory and can be postponed.

12. Cyclones separate minerals differing only slightly in specific gravity by applying centrifugal force to magnify that slight difference into an effective figure. The cyclone size necessary for separation of the kaolin of the extremely fine particle size demanded for paper use is 30 millimeters, or 1.18 inches in diameter, with an overflow (degritted kaolin slip) aperture about 7/32 inch diameter and a fine grit reject aperture about 3/16 inch diameter. Such a cyclone is made of moulded rubber and has a feed capacity of 3 to 4 gallons of thin slurry per minute, so that 20 to 25 of them would be required for the S.M.C. plant. In the hands of technicians the 30 m.m. cyclone was shown by the research investigation results to be capable of producing degritted kaolin slip of satisfactory quality. Fundamental reasons prevent cyclones of larger diameter from separating kaolin down to the required fineness and so exploratory laboratory tests with a 3 inch size, which would be free of the 30 m.m. unit's susceptibility to blockage, were unsuccessful.

The 30 m.m. cyclone is a commercial scale unit notwithstanding its miniature dimensions. The capital and installation cost of a cyclone assembly for the final degritting operation would be only a fraction of the cost of equivalent hydroseparator equipment. Power consumption would not be much greater.

13. Whilst the virtues are substantial, so are the disabilities:

(a) A cyclone degritting assembly would probably require constant skilled operating attention and supervision - a cost burden for a small plant in an isolated locality.

(b) Unless the prior removal of potential spigot blockage material from their feed - fibrous vegetable trash and over-size grit - is 100 per cent effective, blockage of the tiny grit reject spigots will occur without warning, instantly diverting excess grit into the sale kaolin overflow. Such efficiency in feed pre-screening is neither easy to achieve nor reliable in operation, so that the tiny apertures are a major obstacle to the use of these cyclones for commercial mineral dressing.

(c) Variation in other feed conditions also instantly affects cyclone performance, vitiating the kaolin overflow. Always it is the quality of the sale kaolin that suffers deterioration.

(d) The cyclones produce an appreciably more dilute kaolin over-flow product than do hydroseparators, correspondingly increasing the water elimination task of the Finishing Section.

(e) To date no commercial application of 30 m.m. cyclones has been uncovered. All U.S. kaolin producers depend on some form of water settlement device for the final degritting operation, though some share the task with mechanical centrifuges.

(f) Cyclone diameter is normally determined solely by the mesh of separation. But more than 2 or 3 per cent of clay raises viscosity sufficiently to coarsen the overflow so that the presence of 75 per cent of kaolin in the feed probably means that the 30 m.m. cyclone is the largest that can be used to separate kaolin finer than 30 microns.

14. Perhaps the situation can be summed up most simply by recalling a thought provoking Dorr Company advertisement of some 30 years ago - "Nothing can cost so much as plant that costs too little". It is my conviction that in the scale of importance, security of quality of the sale kaolin and dependability in operation both rank ahead of the additional capital outlay involved in hydroseparators. Each occasion of cyclone trouble would mean a batch of degritted kaolin rendered valueless and destined for the residue dump, or alternatively an expression of dissatisfaction or worse from A.P.P.M.

B. The Finishing Section

15. From its storage tank of degritted kaolin slip the Finishing Section operating 24 hours daily 5 days a week, will draw off slip and process it to the brightness - i.e. color - required by the specifications, and to the form required by the sale contract. The section is housed in an extension of the Degritting Section building at S.M.C. the most convenient position from the operational point of view. It discards only water - it cannot discard any other contamination, it corrects color deficiency arising only from vegetable matter - it cannot remove iron staining.

16. These processing operations consist of -

(a) Chlorine bleaching of all or selected batches of slip to correct deficiencies in kaolin brightness.

(b) Heating and flocculation of the bleached kaolin slip to facilitate filtration.

(c) Filtration to remove about half the water content of the slip so reducing the kaolin to the form of a viscous filter cake mud still containing about 46 per cent of water.

(d) Elimination of this remaining water by drying.

(e) Bagging and storing the dried kaolin ready for transport.

17. If, at some future date, the sale Kaolin has to be split into two size fractions, "Coating Clay" consisting of only the very finest size fractions, and "Filler Clay" consisting of the residue of coarser fractions, then an additional

operation, centrifuging, would have to be included. This equipment and operation would precede 16 (a), and its inclusion would necessitate considerable rearrangement of all of the final processing operations listed above. Thus the bleaching system would have to be reorganized into two separate circuits. A division would also have to be made in the filtering, drying and bagging sections, and duplicate equipment installed to keep the two size fractions apart. There is an alternative that could eliminate the centrifuge and all the above consequential rearrangements. Reported from U.S.A. is the development of a specialized grinding technique for reduction of all of the undifferentiated kaolin to coating clay fineness.

Other General Aspects of the Treatment Plant

18. Provided the risk to the quality of the sale kaolin is fully appreciated and adequately guarded against, the new plant starting up problems could be lightened by temporary postponement of bleaching. The objective is to relieve the technical Staff from the full time attention that initiation of bleaching will demand, and leave it free for the establishment of the circuit conditions necessary for the separation of specification quality kaolin and for coping with the starting up problems common to new plants. The consequential saving of £1 or so in bleach chemicals per ton of kaolin is of secondary importance at this stage.

19. In preparation for this postponement, initial quarry operations would need to be confined under laboratory sampling guidance, to stain free areas so that a separate stockpile of this special raw material could be provided at the plant prior to the start up. This selective mining might delay the development of a tidy economical quarry, but it will give the laboratory technician an opportunity to establish routine quarry face sampling, and sample evaluation before he becomes immersed in plant problems.

20. During the currency of this no-bleach operation the degritted kaolin slip stored in the Finishing Section in flowsheet item No. 20 will at first go direct to the heating and flocculating tanks which supply the filters. Next step will be to route the degritted kaolin slip through the batch agitators of the bleach system, omitting the bleach chemicals with their associated control problems, until this circuit is satisfactorily established. For the final step - the initiation of bleaching - the Company should seek the laon of an experienced chlorine technician from I.C.A.N.Z. Ltd., to help establish the bleaching routine and to instruct the Staff on the safe handling of chlorine gas.

Treatment Plant Capacity

21. The normal procedure when designing mineral dressing plant is to provide a moderate reserve of unused capacity, usually by specifying oversize equipment. Here no material over-capacity has been built into the degritting section for increased production could be obtained simply by increasing running time from the designed 40 toward the possible maximum of 168 hours weekly. The finishing section could similarly increase from 5 to 7 day running, improving output by 80 tons weekly, but for any increase beyond this, operating experience will provide a better answer than present opinion. The finishing section equipment units herein specified are considered to have sufficient capacity for the daily 40 ton kaolin output, but their maximum capacity is not known and will be revealed only by actual operating data. Where this shows inability to meet the demand, the deficiency will have to be made up by installation of additional units.

Corrosion in the Plant

22. The raw material has a fairly low pH value around 4, and hence is somewhat corrosive. However, this kaolin acidity is easily neutralized, so that after addition of 10 lbs. sodium silicate per ton of raw feed at the beginning of treatment the slurry is practically neutral, its pH having risen from 4 to 6.8-7.3 (p.22, 26, O.D.I.). At this level there may be slight corrosion at the air-water line in steel tanks but nothing worse. Routine testing of the Dorr classifier overflow with a suitable indicator solution is a simple way of ensuring that it remains faintly alkaline, and is thus receiving sufficient sodium silicate for the required degree of dispersion. This faint protective alkalinity carries through to the finishing section where it will be destroyed by the chlorine bleach and the aluminium sulphate used for flocculation. Special anti-corrosion materials and precautions, starting at the bleach, have thereafter to be used to prevent iron going into the solution and permanently staining the kaolin. This is the procedure at Ballarat where no precautions in the way of costly stainless steel equipment, or even protective coatings, are taken until the hot slip is flocculated prior to filtration. Their Dorr classifier overflow is maintained at pH 8.3, or a faint pink with phenolphthalein indicator. This higher alkalinity no doubt means a little excess sodium silicate is being added, but it also guarantees a lot of protection against corrosion and non staining.

The Dust Problem

23. The Company will have a dust problem to contend with. This is prominent even in the small Ballarat plant, and is known to be of very large proportions in the big U.S.A. plants. Because of its extremely fine grain size dry kaolin readily becomes air borne, and so the stockpile will generate much dust during the dry months. Dry raw material on the floor of the roofed area will be churned up by the front end loader throughout the year into a deep layer of aerated fluffy powder, providing another fertile dust source. When dried, the sale kaolin will be a particularly dusty product. Because of its close proximity to the stockpile and loading area the degritting section needs to be totally enclosed in a building, including the two degritting hydroseparators, to protect equipment and operators, and prevent contamination of the degritted kaolin slip. If the hydroseparators are left in the open in order to economise on the building, they will need top cover protection against wind blown grit and also from vegetable trash from the surrounding scrub. Similarly, with the bleaching tanks which will necessarily be in the open because of their chlorine health hazard. All other equipment will be storing or handling the sale kaolin in one form or another, and so will have to be securely housed against wind blown contamination.

The Sale Product Specifications

24. At the date of this report the sale product will be required to meet the following minimum specifications:

Brightness:	A brightness 77 as determined by A.S.T.M. Directional Reflectance Method designation E.97-55.
Sizing:	Plus 200 mesh - Nil. Plus 30 microns - less than 2 per cent. The 1 micron requirement is not specified, but the natural content of 25 to 40 per cent of this size kaolin has been stated to be satisfactory.

Grit:	Less than 3 per cent of quartz grit, as determined by chemical method.
Ignition Loss:	Not specified, but of the order of 12 per cent.
Redispersibility:	If delivered in dry form, the sale kaolin must be redispersible.

Partial Estimate of Treatment Costs at S.M.C.

For the Weekly Production of 200 Tons Dry Sale Kaolin in Bin at S.M.C.

25. The following preliminary estimate takes raw material from the plant stockpile and delivers dry sale kaolin into the plant bulk storage bin at S.M.C. ready for bagging for transport. Wages are as fixed by the Tasmanian Clay Products Wages Board, which covers all operators except boiler attendant and maintenance men, and power charges are those ruling in August 1962. No provision is made for fire insurance on the plant buildings for the nature of the industry makes fire danger negligible. Moreover, their isolation and location in scrub country could mean a prohibitive premium. Finally, the capital cost figure £130,000 used as a basis for the plant amortization charge is necessarily a guess unsupported by a single quotation. With so much of the plant design incomplete there is no justification for commencing quotation enquiries at this stage.

All figures are bare, without any addition for contingencies or forgotten items of expenditure. The allowance for contingencies is best postponed until all other bare production cost estimates have been assembled. It can then be added by one person, at the end where the whole of it can be seen.

Power

26. Power has been estimated for this report by the H.E.C. on the basis of the following information given to it, at £534 per 13 week quarter, plus a further £29 per quarter for lighting.

8 motors totalling 84 name plate H.P., running 8 hours daily, 40 hours per week.
14 motors totalling 48 name plate H.P., running 24 hours daily, 120 hours per week.
1 motor (water supply) totalling 40 name plate H.P., running 8 hours nightly, 40 hours per week.

Utilization factor 70 per cent., and a minimum demand of 92.4 B.H.P. brings the total to £534 per quarter, or £41 per week.

Lighting say 1 K.W. for 60 hours weekly, total 780K.W.H., or £29 per quarter, £2.4s.0d. per week.

Front End Loader

27. This has been estimated separately, being particularly vulnerable to driver abuse and hence liable to be short lived with an expensive maintenance record. Earth moving equipment units are notable for their habit of exceeding estimates of operating costs, maintenance expenditure rising each year. A sinking fund charge to replace it in 5 years looks advisable against the 20 years life for the treatment plant. Weekly charges are estimated as follows

Amortization of say £2,500 over 5 years at 5% = £450 annually =	£9	weekly
Tyre renewals, at 1 complete set of tyres & tubes £196, life 2 years =	£2	"
Fuel =	£3	"
Mechanical maintenance over first year - subsequent years more =	£2	"
Total	£16	"

Regarding the last item, a local brick company has spent this rate over the first 9 months life of a new heavy duty $\frac{7}{8}$ yard front end loader. Tyres also suffered badly from sharp ends of steel sections left around by the oxy-acetylene crew. Such carelessness and similar untidiness on the loader floor would be very costly.

28. Itemised Estimate of Weekly Expenditure.

		Per Week	Ton Sale Kaolin
Wages:			
Degritting Section	1 front end loader driver.	£18: 1: -	
	1 plant operator (and lunch hour relief driver.)	17:13: -	
Finishing Section	3 filter operators.	53: -: -	
	3 dryer operators.	53: -: -	
	3 boiler attendants.	51: 3: -	
	1 general hand, dayshift only.	17:10: -	
General	Leading hand allowance, 1 man per shift.	3: -: -	
	1 maintenance fitter.	19:16: -	
	1 maintenance electrician.	20: -: -	
	Overtime allowance for maintenance alternate Saturdays.	7: -: -	
		£260: 3: -	1: 6: -
Stores and Spares.			
Sodium silicate 3 tons per week at £30 per ton at S.M.C.		£90: -: -	
Aluminium sulphate 3 tons per week at £40 per ton at S.M.C.		120: -: -	
Chlorine gas 1 ton per week at £122 per ton at S.M.C.		122: -: -	
Sodium bi-sulphite 1 ton per week at £80 per ton at S.M.C.		80: -: -	
Coal for pulp heating and steam drying 65 tons at £6 per ton at S.M.C.		390: -: -	
Agitator liners 2 tons at £80 per ton M.S. plate. Life 6 months.		6:10: -	
Agitator impeller covers, 8 sq. ft. Linatex, life 2 weeks.		4: -: -	
Screen cloth for trash removal wear allowance.		2:10: -	
Sundry minor stores and spares, pump liners, etc. say		10: -: -	
Front end loader fuel tyres and spares.		7: -: -	
		£832: -: -	4: 3: 6
Power and Lighting.		£43: 4: -	-: 4: 6
Local Office Overheads.			
Plant superintendent.		£50: -: -	
Laboratory technician.		20: -: -	
Accountant.		30: -: -	
Typist.		12: -: -	
Office 'phone and supplies.		10: -: -	
Office car.		9: -: -	
Public holidays, annual and sick leave, total 4½ weeks annually per employee or £1,070.		20:10: -	
Workers compensation premium at £1:18: - per £100 of wages = £264 annually.		5: -: -	
Payroll Tax at 2½ per cent on excess of wages over £10,400 annually.		2: -: -	
Maintenance of plant buildings and surroundings - say		5: -: -	
Amortization of say a £130,000 plant over 20 years at 5 per cent requiring an annual charge of £3,930.		76: -: -	
Amortization of a £2,500 f.e. loader over 5 years at 5 per cent = annual charge of £450.		9: -: -	
		£248:10: -	1: 5: -
	Grand Total	£1,384: -: -	6:19: -

PART II.

RAW MATERIALS SUPPLY AND RESIDUE DISPOSAL.

Raw Kaolin Material.

29. The plant will consume 120 tons dry weight of raw kaolin feed per day, 600 tons per week, 30,000 tons per year. For the quarry staff and the plant front end loader driver, who deals in cubic yards, the terms dry tons and cubic yards, broken volume can be taken as interchangeable irrespective of moisture content: 1 cubic yard of raw material, broken volume, contains 1 ton dry weight. This figure was obtained from the laboratory measurement of the volume of 1 ton gross weight of loose raw material containing 15 per cent moisture - i.e. 23 cubic feet. It is also the figure used by Ballarat.

30. Since any clay pit becomes unworkable in winter, the excavation of the 30,000 cubic yards must be completed during the dry summer - autumn months, the surplus over current consumption being stored on a stockpile adjacent to the plant. The stockpile site will require to be dozed to slope gently away from the plant and the surface then well stabilized with cement. The area of the paddock will be determined by the duration of the quarrying period. Thus quarry operation for 5 months annually would mean about 18,000 cubic yards to be stockpiled for the remaining 7 months, ramped to a height of 12-15 feet on a paddock about 70 yards square. This ramping by the trucks will so consolidate the top surface of the dump that no rain will penetrate it whilst rain falling on the sides will drain off with only a few feet penetration. A ring drain will be required to carry away this run-off and prevent it accumulating at the working face.

31. To produce a finished product of uniform quality any mineral dressing plant requires raw feed of reasonably uniform composition. Quarry practice will assist in achieving this by drawing from a number of working faces to average out variations. And to facilitate the recommended by-passing of bleaching during initial operations, (paras. 18-20) faces showing a material degree of vegetable staining will need to be left untouched for the time being. Iron stained areas will need to be identified, isolated and avoided altogether as chlorine will not remove this discoloration. The daily quarry schedule may therefore need to be guided by regular sampling of the working faces with plant laboratory evaluation of the samples for quality. With draw-off from the various faces proportioned on the sampling results, variations will be smoothed out into a raw feed mixture of uniform quality. A standard sampling procedure will need to be developed along with a rapid laboratory evaluation technique.

32. The front end loader driver will soon evolve a satisfactory routine for feeding the degritting section from the stockpile during the winter months, just as the Ballarat driver has. Rain penetration of his working face is reduced by maintaining the face as nearly vertical as possible and by working the lee side of the dump during directional rain. The interior of the dump being at summer dryness rain slurrying can thus be confined mainly to the toe of the working face. The conveyor belt trouble (build up of clay accretions on the return idlers) arising from this over wet feed is minimised by mixing in some of the reserve of dry raw material. On some days the plant will have to be supplied entirely from

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the dry reserve (para. 33). This stock under cover is replenished in dry weather using a heavy duty dozer to push in from the stockpile. This task is much too heavy for most front end loader-dozer combinations, which are primarily loaders and can double for light duty dozers only for tidying up the working face.

33. In addition to the open air stockpile there is need for 600 to 1,000 tons of dry raw material stored under a roofed area connecting the stockpile to the degritting section, to ensure the feed supply to the plant when very wet weather makes the dump unworkable. It would have a single span arched lattice girder roof free of horizontal rafters that would restrict the movement of the dozer stacking the raw material to a height of 12 feet or more. The weather side could be walled with strengthening of the bottom 4 feet or so to confine the toe and increase storage. This roofed area needs to have a concrete floor, otherwise the loader wheels soon cut up a dirt floor and so greatly increase the depth of aerated dust layer always present on the floor.

Water.

34. The estimated water requirements for the various tasks are:-

(a) Treatment (para. 35.). 2,620 to 3,160 Gal. per hour 7 hours daily.

(b) Residue disposal (para. 40-41.). 6,700 to 2,500 Gal. per hour 7 hours daily.

(c) Cleaning, general. 500 Gal. per hour 7 hours daily.

(d) Steam Generation (para. 93-95.). 660 Gal. per hour 24 hours daily.

Total, including some surplus, 100,000 gallons per day if the residue be pumped to disposal or 250,000 gallons if disposal be by gravity launder.

35. These estimates are derived as follows:

(a) The 17 tons raw feed per hour will bring in about 4.3 tons of natural moisture (20 %) and this will need to be raised to 16 tons per hour by addition of 11.7 tons of new water during treatment. But if degritting difficulties or methods result in the degritted kaolin slip being lower in solids (say 31 %) than the 36 per cent of the design estimate, the hourly addition of new water might rise to 14 tons, increasing the above 2,620 G.P.H. to 3,160. As will be seen later, strict control of this added water is essential.

(b) Disposal of the waste residue is detailed in paras. 40 and 41.

(c) Provision for dust suppression and cleaning the plant floors.

(d) This is boiler feed water based on estimated steam requirements for pulp heating to improve filtration and for drying the filter cake. See paras. 93-95 below.

Practically all of this water is consumed on day shift, but a reservoir would permit night pumping from the Ringarooma River at an economy power rate.

36. The river water is visibly brownish in winter and is said to be more highly colored during the reduced flow rate of summer. This vegetable coloring is accompanied by an appreciable amount of green algal growth in suspension. Bleaching probably destroys the color, but with intermittent bleaching a possibility, the color (and suspensiods) in 2 tons of this water will remain with each ton of dry sale kaolin and could reduce its brightness. The suspensoids will accumulate in the boiler from the 16,000 gallons daily converted to steam, but their probable effect therein is a matter for the opinion of a steam engineer. The remedy

could be a simple sand filtration plant though the installation of this would remain in abeyance until an investigation of the water contamination problem has been made. The amount of filtered water required would be the sum of (a) and (d) para. 34, or 40,000 gallons per day.

37. To facilitate the application of strict control over the amount of new water added to the raw material entering the Degritting Section, the whole of this new water must originate from a constant head tank. This tank must be of sufficient capacity to supply the plant for several hours to provide against accidental failure of its own supply which would either come in direct from the river pump or by gravity from a reservoir. It requires to be positioned at a sufficient elevation to feed into the Degritting Section by gravity.

38. The purpose of this constant head tank is to ensure that the control valves and rotameters metering exact amounts of water to the various equipment units operate under a fixed head. The delivery main from this constant head tank therefore includes a master valve which alone is opened at the start and closed at the end of each day shift, leaving the preset individual valves within the plant untouched. This constant head tank is an essential part of the water supply system and will be required irrespective of whether it handles filtered or unfiltered water.

39. The river pump and pipe line will thus be required to deliver either 100,000 or 250,000 gallons over 8 hours pumping each night shift, depending on the method used to get rid of the degritting section residue (paras. 40,41). A hilltop reservoir scooped out to hold say 500,000 gallons - a pond 100 feet square by 8 feet deep - would protect the plant's supply for two to five days. The sites, distances and levels and the method of getting rid of the plant residue will all have to be determined before the water supply system can be designed.

Disposal of Degritting Section Residue.

Laundering by Gravity.

40. The residue will be produced at the hourly rate of 11.3 tons of quartz grit, angular and of about $\frac{3}{8}$ inch maximum size but mostly much finer, accompanied by 5.7 tons of water, with totals of 80 and 40 tons each day shift. The simplest method of carrying it away would be by gravity flow in a launder leading to a nearby gully. For this the residue would have to be diluted with sufficient new water to ensure transport. On the assumption that 10 tons of water would be required to move 1 ton of grit along the launder, the additional water would amount to 760 tons spread over 7 operating hours or 25,000 gallons per hour. At a flow rate of 7 feet per second induced by a fall of $1\frac{1}{2}$ inches per foot, a launder 9 inches wide by 8 inches deep would run about 3 inches deep. It would need to be lined with renewable boards or a skin of concrete to cope with erosion by the sharp grit. Given a suitable adjacent gully and a river water supply operating against only a moderate delivery head, the trouble free reliable disposal by launder would be more attractive than pumping.

Pumping.

41. Disposal by sand or gravel pump, whether downhill into a gully or up onto a dump above pump level, will also require dilution of the daily 80 tons of residue grit and accompanying 40 tons water with an additional 30 tons new water, or 6,720 gallons per hour to dilute the residue pulp to 50 per cent solids. This

would give a residue pulp volume of 1,411 cubic feet per hour over 7 hours daily or 0.392 cubic feet per second. At the 8 feet per second velocity necessary to keep the coarse solids in suspension, this means a 3 inch discharge pipe line. In terms of gallons, the 1,411 cubic feet becomes 8,780 gallons per hour or 146 gallons per minute, which could be handled by a 3 inch (discharge size) rubber lined Warman sand pump. Motor size would depend on the discharge head against the pump. If the river supply pump has to operate against a considerable delivery head pumping residue might be cheaper than pumping the extra water for launder disposal. Again, the absence of a suitable gully would necessitate the use of a pump to elevate the residue onto a dump. And in the rather unlikely event of a market developing for the residue, the pump could build a stockpile of it at any convenient position.

PART III.

TREATMENT PLANT EQUIPMENT IN DETAIL.

Flowsheet Item 1. Stockpile of raw feed at plant. See Part II.

42. Item 2. Roofed area of dry raw feed.

This is a concrete floor, say 25 yards square, covered by a single span roof at a sufficient height to allow under-cover storage of up to 1,000 cubic yards of raw material and still leave a travelling lane for the loader between stockpile and feed hopper. This dry stock would supply the Degritting Section for 8 shifts should abnormally wet weather render the stockpile unworkable. To ensure free escape for loader exhaust fumes, the building has no side walls.

43. Item 3. Front End Loader.

There does not seem to be any practical alternative to this type of loader for feeding the Degritting Section from the stockpile. The concrete floor will be ideal for a loader running on rubber wheels, and this type is more nimble and cheaper to operate than a track unit. The only threat to the tyres will be carelessness in leaving razor edged off-cuts of angle and other steel sections or jagged quartz rocks lying on the loader working area, to become hidden in the dust. A more expensive heavy duty machine is likely to prove cheaper in the long run than a lighter unit, for it will be better able to withstand driver abuse. Also, it must be heavy enough to do light bull-dozing for tidying up the working face, either with an attachable dozer blade, though preferably with a permanently attached blade.

44. As regards the make and size of loader, talks with users of various makes working under similar conditions could pay a dividend. A local brick company has had disappointingly high maintenance costs with heavy duty $\frac{7}{8}$ yard loader of reputable make, apparently not all attributable to driver abuse. The Ballarat Clay Company uses a $\frac{3}{4}$ yard Cranvel. The above two makes have the same disability - a separate dozer blade which requires time and the assistance of extra labour to secure in place. This is an open invitation to avoid its use and consequently to clean up the working face toe by pushing and side swiping with the bucket in ways the machine is not built to cope with. The I.H. Drott "4 in 1" loader mentioned below (para. 45) because of its special bucket design appears to merit close investigation, especially as it also has a permanently attached scraper-dozer blade. Working 7 hours daily to deliver 120 cubic yards raw feed to the Degritting Section with a 25 to 40 yard run between stockpile and plant, a $\frac{3}{4}$ yard loader would need to make the round trip in $2\frac{1}{2}$ minutes, non stop throughout the shift, with no time for tidying up the working face. This looks like an impracticably tight schedule. A 1 cubic yard loader would need only 17 trips per hour as against 24 for the smaller unit, and so looks the better size. However, local suppliers, like Wm. Adams or I.S.A.S., could visit the plant and give experienced advice. The lift - i.e., the height at which the loader must be able to discharge its load is not presently known. The finished design of the loading hopper (Item 4) and the ramp leading up to it will determine th lift. It will be about 8 feet above ground level.

45. The design of the bucket is of vital importance. Wet sticky clay is probably the most troublesome and unco-operative raw material any mineral dressing

plant has to handle. The wedge shaped buckets normally fitted to loaders are useless for wet clay. Compacted clay soon fills the four corners to considerably reduce working volume. It adhers so tightly as to be resistant even to manual chipping. Meantime it tempts the driver to try and dislodge it by dropping the bucket repeatedly onto the ground. Designed to discharge by simply over-turning the bucket, bottom suction further prevents a clean discharge. Required is a bucket with vertical sides and rounded corners, discharging through a bottom trip-door. This is the shape evolved by experience by the Ballarat Clay Company and specially made for its loader. It appears to be satisfactory. The I.H. Drott "4 in 1" loader has a clam-shell bucket that appears to be an even better design. Hinged at the top, it opens wide to let the whole load drop out cleanly. Moreover, one half of the clam-shell serves as a permanently attached blade for cleaning up either by dozing or scraping. The 1 cubic yard size runs on rubber tyre wheels, but all other sizes appear to be track mounted. Finally, since the raw material is acid and full of sharp grit, wear on the bucket digging edge will probably require a hard alloy coating.

Item 4. Feed Hopper.

46. A feed hopper of 3 to 4 cubic yard capacity, to sit on the conveyor belt, is practicable and when filled would allow the loader driver 10 to 15 minutes freedom to reshape his stockpile face. Although necessarily wedge shaped, the slow moving conveyor belt beneath will drag the feed out through the open bottom and front end and so prevent hang-up. It can be of steel or of wood lined with M.S. plate, with sides not flatter than 60° from the horizontal, and with rounded corners. It would be hung from a frame carried on fore and aft columns. This frame needs to be rugged, for the driver will often enough be tempted to drop his bucket on it to help empty it or to dislodge a hang-up in the hopper. The floor is ramped up to it to reduce the lift of the loader.

Item 5. Belt Conveyor.

47. This is a 24 inch wide belt, troughed over its whole length on standard idlers except under the feed hopper, where it is supported on a close spaced flat rollers. It drags raw material out of the feed hopper at the required uniform rate of 17 cubic yards (or tons) per hour and elevates it into the slurrying agitators. It will be about 60 feet long between head and tail pulley centers and inclined at 18° from the horizontal. Carrying a ribbon of feed 18 inches wide by 6 inches deep, it would deliver the 17 cubic yards per hour with a belt speed of 12 feet or so per minute, and at this sort of speed would be its own feeder from the hopper. The motor-conveyor drive pulley ratio will be chosen to give a belt speed of say 20 feet per minute at mid setting on a 4 : 1 ratio gear motor of probably 5 H.P., so that a speed range of 10 to 40 feet per minute will instantly be obtainable. This variable speed provision is essential for maintaining the hourly 17 ton feed input in the face of daily variations in the physical condition of the raw material.

The discharge from this conveyor drops vertically into the first of the two slurrying agitators through a vertical steel chute with about a 24" x 15" top opening. The sides of the chute flare outwards slightly, giving it a bottom appreciably larger than the top, to discourage the build up of side accretions in the face of splashing from within the agitator. The bottom of the chute is a fairly neat fit in a hole in the top corner of the agitator to prevent escape of splash.

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Item 6. Twin Slurrying Agitators.

48. These two agitators, operating in series, reduce the raw feed to a highly dispersed slightly alkaline slurry containing 60 per cent solids.

Tank.

A single rectangular tank of $\frac{1}{2}$ inch M.S. plate, 10 feet long by 5 feet wide by 5 feet deep vertical sides, divided by a central partition into two compartments each 5 feet square. These two compartments have pyramidal bottoms about 3 feet deep. All interior vertical surfaces are protected by renewable M.S. liner plates, 2'6" wide by 4'6" long, held in place by joint covering vertical bars 2" by 1" bolted to the tank shell by through bolts of ball mill liner type with countersunk heads inside and nuts outside. Hemp grummets under the nut washers prevent leakage. The pyramid faces are similarly lined with triangular liner plates secured by corner strips. The tank will be on columns supporting it well above floor level and the two compartments will have removable splash tight split covers. The Ballarat agitators are cylindrical, apparently having been designed to make use of available idle tanks. The staff is understood to prefer a rectangular shape, but this point (and later the completed S.M.C. design) should be checked with Ballarat staff.

49. Discharge Ports.

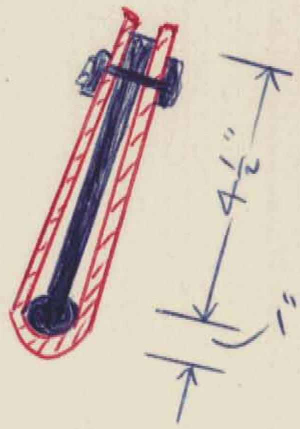
No. 1 or feed compartment discharges into No. 2 through a coarse grit relief port 16" long by 9" deep in the partition wall at impeller level - i.e., extending upwards for 9 inches above pyramid level. No. 2 has a similarly positioned relief port 6 inches diameter which discharges through a 6 inch rubber pinch valve and 6 or 7 inch pipe into the feeder launder of Dorr classifier Item 9. No. 1 agitator also overflows into No. 2 through a 12" by 9" deep port in the partition wall, the bottom of this opening being 2'6" above pyramid level, so that the slurry level in the two agitators will be 2'9" to 3' above pyramid level. The finished slurry will flow through the combined discharge ports with a velocity of 3 or 4 inches per second. The upper or overflow discharge port of No. 2 agitator delivers into a 12" wide by 9" deep splash proofed launder terminating in the feed launder of the Dorr classifier. Both overflow ports of the two agitators have vertical baffle plates bolted to the tank walls at 45° angles to guide the swirling slurry into the ports.

50. Scuttling Doors.

The two pyramid bottoms terminate in hinged discharge doors of the type formerly used for emptying cyanide sand percolation tanks, closing openings about 9 inches square, or larger, for dumping the contents of both agitators after bogging due to power failure. For design details see Rand Metallurgical Practice or similar early cyanidation text books. The scuttled contents fall into the degritting section residue disposal launder which runs longitudinally under the tank.

51. Impellers.

These are 24 inches diameter, one per compartment. The impression was gained that the Ballarat impellers are rotated to <u>lift</u> the slurry instead of giving it the down-throw of normal agitators. This up lift would help to keep the conveyor discharge chute running freely, as delivery would be onto a down current. But it would increase thrust bearing pressure. Ballarat should be



consulted on the point. Four blades, 12 inches long from center of shaft to blade tips, made of $\frac{3}{8}$ M.S. plate $4\frac{1}{2}$ inches wide are welded to a boss at 45° angles, and a 1" diameter M.S. rod is welded along the bottom edge of each blade. A renewable wearing cover of $\frac{5}{8}$" thick linatex rubber is wrapped round each blade and secured by two M.S. strips $1\frac{1}{2}$" wide by 8" long by $\frac{1}{4}$" thick, bolted through the top of each blade. The cut edges of the linatex project $\frac{3}{4}$" above the top edge of the impeller blade and $\frac{3}{4}$" beyond the end, and at the other end butt hard against the boss. The boss has an internal taper and slot to fit over a fixed key on the tapered end of the vertical impeller shaft, and is secured there-on by a heavy bronze cap with a female thread matching a reverse male thread on a straight extension of the shaft below the taper section. This cap has no spanner flats - they would not survive - and so is removed by stillson. Bronze appears to last better than steel.

52. Shaft.

The vertical impeller shaft, protected against erosion by a sleeve of steam pipe, is carried in a bearing assembly of the design used for Denver flotation cells, mounted on horizontal beams which leave half the top area of each agitator unobstructed for access with liner replacements. The shaft has a substantial thrust bearing to balance the impeller.

53. Motors.

Each agitator is driven by a totally enclosed 15 H.P., 960 R.P.M. vertical motor. The coarse thick slurry these agitators handle results in quite small variations in impeller dimensions, particularly diamter, exerting a considerable effect on power consumption. Hence the importance of providing a means of experimenting with shaft speed - i.e., impeller peripheral speed - to secure adequate slurrying with minimum power and wear. Ballarat impellers appear to run at a peripheral speed of about 1,800 feet per minute - a high figure in comparison with Denver flotation cell and conditioner speeds, but which experience has no doubt shown to be necessary. The following pairs of motor and impeller shaft sheaves give a range of shaft and impeller peripheral speeds, and they should be provided for experimentation. The same V belts would suit all pairs. They need to be heavy duty belts for this difficult drive.

9 inches and 31 inches p.c.d. for 280 R.P.M. shaft and 1758' peripheral.

8.5 inches and 31.5 inches p.c.d. for 260 R.P.M. shaft and 1633' peripheral.

8 inches and 32 inches p.c.d. for 240 R.P.M. shaft and 1507' peripheral.

7.5 inches and 32.5 inches p.c.d. for 218 R.P.M. shaft and 1370' peripheral.

A first attempt could be made with the 1633 feet speed.

54. Feed Components and Agitation Time.

Raw feed: 17 cubic yards = 17 dry tons per hour bringing 4.3 tons natural moisture.

New Water: 6.2 tons = 1390 gallons per hour.

Dispersant: 0.08 tons = 170 gallons per hour 10 per cent solution of N84 grade sodium silicate.

Totals: 28.3 tons of slurry 60 per cent solids, with a volume of 638 cubic feet per hour or 10.6 cubic feet per minute.

The slurry is alkaline with a pH around 8 and is in a dispersed condition.

Rigid control over the new water addition is essential. All water carelessly added in the degritting section inexcess of the design flowsheet figures, will end up in the finishing section kaolin slip, from which it will have to be eliminated at added cost. The new water additón to No. 1 agitator will therefore come through a supply pipe of generous diameter from a constant head tank of (probably) filtered water, and the quantity will be metered through a rotameter by a control valve adjacent thereto. Once set, this valve would not be touched except for adjustment of the slurry pulp density, a master valve on the constant head tank side of the rotameter being used to turn on and shut off water flow each day.

As described in para. 57, the dispersant addition will be similarly controlled. The above figure 0.08 tons silicate in a 10 per cent solution is based on the quantity used in the research investigation. It secured adequate dispersion and resulted in a slurry that was practically neutral at pH 6.8 to 7.3. In the plant, the addition rate will be adjusted to give a pH of 8 to 8.4 as insurance that the completeness of dispersion will not be in doubt.

The agitation time in each tank is derived as follows:

$$\frac{\text{Slurry volume in each agitator at 3' pulp depth}}{\text{Volume of feed components per minute}} = \frac{72.6 \text{ cubic feet}}{10.6 \text{ cubic feet}} = 7 \text{ minutes.}$$

The combined total of 14 minutes violent agitation should completely disintegrate and slurry the raw feed material, and scrub the kaolin off the quartz grit. (O.D.1. p.8). If not, the overflow level can be raised to increase the agitation time a little.

55. Operation.

The impellers are started with tanks empty, then dispersant, water and raw feed commenced. The bottom discharge port of No. 2 agitator is kept closed until overflow from the top port commences. Then the squeeze valve of the bottom part is opened only just enough to keep it discharging the oversize grit. At the end of the shift both agitators are run empty through No. 2 bottom port before stopping the impellers.

Item 7. Dispersant Mixing Tanks.

56. Sodium silicate of N.84 grade is a highly viscous liquid, S.G. 1.42, containing 9 per cent Na_2O, 29.6 per cent SiO_2, 61.4 per cent water. The weekly consumption of 3 tons thus means freight is paid weekly on nearly 2 tons of water from Sydney to S.M.C. This could be saved by buying the dry silicate glass and breaking it down with steam to N.84 grade at S.M.C. The N.84 grade dissolves satisfactorily though slowly in cold water. The commercial pack is a 44 gallon drum containing about 620 lbs. and the daily requirement is 2 drums.

Two dissolving tanks are recommended, one storing the day's supply, the other dissolving the next day's needs. Each tank is 7' square by 6' deep and is set on concrete piers 2' above floor level, if steel or wood be used, on the floor if the tanks are concrete. Each is equipped with a 3 or 4 blade 15" diameter down-throw impeller placed about 18" above tank bottom and driven at 250 R.P.M. by a 3 H.P. vertical motor. The impeller can be ship type or consist of 4 M.S. plate arms welded on 45° angles to a boss. A 3' wide by 7' long flat bottom tray of M.S. plate punched with many $\frac{1}{4}$" holes is fixed in the tank 2'6" below the top. Tilted drums of N.84 are allowed to empty slowly into the tray and thence through into about 3' of water beneath. When dissolved, the volume is made up to about the 4' mark to make one day's requirement of 1,200 gallons of 10 per cent solution. A small clean water pump delivers 200 gallons per hour from mixing tank to constant head tank, Item 8, some 20' above.

Item 8. Constant Head Dispersant Feed Tank.

57. A steel tank 3' by 3' by 3' deep, with removable cover, is placed at a sufficient height above No. 1 agitator feed chute for gravity flow to operate a rotameter reliably. This tank holds an hour's supply of dispersant, the excess delivery from the pump below overflowing back to tank, Item 7, through a plastic pipe. Intake from the pump is through a fine screen (bronze) to catch any lint or trash that could block the rotameter from reliably metering 170 gallons per hour through master and control valves rotameter and plastic hose into No. 1 agitator. Creep over the tank top needs to be washed off frequently to prevent accretions. Ballarat should be asked for advice on a rotameter type that will operate reliably on sodium silicate solution.

Item 9. Dorr Straight Classifier.

58. This unit, in one simple operation, roughs out and discards almost 70 per cent of the total grit in the slurry fed to it from Item 6, and overflows a thinned slurry for final degritting by the hydroseparators. Its size has been determined as follows on Dorr Company catalog data:

(a) Slope: for an overflow of minus 60 mesh solids of S.G. 2.7, the slope required is $2\frac{3}{4}$" to 3" vertical per foot horizontal.

(b) Length: at 3" slope a length of 28' is required in order to leave 7' of deck length dry at the sand discharge end for spray wash and drainage of entrained kaolin back into the pool.

(c) Raking Capacity: For F. and H. type classifiers at 20 strokes per minute, the Dorr Co. quotes a safe maximum of 280 short tons of sand per foot width per 24 hours. The plant requirement is only 200 short tons so a single rake 1 foot wide would do as regards raking capacity. But by using a unit 5 or 6 feet wide, the sand bed will be thin enough to ensure that washing and drainage will prevent the loss of entrained kaolin of minus 30 micron range from exceeding the research figure of 2 or 3 per cent. (O.D.I. p.10).

(d) Overflow Capacity: For minus 60 mesh S.G. 2.7 solids and 20 strokes per minute Dorr regards 5.3 short tons overflow solids per square foot of pool area per 24 hours as a safe maximum. At 3 inches slope, the pool area is given as 13.8 square feet per 1 foot width of classifier. The pool area required for the flowsheet overflow figure of 9.5 long tons per hour, or 250 short tons per 24 hours is $\frac{250}{5.3}$ = 47 square feet, and the width to provide this is $\frac{47}{13.8}$ = 3 feet 6 inches. This is the safe minimum width for all ordinary mineral dressing requirements. For kaolin processing it would be better to operate with a less crowded pool so a classifier at least 5' and better 6' wide looks preferable. If the 6' width proves too large, the pool area can easily be reduced but it is less easy to increase a pool that is too small.

Dorr F. Classifiers are made up to 6' wide, type F.H. in 4' and 5' widths and type H. in 3', 4' and 6' widths. All have similar raking mechanisms and can be had in any tank length from 18 to 30 feet. A standard duty machine of normal mild steel construction will suffice. A 6 feet wide D.S.F. or D.S.H. 28 feet long unit (meaning duplex rakes with steel tank and either F. or H. type mechanism) will handle the tank. The 4'6" wide by 26' Dorr used by Ballarat to handle half the S.M.C. throughput, lengthened to give 7' of dry deck for washing, has proved satisfactory. Although of mild steel construction without any protective coating, there is no trouble with corrosion.

59. Required then is a 6' wide by 28' long Dorr D.S.F. or D.S.H. straight classifier, operated at 20 strokes per minute by about a $7\frac{1}{2}$ H.P. motor. Slope 3" vertical per 12" horizontal to give 7' length of dry deck at rake discharge end. One row of new water sprays about half way up the dry section. Normal mild steel construction throughout.

Feed: 17 tons solids plus 11.3 tons water per hour as slurry 60 per cent solids, 638 cubic feet volume, from No. 2 agitator.

Dilution Water: comprising new water spray plus return water from Item 17, total 4.7 tons or 1,050 gallons per hour.

Rake Discharge: 7.5 tons plus 60 mesh quartz grit plus 2.2 tons water per hour delivering into the waste residue launder below the classifier.

Overflow: 9.5 tons minus 60 mesh solids plus 13.8 tons water per hour, forming 626 cubic feet of 40 per cent solids slurry delivery to surge tank, Item 11, through trash removal unit, Item 10.

60. <u>Operation</u>.

The feed slurry has to be diluted considerably to thin down the pool to permit grit coarser than 60 mesh to settle through to the rakes and leave the minus 60 mesh remaining solids to overflow as a thin slurry of 40 per cent solids. Dilution amounts to 4.7 tons added water per hour, some of it being kaolin bearing return water from the scavenging hydro-separator, Item 17 - around 2 tons or 450 gallons per hour. The remaining 2.7 tons or 600 gallons will be fed through the spray pipe measured by a rotameter with the adjusting or control valve placed alongside the classifier. Assuming the amount of return water has settled down to a steady figure, the amount of spray water will be adjusted to maintain the overflow pulp density, <u>measured at regular intervals</u>, at 40 per cent solids. At end of each day a master valve shuts off spray water, which comes from the constant head supply. An alkalinity test is made on the overflow at the same time as the density check. The overflow must show a faint pink at least, with phenolphthalein indicator. The rate of sodium silicate addition to the agitators must be increased if there is no color, (reduced if the color is a deep pink), for the classifier cannot do its designed task unless the pulp in its pool is fully dispersed. This adjustment will be dangerously slow to take effect through the circuit. Hence containers holding a few gallons of dispersant need to be kept handy for dumping into agitators and classifier pool for immediate alkalinity correction. Density is more likely to fall below than rise above the 40 per cent level, and it will be a matter of judgment which of the three water addition points, to the No. 1 agitator, to the scavenging hydroseparator for return water or to the Dorr classifier sprays, can best stand being cut back. Incidentally, a density reading greater than 40 per cent is much more likely to mean inadequate dispersion - insufficient alkalinity rather than insufficient water input.

Item 10. Trash Removal Screen.

61. The raw material, coming from shallow quarry pits, will be contaminated by some - at times possibly considerable amounts of vegetable litter and roots. All this trash will end up in the Dorr classifier overflow from which it must be removed before this product goes to the final degritting units. If ultimately these be 30 m.m. cyclones, removal must be 100 per cent and reliable. For hydroseparator degritters, trash removal need not be so complete for they are not subject to blockage, and a small amount of vegetable trash reporting in their overflow will be removed by screen, Item 19.

Since the amount of this vegetable trash is not known, and so will have to be learned from plant operation, it might be sufficient for a start to let the Dorr classifier discharge onto a 2'6" wide by 6' long stainless steel plate set on a downhill slope of say $1\frac{1}{2}$ inches per foot punched with 1/16 inch diameter holes spaced 1/16 inch apart, diagonal spacing. The maximum plate thickness that can be used for this diameter of hole is 0.049 inch. After punching the plate surfaces must be buffed mirror smooth to remove all projecting feather edges so that the trask will slide freely across the surface as accumulations are pushed downhill by the slurry into a disposal launder delivering to the waste residue launder. Such plate screen would have a total surface area of 15 square feet and an open area of 22.7 per cent or 3.4 square feet to pass 626 cubic feet of pulp per hour or 10.4 cubic feet per minute. Pulp velocity through the plate to undersize will theoretically be only 3 feet per minute - 0.6 inches per second - so the plate will have ample screening capacity.

Should the above plate fail to cope with the trash problem, it will have to be replaced by a trommel, room for which has been reserved in the layout. This trommel is a cylinder 3 feet diameter by 6 feet 6 inches long, made of the same 0.049 inch thick S.S. plate punched with 1/16 inch holes, all as above and buffed smooth. Its open area will be 12.8 square feet. Situated between the Dorr classifier overflow and single tank, Item 11, it would have a slope of about $1\frac{1}{2}$ inches per foot and be driven at 15 R.P.M. by a 2 H.P. motor. Such a trommel would cope with a considerable amount of trash and handle fibrous litter better than any other device. Stainless steel is essential - oxidation rapidly blinds mild steel screens. The trommel oversize rubbish would be sluiced down a launder to the residue disposal launder, and the undersize collecting in the semi-circular shroud laundered by gravity to Item 11.

Item 11. Surge Tank.

62. This unit does nothing to the Dorr classifier overflow slurry it receives and hence its purpose will not be immediately apparent. It serves as a reservoir cushioning all variations in the slurrying rate such as are inevitable in wet weather - even to suspension of raw material feed for an hour - while still providing an uninterrupted flow of uniform quality feed to the hydroseparator degritters. Ballarat feeds its two hydroseparators from a surge tank of 600 cubic feet volume for the 400 cubic feet of hourly overflow from a Dorr classifier, and has found it advantageous. Moreover such a surge tank completes the smoothing of variations in raw feed composition into a uniform mix.

A 12' diameter by 10' deep tank, of steel or concrete, equipped with a Denver conditioner agitator assembly consisting of a 10 H.P. vertical motor, vertical shaft and a 30" diameter 4 blade downthrow impeller running at 150 R.P.M. or 1,100' peripheral per minute. This impeller is 3' above the tank bottom which is protected by a 3'6" diameter by $\frac{1}{2}$" thick M.S. wear plate. The central stand pipe of the standard Denver conditioner will not be required, but several anti-swirl baffles fixed to the tank wall could help to keep the coarser solids in suspension. A concrete tank would be on floor level, but a steel tank should sit on a heavily bitumen surfaced concrete pedestal a few inches above floor level to exclude floor water.

The Denver Company will be asked for advice.

Discharge is by diaphragm pump, Item 12, fed from a pipe in the tank wall close to the bottom, fitted with a plug valve hard against the outside of the tank. The entrance to this pipe inside the tank is kept clear of sand accumulation by positioning it on the swirl side of a 45° baffle plate attached to the tank wall. The surge tank is normally operated little more than half full to allow latitude for temporary stoppage of either slurrying operations or of the degritters. A lengthy power failure would cause no trouble even if it caught the tank full of slurry and so containing 130 tons of solids. When settled this would occupy 300 cubic feet and so barely reach the level of the impeller, which could safely be restarted. But a compressed air lance would have to be used to loosen solids around the tank wall and get them back into suspension quickly.

<u>Item 12. Diaphragm Pump.</u>

63. This delivers from the surge tank to No. 1 Degritter. In common with all the diaphragm pumps in the degritting section of the plant, it serves to accurately meter the volume of the slurry as well as transfer and elevate it. For uniformity, simplification of spares stock, ease of quick adjustment of delivery volume and freedom from grease contamination of the kaolin slip, all the diaphragm pumps in the plant are Oliver Diaphragm Slurry (O.D.S.) pumps made in Melbourne by Dorr-Oliver Pty. Ltd., and as used at Ballarat. Item 12 will have the following estimated duty:-

<u>Feed</u>: 9.5 tons solids plus 13.8 tons water per hour = 626 cubic feet or 80 U.S. G.P.M.

Sizing: approximately - 60 + 200 mesh 1.25 tons or 13 per cent mainly quartz grit.
- 200 + 30 microns 1.15 tons or 12 per cent mainly quartz grit.
- 30 micron 7.1 tons or 75 per cent wholly kaolin.

Pulp Density: 40 per cent solids by weight, pulp S.G. 1.33.

<u>Feed Intake</u>: positive pressure ranging from 2' to 10' head of above pulp, depending on depth of slurry in surge tank. Horizontal distance about 22' pump to tank.

<u>Feed Delivery</u>: 21 feet vertical lift 44 feet horizontal travel. The delivery pipe to rise 8' vertical above pump, then on self draining incline up to discharge end in feed well of No. 1 degritter.

The above details checked against the final drawings for accuracy, should be given to Dorr Oliver Pty. Ltd. for determination of the correct pump size and pipe details. Galvanized water pipe can be used and this pump need not be installed in duplicate, for it operates only 8 hours daily and can be stopped any time for brief attention. If practicable all O.D.S. pumps throughout the plant should be the same type and size.

Item 13. No. 1 Degritter.

64. This unit is the first of two identical hydroseparators which operate in series to complete the grit removal commenced by the Dorr classifier. This critical final degritting produces a slip consisting of paper quality kaolin accompanied by twice its weight of water, doing this with a current of water rising at a velocity high enough to lift kaolin particles of up to 30 microns diameter and finer into the overflow laurder, but not high enough to lift over anything coarser than 30 microns. The specification does not permit more than 2 per cent coarser than 30 microns. Whilst there is no material difference in specific gravity between kaolin and quartz grit to promote a differential settling rate, separation by water settlemtn in the hydroseparator is made possible by the fact that the kaolin particles are lamellar and hence slower to settle than the granular grit.

The design problem is to determine what this rising velocity must be and then to use this in conjunction with the flowsheet estimates of feed, reject grit, kaolin and water tonnages to be handled to calculate the size of this first hydroseparator degritter. Since each of the two hydroseparators is to produce roughly half of the total daily tonnage of degritted kaolin slip, No. 2 degritter will be a duplicate of No. 1.

65. Determination of the rising current velocity for a 30 micron kaolin overflow is based on the results of research into clay settlement rates, carried out some years ago by Mr. G. J. Robertson of the Ballarat Clay Co., and kindly made available by him. The formula below actually provides for a 20 micron overflow which is a precaution against overflow of anything coarser than 30 microns.

The standard Richards equation for estimating the settling rates of small quartz particles between 0.16 and 0.015 millimeter diameter when falling freely in the water is:

$V = 450\,D^2(a-1)$ V = the settling rate in millimeters per second.
D = diameter of the particle in millimeters.
a = specific gravity of the quartz (= 2.66).

This equation can be modified to the following form (which is rather different from that derived by Mr. Robertson, but yields similar results).

$V = 0.106\,D^2$ V = settling rate of quartz particle in inches per hour.
D = diameter of quartz particle in microns.

For quartz particles 20 microns diameter V = 0.106 by 20 by 20 = 42.4 inches per hour when falling freely in clear still water. The specific gravity of quartz and kaolin being practically identical, this can be accepted as the settlement rate of agglomerates of the lamellar kaolin. This 42.4 figure is close to the 36 inches per hour of the Mines Department evaluation method for separation of the nominal 20 micron kaolin fraction under still water conditions. The 97.5 inch rate yielded by the formula for 30 micron quartz or kaolin is similarly close to the 90 inches per hour of the A.P.P.M. evaluation method for separation of 30 micron fraction under still water conditions. (O.D.I. p. 54, 56).

66. But in the body of pulp filling the hydroseparator, the water is neither still nor clear, but is crowded with 40 per cent by weight or 20 per cent by volume of solids. These conditions greatly interfrere with the separation by settlement slowing down the settling rate of coarser particles, buoying them up until they

tend to overflow with the wanted kaolin. The 42.4 inch rate has to be reduced by a factor which measures the amount of this interference. Mr. Robertson has obtained a value for this factor by measuring the settling rate of clay in laboratory suspensions of various densities:

Per Cent of Solids in Suspension.	Settlement Rate Adjustment Factor or Multiplier.
0	1.0
10	0.74
20	0.578
30	0.437
36	0.36
40	0.305
50	0.21
60	0.118

The hydroseparators are to overflow a degritted slip containing 36 per cent of kaolin, as at Ballarat, so this will be the density of the uppermost layer in the pulp body, through which layer final grit elimination is effected. Using the factor for 36 per cent solids, the calculated settling rate for No. 1 degritter reduces to $15\frac{1}{4}$ inches per hour (i.e. 42.4 by .36) for operation at 100 per cent efficiency, as in the laboratory settlement tests. He assumes a separation efficiency of 70 per cent for a plant scale hydroseparator and makes a further small reduction for viscosity increase in cold weather. The $15\frac{1}{4}$ inches therefore further reduces to:

15.25 by 0.7 by 0.9 = 9.6 inches per hour for a 20 micron kaolin overflow. Calculation indicates that this is about the rising velocity in the Ballarat hydroseparators. It is the rate the S.M.C. hydroseparators are designed to provide.

67. No. 1 degritter is estimated to overflow 2.85 tons of 20 micron kaolin and 5.16 tons of water per hour as 225 cubic feet of kaolin slip 36 per cent solids. For a rising velocity of 9.6" = 0.8' per hour, the required surface area is $\frac{225}{0.8}$ = 280 square feet or a M.S. (or concrete) tank 19' diameter. The tank wall should be 10' deep in order to ensure that eddy currents generated by the motion of the rakes do not reach the surface to throw oversize grains into the kaolin overflow. The bottom is a cone with a slope of about $2\frac{1}{2}$" per foot to assist the movement of settled solids to the center discharge by the rakes. The rake mechanism, consisting of four arms driven at some rate between 1 and 4 R.P.M. by a H.P. variable ratio gear motor, can best be supplied by the Australian manufacturers of Dorr or Denver hydroseparators. The M.S. tank could be fabricated in Launceston.

The central shaft of their standard raking mechanism would have to be lengthened to suit the 10' deep tank, as standard hydroseparators have very shallow tanks. The overflow rate of 2.85 tons of kaolin from the 280 square feet of tank area is equal to 22.8 lbs. per square foot per hour, which is about the same as the Ballarat figure of 23.2 lbs. from their 14' diameter units. The collecting launder can be either on the inside or the outside of the top of the tank - usually the former, and the wood strip over which the kaolin slip flows into the launder must be carefully planed (and maintained) absolutely level. For No. 1 degritter shut down routine see para. 77 below.

The overflow of No. 1 (and No. 2) hydroseparator must be checked by routine testing at regular intervals - and every time there is any doubt - primarily for alkalinity. If this has fallen below the safe minimum growing flocculation of the pulp body will throw increasing amounts of oversize into the overflow, vitiating all of the degritted kaolin in storage in agitator No. 20. Prompt restoration of alkalinity to a safe level would be achieved by feed well addition of a strong solution of caustic soda from a container kept nearby for such an emergency. A simultaneous pulp density reading will indicate any necessary adjustment of the underflow diaphragm pumps.

68. Calculation of the correct dimensions of a hydroseparator is not a matter for which precision can be claimed. No harm will result in this case if the 9.6" per hour rising rate proves to be on the low side - meaning the 19' tank diameter is a little oversize; there are two more hydroseparators further on in the flowsheet to recover escaping kaolin. But a tank on the small side will do irreparable harm by forcing grit and oversize kaolin into the overflow, and there is no way of correcting this short of enlarging the tank.

Item 14. Diaphragm Pump.

69. This Dorr O.D.S. diaphragm pump removes the settled solids from the bottom of No. 1 degritter and delivers them into No. 2 degritter for treatment. Since all O.D.S. pumps are operated by electronic timers the rate of solids removal can be instantly varied to correct any discrepancy in the sizing of the kaolin overflow - the faster the settling solids and accompanying water are removed the less the overflow volume, the lower the rising velocity, and hence the finer the overflow kaolin, and vice versa.

Feed: 6.65 tons solids plus 8.64 tons water per hour as pulp containing 43.5 per cent solids, S.G. 1.4, and volume 400 cubic feet per hour or 50 U.S. G.P.M.

Approximate sizing of solids:

- 60 + 200 mesh 18 per cent or 1.24 tons per hour of quartz grit.

- 200 + 30 microns 17 per cent or 1.14 tons per hour mainly quartz grit.

- 30 microns 65 per cent or 4.27 tons per hour kaolin.

Feed Intake: approximately 3' below the bottom of No. 1 degritter, giving a positive feed head of 15 feet of pulp of S.G. 1.36.

Feed Delivery: 18 feet vertical then about 24 feet horizontal travel on a self draining gradient of about 2 feet fall, into feed well of No. 2 degritter.

Dorr Oliver Pty. Ltd. to be asked for their recommendation for this pump just as for Item 12, para. 63. This pump does not need to be installed in duplicate for it can be stopped for brief attention at any time and is idle 16 hours daily for any major repair.

Item 15. No. 2 Degritter.

70. Since an overflow of high quality is the primary function of No. 1 degritter, it will certainly show a poor recovery of the total amount of 30 micron kaolin in its feed, and so its underflow will require retreatment in No. 2 degritter. This is a duplicate of No. 1 in all details and its overflow rate is estimated to be the same as for No. 1. For shut down routine see para. 77.

Item 16. Diaphragm Pump.

71. This O.D.S. pump removes settled solids from the bottom of No. 2 degritter

and delivers them into scavenging hydroseparator Item 17, which provides the last chance of recovering paper quality kaolin still remaining in the residue from No. 2 degritter.

Feed: 3.8 tons solids plus 3.48 tons water per hour as pulp containing 52 per cent solids, S.G. 1.48, and volume 176 cubic feet per hour or 22 U.S. G.P.M. Sizing approximately 63 per cent of minus 60 mesh plus 30 micron quartz grit, and 37 per cent of minus 30 micron kaolin or 1.42 tons thereof.

Feed Intake: Approximately 3' below the bottom of No. 2 degritter, giving a positive feed head of 15 feet of pulp of S.G. say 1.45.

Feed Delivery: 21 feet vertical then about 65 feet horizontal travel on a self draining gradient of about 5 feet fall into the feed well of Item 17.

Dorr Oliver Pty. Ltd. to be given the above details and asked for their recommendation as for Item 12 and 14.

Item 17. Scavenging Hydroseparator.

72. As with No. 1, No. 2 degritter's primary task is a high quality overflow product and hence it will also fail to recover all the 30 micron kaolin in its feed. Its underflow can be retreated at a minor operating cost by a third hydroseparator of smaller diameter, but with a rake drive mechanism a duplicate of Items 13 and 15. This third unit will be a washing hydroseparator and will have a tank diameter of only 12' so that good use can be made of the small amount of wash water permissible. The feed solids containing an unknown, but probably considerable amount of valuable micron kaolin, will settle to the bottom and be raked into a central discharge cone. From this cone they enter the cylindrical discharge pocket and fall through a rising current of added clean water the objective of which is to wash out entrained 30 micron kaolin and lift it up into the overflow. But since no forecast of the probable loss of saleable kaolin in the underflow of No. 2 degritter can be made, the installation of Item 17 will best be postponed until plant operation indicates its economics.

73. This overflow will again be degritted kaolin of paper quality, but will be much too dilute for addition to the degritted kaolin slip produced by No. 1 and No. 2 degritters. It will therefore re-enter the degritting section circuit as return water forming part of the dilution water that has to be added to Dorr classifier Item 9. By removing the underflow pulp containing the washed solids at about the same rate as the feed pulp enters the feed well above, i.e. 176 cubic feet per hour, the added water will, in effect provide the whole of the overflow water. Since the kaolin in this overflow must still be of paper quality, the rising velocity in this scavenging hydroseparator will be adjusted to equal, but never to exceed, that in No. 1 and No. 2 degritters. A rising rate in excess of their 9.6 inches per hour would lift grit and plus 30 micron kaolin into the overflow and this grit would become a circulating load, building up from new increments each time it completed the closed circuit - scavenger overflow into Dorr classifier overflow into No. 2 degritter underflow and back into the scavenger unit and its overflow. Ultimately it could only escape by being forced into the overflow of No. 1 and No. 2 degritters, vitiating the sale kaolin with amounts of grit and oversize in excess of permissible limits. The addition of 2 tons of wash water or 450 gallons per hour would provide a rising velocity of a safe 8 inches per hour. This wash water would come from the constant head

supply through master and control valves and rotameter. Finally there is the question of the tank depth. The danger from vitiation of the overflow by eddy currents caused by the rakes will be very much less than with the two degritters, for the pulp body will be far less dense and viscous and rake speed will be lower. So a tank wall only 5' deep to the start of the coned bottom would probably be safe, but all risk could be eliminated at a minor addition to overall cost by making the tank 10' deep.

74. The feed to this scavenging unit will be dense at 52 per cent solids so about 1.5 tons of the 2 tons of wash water will be used to dilute the feed as it enters the feed well. The remaining 0.5 tons will be fed into the discharge pocket. Since the operation of the underflow removal pump is an intermittent one, all of the washing action will be concentrated into the short period when the pump is discharging. Over this short period, the rising velocity in the discharge pocket will be perhaps 10 times the 8" per hour tank average, and so washing will be thorough. During the pump intake, stroke velocity will fall to a negative figure and the lifted solids will settle into the pump intake. Nevertheless there may gradually accumulate in the tank a body of solids too coarse to escape in the overflow, yet too fine to reach the pump intake. A slight reduction in the amount of wash water and corresponding increase to feed well dilution will correct this situation. Shut down routine is given in para. 78.

75. Instead of using a 12' diameter hydroseparator and its O.D.E. discharge pump for scavenging, this perhaps might be done - though very inefficiently - with a device that could be manufactured in Launceston. This is the Edgar clay drag. Ballarat installed one of these units to dewater the degritting section waste residue to make it sufficiently dry to be stored in a bin and trucked to the disposal area. This it does satisfactorily. The water it extracts during this dewatering operation does contain some valuable kaolin and so is sent back to the degritting section as return water. But such scavenging as it does so do is only a side effect of its primary function.

The machine is operated intermittently by an electric timer to provide quiescent pool conditions to allow the residue solids to settle for dragging out. These solids are not washed except very incompletely by a single row of sprays at the sand discharge end. Feed is continuous and the pool area can be designed to provide a rising current of 8 or 9 inches per hour. While the rake chain is stopped, the pool is quiescent and the overflow may not carry any solids coarser than the 30 micron limit. But when running, the rake chain creates substantial disturbance to the shallow pool and so must set up eddy currents that will force grit into the overflow. There is thus neither efficient scavenging of valuable kaolin nor any control over the quality of such as is recovered - only satisfactory dewatering of the residue for disposal. The machine is not recommended for S.M.C. To repeat, security of quality of its sale kaolin is of greater importance to S.M.C. than a saving in capital expenditure. In a private communication (18/12/61) Mr. Robertson indicates a preference for a hydroseparator instead of an Edgar Drag for this scavenging.

Item 18. Diaphragm Pump.

76. This pump removes the underflow from scavenger unit, Item 17, and delivers it into the waste residue tail race for disposal. Its primary purpose is to act

as an intermittent discharge valve to permit safe, yet thorough washing of usable kaolin from the residue prior to disposal. Since this pump does not have to elevate the underflow, but merely delivers it downhill into the residue launder immediately below, it might be thought some type of spigot discharge could be used instead. A spigot discharging intermittently by cam and ball gear could be used, though this gear works better on paper than in practice. To maintain the solids outflow in balance with inflow, its motor would require mechanical speed variation equipment equal in capability to the pump electronic gear. On the other hand, a simple straight spigot would require the addition to Item 17 of some 13 to 17 tons of dilution cum wash water (instead of only 2) to reduce the spigot effluent density to the 25-20 per cent solids necessary for continuous discharge. Control over the critical rising velocity would be lost, resulting in a circulating load of difficult size grit, building up in the 17-9-13-15 overflow circuit from which it would ultimately escape via the only avenue possible - the overflow of 13, to vitiate the sale kaolin. Spigot wear and occasional blockage by slumping solids would aggravate the situation. The pump is the best solution.

<u>Feed:</u> 3.8 tons solids (approximately) plus 3.5 tons water per hour in a pulp of S.G. 1.48, containing about 52 per cent solids. Volume 176 cubic feet per hour or 22 U.S. G.P.M. The solids sizing ranges from 60 mesh down to 30 microns.

<u>Feed Intake</u>: About 4 feet below the bottom of hydroseparator, Item 17, giving a positive feed head of some 10 to 14 feet of pulp of S.G. say 1.45.

<u>Feed Delivery</u>: A short downhill discharge pipe about 8 feet long with 2 or 3 feet fall.

Dorr Oliver Pty. Ltd. recommendation to be obtained for this single pump.

<u>Shut Down Procedure for the Three Hydroseparators.</u>

77. The shut down at the end of each shift (day) and over each week-end will of course find all three hydroseparators full of pulp. No. 1 and No. 2 degritters will each be holding 2,550 cubic feet of pulp containing 40 tons of solids largely -30 micron kaolin. About 70 per cent in the case of No. 1 and 50 per cent for No. 2. If allowed to settle these highly dispersed solids would certainly pack tight into a hard layer 3 feet thick. It will therefore be best to keep the rakes running in both degritters throughout the 16 idle hours daily and recirculate the underflow of each unit back to its feed well, for which re-routing a simple splitter box will be provided above each degritter. On plant resumption each morning the pump discharges are re-diverted to their normal delivery destinations.

The same procedure could be followed over the week-end, but more likely both units will be shut down for normal inspection and maintenance. The rakes would be raised 3 feet or so just before stopping, and the underflow shut off from the diaphragm pumps. On restarting the pump discharges are diverted to recirculate back to the feed wells, and the settled solids loosened with compressed air lances. The rakes are started and gradually lowered to their normal bottom position, a process that will take an hour or so of considerable patience and care. Meantime the overflow of Dorr classifier item 9 will be accumulated in the surge tank until the degritters are back to normal and ready to take new feed.

78. The situation with the scavenging hydroseparator is simpler. At the end of the day, pumping of its underflow to residue disposal is continued for 15 minutes

or so after which the rakes are raised several feet and stopped. The pump is then stopped, and the tank left three parts full of pulp for the solids to settle. Being granular and largely denuded of kaolin these solids will not pack tight so next morning the rakes can be restarted and gradually lowered without trouble.

Item 19. Final Trash Screen.

79. This screen is the final trap for the removal of the last traces of wood wool, vegetable trash and plus 200 mesh tramp oversize from the degritted kaolin slip before it enters the finishing section. The screen used by all the South Carolina, Georgia, Florida kaolin recovery plants is the SWECO made by the South Western Engineering Co., 4800 Santa Fe Avenue, Los Angeles 58, California. The size they use and the size recommended for S.M.C. is the stainless steel 48 inch diameter unit, S48588 "Vibro-Energy Separator" fitted with 230 to 250 mesh stainless steel cloth, and 1 H.P. motor. It was selling at $2700 December 1961, exclusive of screen cloth. The screen has no real work to do in the way of oversize elimination - in fact it is quite unable to cope with more than trace amounts of oversize though it can pass the 20 micron kaolin slip through its screen at a deluge rate. Its primary task is that of policeman trapping the last traces of contamination including grit coarser than 200 mesh. To make certain no plus 200 mesh grit gets through - the specification does not permit any in the sale kaolin - a screen cloth rather finer than 200 mesh is chosen to allow for aperture wear. Given an even feed and only trace amounts of oversize the screen will require no operator attention and the only maintenance is replacement of screen cloth. The only indication of screen cloth life is a SWECO Catalog reference to a unit which handled 50 U.S. G.P.M. of coating clay slip 70 per cent solids for over 10,000 hours without change of screen cloth.

Feed: 5.7 tons 20 micron kaolin plus 10.3 tons water per hour, equivalent to 450 cubic feet of pulp 36 per cent solids, or 56 U.S. gallons per minute. This task is well within the unit's capacity.

This completes the Degritting Section equipment.

Item 20. Storage Agitator for Degritted Kaolin Slip.

80. This first unit of the finishing section receives the screened degritted kaolin slip over the day shift operation of the degritting section, and spreads it over the 24 hour operation of the finishing section. The designed daily production of slip of 36 per cent solids density is 40 tons kaolin in 72 tons water, a total volume of 3150 cubic feet produced at the rate of 450 cubic feet hourly over 7 hours. Should the density fall as low as 31 per cent solids for a shift - this sort of accident will occasionally happen for some hours - the figures grow to 40 tons kaolin in 90 tons water, or 3750 cubic feet at 535 c. feet per hour. This accidental volume must be provided for.

A concrete or M.S. tank 23 feet inside diameter would hold 416 cubic feet per foot of depth, or 3750 cubic feet with a 9 feet working and 10 feet overall depth. A bottom slope of 2 inches per foot to the central discharge cone would permit emptying to dryness when required. A 2 or 4 arm rake mechanism 20 feet diameter driven at $\frac{1}{2}$ to $1\frac{1}{2}$ R.P.M. by a 3 H.P. variable ratio-gear motor should be adequate to maintain suspension. But these details will be specified by the manufacturer of the equipment for item 20 - either Dorr Oliver Pty. Ltd. or Denver Equipment Co., to whom this storage problem should be referred because of their long experience. Since the slip will be mildly alkaline the rake gear and four or more vertical anti-swirl baffles attached to the tank walls can all be of mild steel. But added insurance against non staining would be a heavy coating of bitumen, or a rubber coating as applied by Dunlop Rubber Aust. Ltd. (as was done at Mt. Morgan) to all steel surfaces. Prevention of swirl by deflectors is essential for large steel tank agitators of this type, otherwise the rake arms will soon build up a wave of destructive potential. Likewise it will probably be necessary to steady the rake shaft bottom end in a step bearing filled with mercury.

81. This agitator would be emptied between 8 and 10 a.m. each morning, and be full by 5 p.m. each evening, remaining full overnight and over each week-end. The rakes must be kept running while there is kaolin slip in the tank. A short stoppage would create no problem but a lengthy stop - as due to long power failure - would result in settlement of the kaolin into a hard compacted layer requiring much compressed air lancing to loosen it. Provision will be required for raising the rakes several feet above bottom for such a situation. The re-lowering will require care and patience to avoid twisting the shaft. It would be better to provide against a lengthy power failure by installing a stand-by petrol driven generator, which would automatically cut in to keep this agitator, the hydro-separators and their pumps in operation.

82. Finally all oil and grease on the raw material in quarry and elsewhere will end up in this agitator, floating to the surface just outside the feel well. It will need to be confined by a froth ring of considerably larger diameter than the feed well, and dipping 3 or 4 inches below water level. Floating scum can be removed periodically by a hand operated suction lance.

<u>Item 21. Centrifugal Pump.</u>

83. This intermittently operated pump is required for rapid transfer of batches of slip from No. 20 to bleach tanks item 22, or maybe direct to heating tanks item 25. Each batch to a bleach tank will amount to 1050 cubic feet (6560 gallons) of 36 per cent solids pulp, or as much as 1250 cubic feet (7810 gallons) should slip density have fallen as low as 31 per cent solids. A 4 inch (delivery) Warman rubber lined centrifugal pump speeded to deliver 300 G.P.M. of pulp S.G. 1.3 would require a 10 to $12\frac{1}{2}$ H.P. motor. Such a pump has a 6 inch diameter intake and would be fed from a conventional feed box supplied by a 6 inch pipe from the bottom of the agitator No. 20. This underflow pipe, carried in a tunnel for access, would have a 6 inch plug valve at the agitator end, with a compressed air inlet between valve and agitator for clearing any blockage.

84. A 6 inch galvanised discharge pipe rises 36 feet vertical above the pump, and then continues on a slope of say $\frac{3}{4}$ inch per foot for about 75 feet to the 6 inch header over the bleach tanks so as to drain clean into them. The 7 cubic feet of pulp in the vertical section, the weight of which will be taken off the pump by supports, will drain back through the pump into the feed box. A by-pass back to the agitator will assist in getting settled solids back into suspension after a break-down. The 6 inch delivery pipe gives a velocity of 4 feet per second, which is more than ample, and 20 to 25 minutes will be required to charge each bleach tank.

Item 22. Batch Agitators for Bleaching.

85. Four bleaching tanks are required, each holding one third of the day's production of degritted kaolin slip, the fourth tank being necessary to bridge the overlap in the operating cycle. Each tank is 13'6" diamter by 10' deep, and holds 1250 cubic feet of 31 per cent density pulp at 9' working depth. Normal contents at 36 per cent solids will be 1050 cubic feet, and 7'6" depth. The details of their construction and type of agitator mechanism will have to be decided in consultation with I.C.I.A.N.Z. Ltd., as will the method of injecting the chlorine gas. Since chlorine gas is a very serious health hazard these four tanks will be in the open air with only an umbrella roof to shed rain. They will have covers to confine the gas and exclude wind blown dust. As the gas progressively lowers the pH of the slip from 8 to 5.5 (O.D.I. p.26) the slip will change from its initial dispersed state to a semi-flocculated condition, easing agitation problems, but becoming increasingly corrosive. Bleaching will be cheapest with bottled chlorine gas (O.D.I. p.27), and so there will be equipment adjacent to the bleach tanks for bottle handling and pressure reduction.

86. The following cycle will permit bleaching to be done entirely on dayshift. It starts with storage agitator No. 20 full of degritted kaolin slip at 8 a.m. Monday (M.T.W. for Monday, Tuesday etc.), and with one bleach tank, say No. 1, full of bleached slip also carried over the week end. Chlorine injection time is assumed to be 30 minutes per batch.

	Bleaching Tank			
	No.1	No.2	No.3	No.4
Bleached slip starts feeding to filter heating tanks	8 a.m.M			
Tank empty	4 p.m.			
Tanks start filling from storage No. 20		8 a.m.M	8.30 a.m.M	9 a.m.M
Bleaching starts after completing chlorine input		9 a.m.	9.30 a.m.	10 a.m.
Bleach ends after 5 hours contact .		2 p.m.	2.30 p.m.	3 p.m.
Antichlor treatment and testing completed		3.30p.m.	4 p.m.	4.30 p.m.
Bleached slip starts feeding to filter heating tanks		4 p.m.	12 p.m.M	8 a.m.T
Tanks empty		12 p.m.M	8 a.m.T	4 p.m.
Tanks start filling	8 a.m.T	8.30 a.m.T	9 a.m.T	
Bleaching starts	9 a.m.	9.30 a.m.	10 a.m.	
Bleach ends	2 p.m.	2.30 p.m.	3 p.m.	
Antichlor and testing completed . .	3.30 p.m.	4 p.m.	4.30 p.m.	
Bleached slip starts feeding to heating tanks	4 p.m.	12 p.m.T	8 a.m.W	
Tank empty	12 p.m.T	8 a.m.W	4 p.m.	
Tank starts filling	8 a.m.W	8.30a.m.W		9 a.m.W
Bleaching starts	9 a.m.	9.30a.m.		10a.m.
Bleach ends	2 p.m.	2.30 p.m.		3 p.m.
Antichlor and testing completed . .	3.30 p.m.	4 p.m.		4.30 p.m.
Bleached slip starts feeding to heating tanks	4 p.m. W	12 p.m.W		8 a.m.Th.

And so on. The cycle is based on 5 hours contact with the chlorine (O.D.I. p.21 states there is no advantage in longer contact). For more than 5½ hours more tanks and a new cycle would be required.

87. This cycle shows the filters switching from one bleach tank to the next precisely every 8 hours. But bleaching is on an 8 hour batch schedule while filter feed heating is on a 3 hour basis (see item 25). By reducing the batch

heating to a 2 hour 40 minute cycle to give 3 heating batches per bleach batch the two schedules would be in step. But it is not credible that they would remain in step for whilst there is little doubt that bleaching will soon operate consistently within its schedule there will certainly be irregularities in filter operation with such difficult feed as kaolin.

Item 23. Agitator Balancing Tank.

88. One practical solution to this reconciliation problem is to make the two time schedules independent of one another by inserting an agitator tank between bleach and filter heating tanks. Into this balancing tank will go any bleached slip still remaining in a bleach tank due to start refilling, to be held therein until needed to make up a deficiency in a heating tank batch. With the filter plant operating trouble free and on schedule about 150 cubic feet bleached slip will go into and be withdrawn from the balancing tank once each 24 hours, for which purpose it would be positioned above the level of pump item 24 and be connected through isolating valves to both delivery and intake sides thereof. By trouble free filter operation is meant that the designed provision of 15 per cent loss in filter running time is spread evenly over the three shifts in which case a balancing tank of about 250 cubic feet capacity would be ample. But should the majority of the 15 per cent down time allowance happen to occur on night shift, a much larger storage space will be required to free the related bleach tank for refill at the beginning of the day shift. Some of this required extra storage will be found in the bleach tank next in sequence to start feeding the filters at 8 a.m. later in the morning. It will have some 2 feet or 280 cubic feet of vacant capacity if the degritted kaolin slip is at the designed 36 per cent solids density. And since this tank is not due to start re-filling for another 24 hours there will be time for the lag in filtration to be overcome.

A Denver conditioner of the simplest type 7 ft. diam. by 7 ft. deep, with a working volume of 250 cubic feet, or an 8 ft. x 8 ft. at 350 cubic feet would make a satisfactory balancing tank.

89. The weight of kaolin in a bleach tank will be 13.3 tons dry weight regardless of density. The requirement of chlorine gas, assuming the O.D.I. maximum rate of 0.5 per cent of the weight of kaolin, would be 13.3 by 0.005, or 150 pounds. If introduced at the bottom of the tank the chlorine pressure would have to exceed the 4.8 lbs. per square inch static pressure of the 9 ft. depth of slip of S.G. 1.24 for 31 per cent solids. Mr. Dawkins of I.C.I.A.N.Z., Launceston, advised that the gas can be taken from the cylinder through a reducing valve set at 10 lbs. pressure at 250 to 300 lbs. per hour, so that 150 lbs. could be delivered into the bleach batch in 30 minutes, or in less time if a safe 15 lbs. pressure be used. (Note that at 20 lbs. freezing and polymerization cause blockage of the chlorine piping). At the moment there is no information on how fast the slip will absorb chlorine. The bleach cycle provides 30 minutes but slower addition continuing into the 5 hour bleach period may be better.

90. It is not presently known how much of the added chlorine will remain unused in the slip at the end of 5 hours. It will certainly be an appreciable amount since considerably more chlorine will always have to be used than the amount actually consumed by the discoloration reaction. To eliminate the health hazard and ease corrosion problems during heating and filtration of the bleached slip

this residual free chlorine will need to be destroyed by the addition of dry sodium bisulphite to the tank at the conclusion of bleaching. The plant laboratory technician will determine the optimum additions of chlorine and bisulphite, and check the bleached slip for quality before releasing each batch to the filters. A batch that fails to satisfy the brightness specification will have to be pumped to waste unless a switch can be made in its position in the filter feed sequence to permit bleaching to be resumed and continued into the afternoon shift. Clearly the way to avoid the threat of such a dead loss is to make full use of sampling guidance in the quarry. Note that chemical destruction of the excess chlorine may cause some degradation in brightness (O.D.I. p.24).

Item 24. Centrifugal Pump.

91. The bleached slip will be transferred to the filter heating tanks in 400 to 470 cubic feet batches at 3 hourly intervals by a pump, probably a rubber lined centrifugal, through corrosion proof piping, probably plastic, but possibly rubber. This pump will have a by-pass delivery to the waste residue launder for the disposal of any bleach batch that cannot be brought up to specification. It will have another by-pass delivery back to all four tanks for the transfer of bleached slip from one tank to another. Its intake and delivery will also have by-pass connections to balancing tank item 23. The pump would best be a duplicate of Item 21 so that with a capacity of 300 G.P.M. in the 400-470 cubic feet batch (2500-2950 gals.) could be transferred to the heating tank inside 10 minutes. A 6 inch delivery pipe would have a pulp velocity of 4 ft. per second, but if this size creates any problems there would be no material disability in the higher velocity consequent on the use of a smaller pipe line.

Item 25. Filter Feed Heating and Flocculating Tanks.

92. These twin batch tanks provide the means for heating the bleached slip from cold to 140°F and then for flocculating it, this heating having the effect of doubling the filtration rate and thus halving the filter area required as compared with filtration of cold slip. Heating is done by injection of steam from the source that supplies the dryer. The twin tanks operate alternatively, one heating a batch while the other is emptying through the filter plant, together handling each 24 hours 8 batches of bleached slip, each batch containing 5 tons of kaolin in 400 to 470 cubic feet of slip, according to pulp density. Heating is estimated to take $2\frac{3}{4}$ hours and to add $1\frac{1}{2}$ tons of condensed steam to the 9 to $11\frac{1}{4}$ tons of water already present in the batch. The remaining 15 minutes of the 3 hour cycle will be taken up with flocculating by addition of aluminium sulphate or other acidifier. This batch is then switched over to filtration and a new heating batch started in the now empty twin tank. The main features and the operation of these tanks are all based on Ballarat equipment and practice, and as a precaution their criticism should be sought on the final design before commencing fabrication. The source of the steam injection calculations is Mr. Kuzniarski of the Launceston Technical College.

93. The equipment consists of a single M.S. tank 16 feet long by 8 feet wide by 8 feet deep vertical sides, divided by a central partition into twin 8 feet by 8 feet tanks, both with pyramidal bottom 4 feet deep. They are lead lined to withstand acid pulp. Each tank has an 18 inch diameter 3 blade impeller positioned well down in the pyramid section, running at 200 R.P.M. (940 feet per

minute) to lift (?) the pulp during heating but increasing if necessary to 320 R.P.M. during flocculation. Drive is from 5 h.p. variable ratio gear motors totally enclosed against steam penetration, and set at tank end clear of splash. Each tank has two 4 inch steam header pipes with each header feeding 12 half inch down pipes delivering steam into the pulp at pyramid level. Header and down pipes will require to be of stainless steel. The tanks have lead lined splash tight cover panels and corrosion resistant hoods connected to stacks to outside air. These hoods will need to have collecting launders to carry away acid condensate clear of tanks and operator. The tanks are lagged to reduce heat loss. The combination of stainless steel and lead in contact with the acid pulp suggests a potential electrical corrosion situation requiring investigation.

94. This heating tank assembly is positioned well above floor level not only to keep escaping steam clear of the filter plant and other finishing section units, but also to provide for delivery of the hot viscous kaolin slip to the filter by gravity flow. The two pyramid bottoms are fitted with 5 inch bronze plug valves and 5 inch pipes set on a gradient sufficient to ensure free flow of the viscous pulp. Contact between different metals used in the tank assembly will need to be prevented by fibre insulating washers to reduce electrical corrosion.

95. The steam requirement is estimated to be 27,100 lbs. per 24 hours (see para. 96 for calculations) or 3,400 lbs. for each of the 8 heating batches. Assume steam at 10 lbs. p.s.i.g. = 25 lbs. pressure absolute. This will be ample to overcome the static pressure of the pulp, 3.4 lbs. p.s.i. Since 1 lb. of steam at 25 lbs. pressure abs. occupies 16.3 cubic feet the volume of steam per batch is 3,400 by 16.3 = 55,420 cubic feet and to promote maximum condensation and heat transfer, injection will be spread over about $2\frac{3}{4}$ hours at 5.7 cubic feet per second. A steam pipe velocity of 200 feet per second is normal for a low pressure drop so the cross section area of the injection down pipes would be 5.7 ÷ 200, or 0.0285 square feet, 4.104 square inches. This would be provided by 21 half inch diameter pipes. So each of the two headers per tank would have say 12 half inch down pipes spaced about 6 inches apart.

96. The steam requirement for heating the slip is derived as follows.

Assume 40 tons degritted kaolin @ 31% solids instead of 36% as in flowsheet = 40 (90) = 130 tons pulp 24 hours.

Theoretical heat requirement to raise 130 tons pulp from 50 degrees to 140°.

40 tons kaolin	= 40 x 2240 x 90 degrees x .2	=	1,612,800 B.T.U.
90 tons water	= 90 x 2240 x 90	=	18,144,000 B.T.U.
	Total		19,756,800 B.T.U.

Steam at 212 degrees falling to water at 140 degrees F releases 970 + 70 = 1040 B.T.U. per pound.

So weight of steam theoretically required $= \frac{19,756,800}{1,040} = 19,000$ lbs.

Assume heat transfer from steam to pulp 70% efficient, then requirement becomes 27,100 lbs. steam.

Items 26, 27, 28, Flocculant Mixing and Feeding.

97. The daily requirement of commercial aluminium sulphate is 1200 lbs. at 30 lbs. per ton of kaolin (O.D.I. p.28). Its cold water solubility being 3 lbs. per gallon - a week's requirement of 6000 lbs. dissolved in 2000 gallons would be a convenient week end job. Each batch of hot slip would have 50 gallons of this flocculant solution fed into it through a rubber or plastic hose from a supply tank above, thorough incorporation in the rapidly thickening pulp being aided by prior speeding up of tank 25 impeller.

Item 26 is a lead lined M.S. dissolving tank 8 feet square by 7 feet deep, equipped with a rubber covered impeller 18 inch diameter driven at 200 R.P.M. by a 3 H.P. vertical motor. The impeller shaft also must be rubber sheathed. Two 8 feet by 2 feet 6 inches by 2 feet 6 inches rectangular stainless steel baskets made of 10 mesh heavy gauge S.S. screen cloth sit on a lead covered grid of cross members positioned 4 feet above the tank bottom. Each basket holds 50 cubic feet, or 3,000 lbs. of crushed aluminium sulphate which weighs 60 lbs. per cubic foot. A charge of 1,600 gallons of water will just flood the bottoms of the baskets, and by completion of solution progressive additions will have brought the volume up to 2,000 gallons mark. A $\frac{3}{4}$ inch rubber lined Warman centrifugal pump $\frac{1}{2}$ H.P. motor periodically lifts a batch requirement of 50 gallons through a plastic pipe up to a lead lined head tank positioned above the heating tank and provided with an overflow back to the dissolving tank. This overflow could automatically trip the pump motor.

Item 29. Filter.

98. Estimation of commercial filter capacity contains an element of chance for laboratory filtration test figures are notoriously hard to translate into probable commercial performance. No firm recommendation for filter equipment can be made at this stage - there is need for more information on filter types in use in U.S. kaolin plants, filtration conditions and unit area capacities achieved, and a round table conference to interpret and condense this data into a unit that will meet S.M.C. requirements. As frequently warned in previous paragraphs there will be occasions when the density of the degritted kaolin slip falls below the designed 36 per cent solids - perhaps to 31 per cent. As a safety measure therefore the filter should be able to cope with 40 tons kaolin and 90 tons water in 130 tons slip per 24 hours, a pulp volume of 3,750 cubic feet. Fortunately the research investigation filtration tests were done on slip of 30-31 per cent solids, and on slip heated to 140°F they indicated a filtration rate of 9 lbs. kaolin per square foot of cloth area per hour. The cake so produced retained 46 per cent moisture. At 9 lbs. a cloth area of 415 square feet is indicated for a filter running trouble free 100 per cent full time - a situation unattainable in practice - or 5,000 square feet allowing for 85 per cent actual running time. This suggests a drum size of 10 feet 6 inch diameter by 16 feet length. The cake will be sticky and very thin - of the order of 3/16 inch thick - and the problem of reliable continuous discharge by scraper will be accentuated by the likelihood of a large drum developing distortion from a true circle, and requiring periodical restoration to accuracy by shaving with a traversing chisel. Some other method of cake removal is indicated.

Item 30. Filter Overflow Return Pump.

99. The pulp level in the filter feed will occasionally rise to overflow level, requiring some means for returning the excess to the heating tanks. This overflow will be hot and viscous, unsuitable for handling with a centrifugal pump. Ballarat appears to use a gear type pump, so their advice should be sought on the point.

Item 31. Filter Cake Conveyor.

100. Since the filter cake is to be dried in its flocculated state (H.K. Turner's letter of 1/2/63), it will be unsuitable for transfer to dryer by pump. Even if its feed level be low enough - or the filter be elevated - to allow the cake to drop vertically on to the dryer, some modified form of blade conveyor of the log washer type will probably be necessary to reconcile cake discharge and dryer feed widths.

Item 32. Dryer.

101. This item also remains unspecified pending completion of current enquiries. Present indication is for an indirect steam-heated dryer of the type in use at Titan Products Ltd., Blythe (H. K. Turner's letter of 1/2/63).

102. The approximate quantity of steam required to dry the filter cake can be derived as follows:-

40 tons kaolin filter cake 46% H20 = 34 tons water to be evaporated per 24 hours.

To raise 40 (34) tons per 24 hours from 50 degrees F to 212 degrees requires:-

For 40 tons kaolin = 40 x 2240 x 162 x .2	=	2,903,040 B.T.U.
For 34 tons water = 34 x 2240 x 162	=	12,337,920 B.T.U.
Total		15,240,960 B.T.U.
Then to evaporate 34 tons water at 212 degrees to steam at 212 degrees requiring 970 B.T.U. per lb. = 34 x 2240 x 970	=	73,875,000 B.T.U.
Total heat theoretically required for drying		89,115,960 B.T.U.
Steam at 212 degrees required = $\frac{89,115,960}{970}$	=	91,870 lbs.
Assuming dryer heat transfer efficiency 70 per cent, steam required	=	131,240 lbs.

Total Steam Requirement and Fuel Equivalent

103. This is a convenient point for collection of the steam requirements into a total figure, and for calculation of the approximate amount of fuel required for generation.

Total steam required:

27,110 lbs. heating + 131,240 lbs. drying = 158,350 lbs. = 70 tons water 24 hours, or 15,680 gals., or 6600 lbs. steam per hour over 24 hours.

Coal required:

To heat 1 lb. cold water from 50 degrees F to steam at 212 degrees requires 162 + 970 = 1132 B.T.U.

To heat 158,350 lbs. water from 50 degrees to steam at 212 degrees requires 152,850 x 1132 = 173,026,200 B.T.U.

Coal at 10,000 B.T.U. lbs. amounts to 17,306 lbs. = 7.7 tons at 100 per cent boiler efficiency.

At 80 per cent efficiency for modern steam generator = 9.6 tons coal/day
At 50 per cent efficiency for old type Lancashire boiler = 15 tons coal/day

Say 10 tons per day or 2,500 tons per year
Oil at 18,500 eq. 5.4 tons per day, 1350 tons per year.
} at 80 per cent boiler efficiency, and 70 per cent efficiency of units.

Note: The coal tonnage used in paragraph 28 for the preliminary estimate of production cost is 65 tons weekly, based on a conservative 60 per cent boiler efficiency for S.M.C. circumstances.

Government Boiler Regulations

104. The use of steam in Tasmania is governed by regulations which require the steam generator to be under the control of a certificated boiler attendant unless - and this is the sole exception - the generator is an electrically heated fully automatic unit. Apart from an odd over-age man there are practically no floating certificated men available. Hence steam users train their own to examination standard, with the co-operation of the State Machinery Inspector for the district, and naturally train sufficient reserve men to provide for emergencies. Being a certificated man his failure to report on shift would shut-down the dryer altogether, reduce filtration rate to half and, within a few hours reach back to shut down the Degritting Section for lack of degritted slip storage room in the Finishing Section. The boiler attendant is thus a key man, and a negative decision coming from him, or through him will quickly bring the S.M.C. plant to a standstill. Regulations permit him to perform other tasks provided he is always within reading distance of the boiler water and pressure gauges. Oil fired semi-automatic steam generators allow him to be used on a rather greater range of jobs in the vicinity.

Item 33. Dried Kaolin Conveyor-Elevator.

105. Considerable thought will have to be directed to the choice of the best means of elevating the hot dry kaolin from dryer into a bulk storage bin. An endless screw conveyor-elevator, or a Fuller-Kynyon pump as used in cement plants are two possibilities that suggest themselves, and there are other specialized units in use in U.S.A. Whatever the type it will have to be totally enclosed right from dryer to the junction of its discharge chute with the bin cover in order to keep the vicinity, and in fact the whole plant, free of kaolin dust. The fine dry kaolin is a prolific dust producer, and must be hermetically sealed within walls right from dryer until enclosed in the bags in order to maintain clean healthy working conditions that promote harmonious labor relations, and comply with Government Health Regulations.

Item 34, 35, 36. Bulk Storage Bin for Sale Kaolin.

106. The capacity of this bin should be not less than 80 tons or two days' production, and the volume necessary to provide this will depend on the figure taken for the weight of a cubic foot of the kaolin as delivered into the bin. This will require some research. Its shape should provide a pyramid, or conical bottom to form a suitable junction with an automatic bagging machine. It will need to be totally enclosed and sealed against dust escape, and to be connected to a small exhaust fan for removal of the steady inflow of air accompanying the

incoming kaolin. This fan would deliver through a nest of small multiclones (Dorr Oliver Ltd.) sitting on the bin cover, delivering the recovered kaolin back into the bin through a motor driven multivane rotary valve seal, and discharging practically clean air to outside the building. Kaolin recovery by these high velocity cyclones would be about 95 per cent. Operation of the bin under reduced pressure of a few inches W.G. in this manner would de-aerate the kaolin and so increase the holding capacity of the paper bags.

Item 37, 38. Bagging.

107. The paper bags used at Ballarat are understood to hold only 60 lbs. of dry (and probably considerably aerated) kaolin. On this basis this is probably the weight of one cubic foot in this condition. On this basis also the daily 40 tons at S.M.C. would need 1500 bags requiring the filling and closing of 4 bags per minute over 7 hours of the day shift. De-aeration should appreciably increase bag content, and to that extent liberalise this tight bagging schedule, but even so a larger bag seems necessary. Automatic bagging equipment for dust free handling of such fine material undoubtedly exists, and will need to be obtained. The bagged kaolin could be moved from bagging units into storage by a belt conveyor, but the use of pallets, loaded at the bagging site and stacked in the store by fork lift truck looks more practicable. An examination of bag handling and storage in cement plants would be worthwhile.

Item 39. Bagged Kaolin Storage.

108. S.M.C. seems to be the best place for holding the whole stock of bagged kaolin. The space needed will be determined largely by the railway time table. The store could consist of a concrete floor covered by a single span roof projecting beyond the supporting side columns to form side verandahs of generous width. End walls, but no side walls. The fork lift could then progressively fill one side free of interference while the transport contractor is emptying the other side, working under the verandah. Facilities will be needed to enable the contractor to pick up the loaded pallets at S.M.C., transfer them into rail waggons at Herrick, and simultaneously reclaim returned pallets from the same waggons. No bag storage at Herrick.

Plant Laboratory.

109. Finally, the Department of Mines Research Laboratories in Launceston will be the best source for advice on the layout and equipment of the plant control laboratory, and for the devising and establishment of each of the routine control tests that will be necessary in plant and quarry. A suggested position inside the plant for this laboratory is shown on the layout plan.

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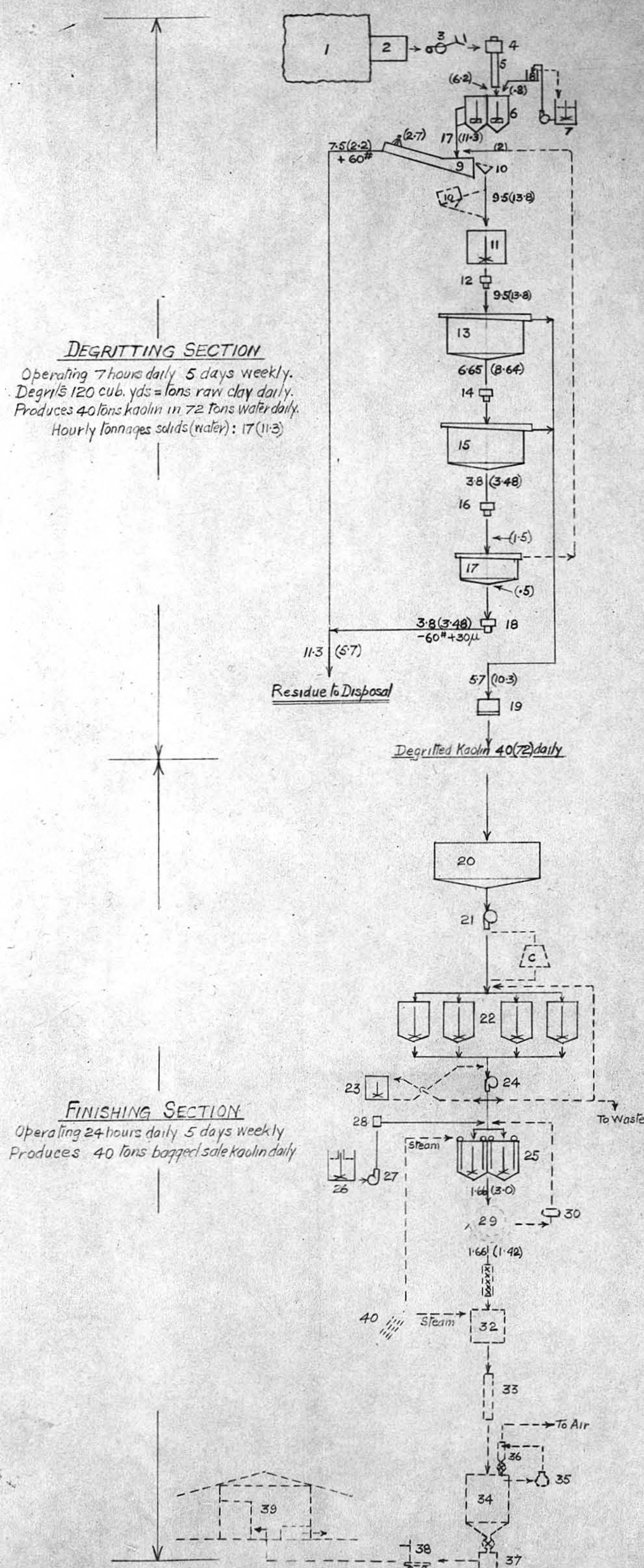
284132



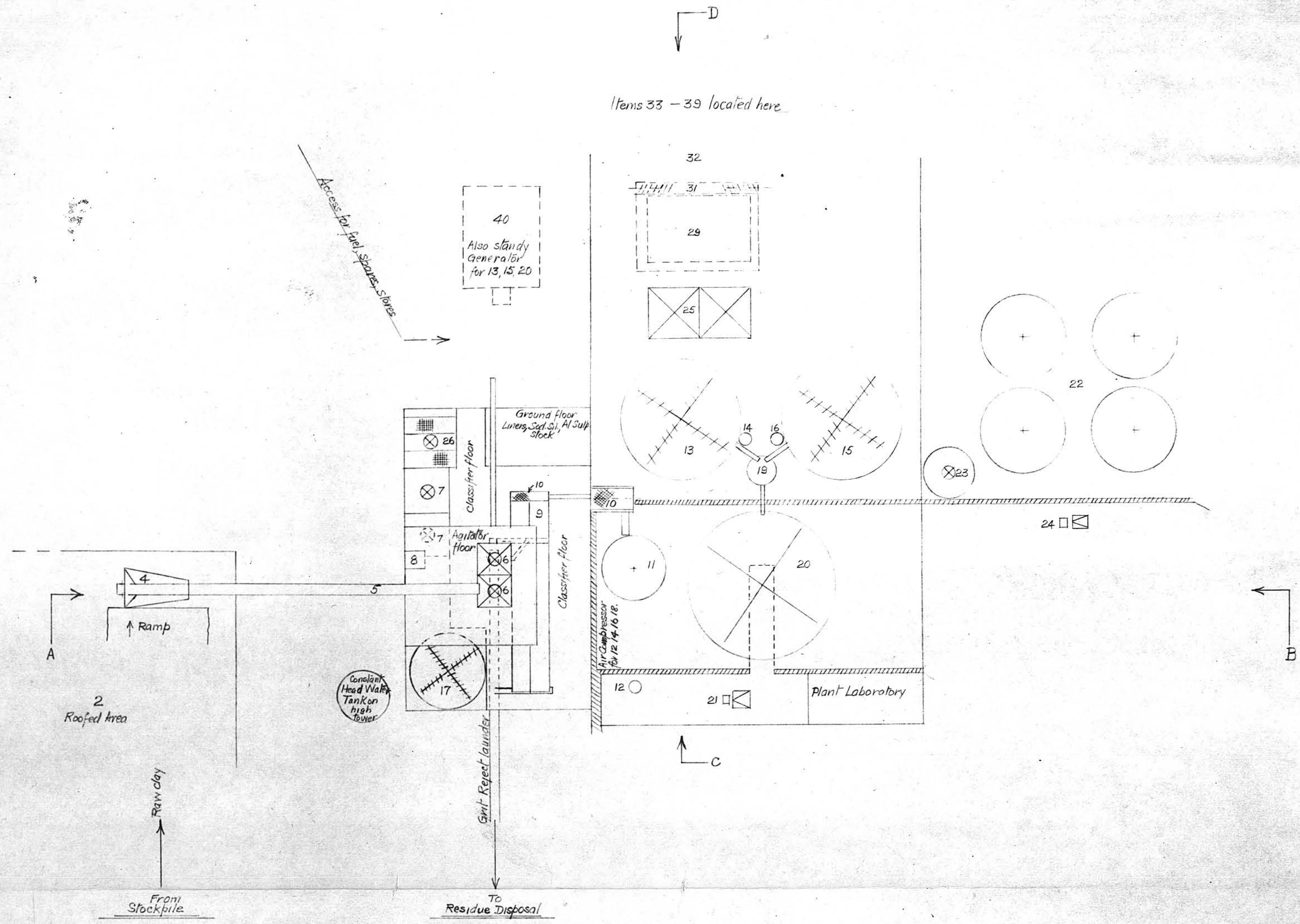
	Product	Solids Tons Hour	Water Tons Hour	Pulp Vol. C.ft	Pulp Solids %	Unit	Function
1	Raw Clay					Stockpile	Stores 18000 cub. yds. raw clay at plant for winter supply
2	"					Roofed ditto	Stores 600 to 1000 cub yds dry raw clay for prolonged wet weather
3	"	17	4·3			F.E. Loader	Hourly delivers 17 cub yds raw clay 20% moisture to 4
4	"					Feed Hopper	Feeds clay on to conveyor at uniform rate
5	"	"	"			Belt Conveyor	Elevates raw clay to agitators
	Raw clay	17	4·3				
6	Dispersant	·08	0·8			Agitators	Slurry and disperse the raw clay
	New Water		6·2				
	Discharge	17	11·3	638	60		
7	Dispersant					Dissolving and	Twin dissolving agitators, pump and head tank for metering
8	"					feeding plant	sodium silicate to slurry in 6
	Feed	17	11·3	638	60		
9	New Water Spray		2·7			Rake	Coarse degritter discarding 45% of the raw clay as clean quartz grit +60 mesh
	Return Water		2·0			Classifier	and overflowing −60 mesh gritty kaolin slurry.
	Grit Reject	7·5	2·2				
	Slurry Overflow	9·5	13·8	626	40		
10	Feed	"	"	"	"	Trash Screen	Fixed screen or a trommel eliminating vegetable litter and roots from slurry
11	"	"	"	"	"	Agitator	Surge tank cushioning final degritters against variation of their feed
12	"	"	"	"	"	Pump	Meters and elevates slurry from 11 into 13
	Feed	9·5	13·8	626	40		
13	Overflow	2·85	5·16	225	36	Hydroseparator	N°1 Fine Degritter overflowing degritted kaolin slip, underflowing gritty kaolin residue
	Underflow	6·65	8·64	401	43·5		
14	Feed	"	"	"	"	Pump	Meters underflow residue out of 13 and elevates it into 15 for retreatment
	Feed	6·65	8·64	401	43·5		
15	Overflow	2·85	5·16	225	36	Hydroseparator	N°2 Fine Degritter overflows second crop of kaolin slip, underflows grit residue still containing some usable kaolin
	Underflow	3·8	3·48	176	52		
16	Feed	"	"	"	"	Pump	Meters residue out of 15, delivering initially to residue disposal but in future probably to 17
	Feed	3·8	3·48	176	52		
17	New Water		2·0			Hydroseparator	Future unit to wash out remaining usable kaolin from 15 underflow if and when operating figures show
	Overflow	?	2·0				favorable economics
	Underflow	3·8	3·48	176	52		
18	Feed	"	"	"	"	Pump	To remove washed clean −60 mesh sand residue underflow from 17 and deliver to residue disposal
19	Feed	5·7	10·3	450	36	Screen	Special 250 mesh screen removing last traces of grit and vegetable trash from sale kaolin slip.

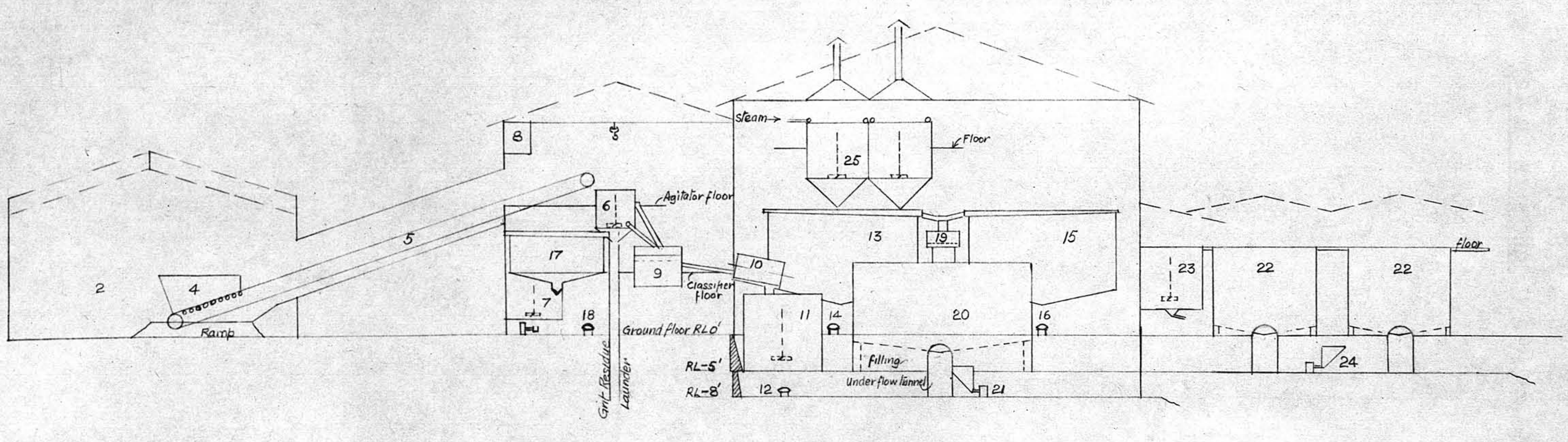
		Solids Tons Per Batch	Water Tons	Pulp Vol. cft	% Solids		
20	Degritted Kaolin Slip	13·3	24	1050	36	Agitator	Storage agitator permitting degritting and finishing sections to handle the kaolin slip at different rates over different time spans
21	" "	"	"	"	"	Pump	Transfers batches of slip from 20 to bleach tanks 22 or direct to heating tanks 25
22	" "	"	"	-	-	Bleach Tanks	Batch agitators for chlorine bleaching of the kaolin slip on a continuous basis
23	Bleached slip					Balance Tank	Small agitator to prevent consequences of irregular filter operation from disrupting bleaching cycle
24	" "	5	9	400	36	Pump	Transfers batches of bleached slip from 22 to heating tanks, or to 23, or back to any one of 22 or to residue disposal
25	" "	"	"	"	"	Heating Tanks	Agitator tanks for heating and flocculating batches of bleached slip on a continuous cycle. Steam ½ tons/batch, 12 tons day
26-28	Flocculant					Flocculant Plant	Agitator for dissolving aluminium sulphate with pump for delivering to elevated feed tank
		Per Hour					
29	Hot Flocced Slip	1·66	3·5	150	32	Filter	Eliminates 60% of the water content of the hot slip, producing filter cake consisting of kaolin 54% moisture 46%
30	" " "					Pump	Gear pump returning occasional filter tank overflows back to heating tanks
31	Filter cake	1·66	1·42		54	Distributor conveyor	A probable requirement for mating filter cake discharge length to shorter feed length of dryer
32	" "					Dryer	Probably indirect steam heat type for eliminating remaining water from flocculated filter cake. Steam 60 tons daily
33	Dry kaolin	1·66				Elevator conveyor	For transferring hot dry kaolin from dryer to bin 34
34	" "					Bin	Totally enclosed bulk storage bin, under negative pressure to prevent escape of dust. Simultaneously deaerates the kaolin
35						Exhaust Fan	Exhausts air entering bin with the kaolin
36						Cyclone	Small high velocity cyclone to recover kaolin dust and discharge clean bin exhaust air to atmosphere.
37	" "					Bagging Machine	Automatic machine filling and closing paper bags
38	Bagged kaolin					Fork lift truck	For handling palleted bagged kaolin into store
39	" "					Store	Bulk store for bagged kaolin
40	Steam		3			Generator	Steam for heating filter feed and drying filter cake

SOUTH MT CAMERON PAPER KAOLIN PROJECT
FLOWSHEET OF TREATMENT

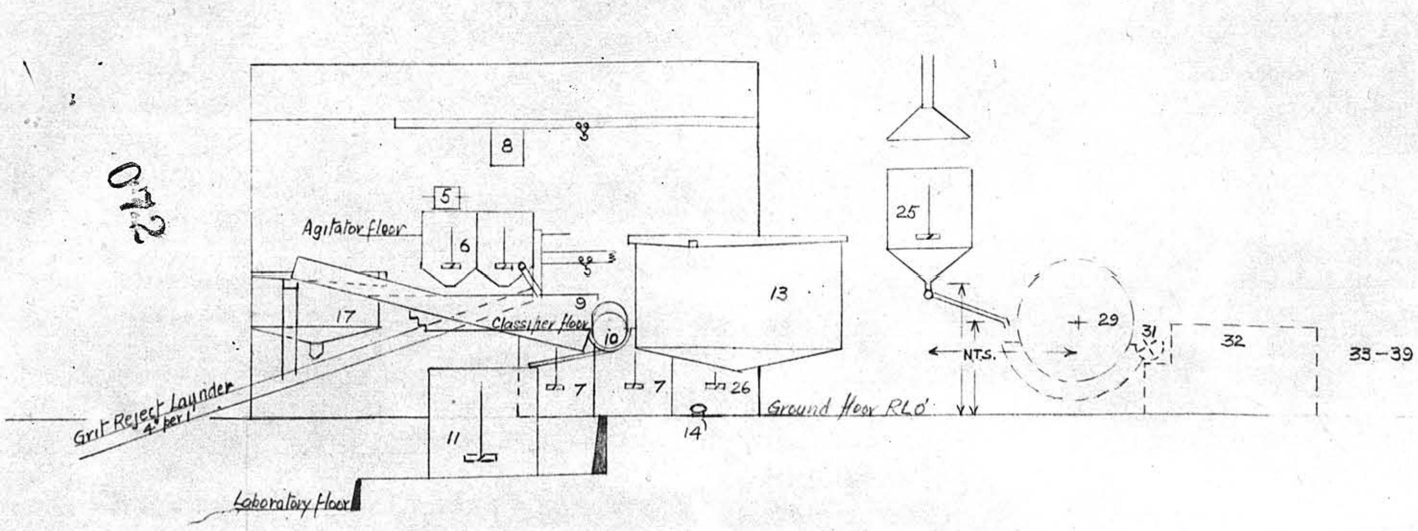
071

W.Kempp
Feb. 1963





ELEVATION A-B



ELEVATION C-D

072

SOUTH MT CAMERON PAPER KAOLIN PROJECT

ARRANGEMENT OF TREATMENT PLANT.

Scale 1/8" = 1'

W.H. Cevolop (signature)
Feb 1963

284133

SCHEDULE 12 (e)

284134

ABERFOYLE HOLDINGS
Establishment of Operations in Terms of Time and Finance
DORSET KAOLIN DEVISION

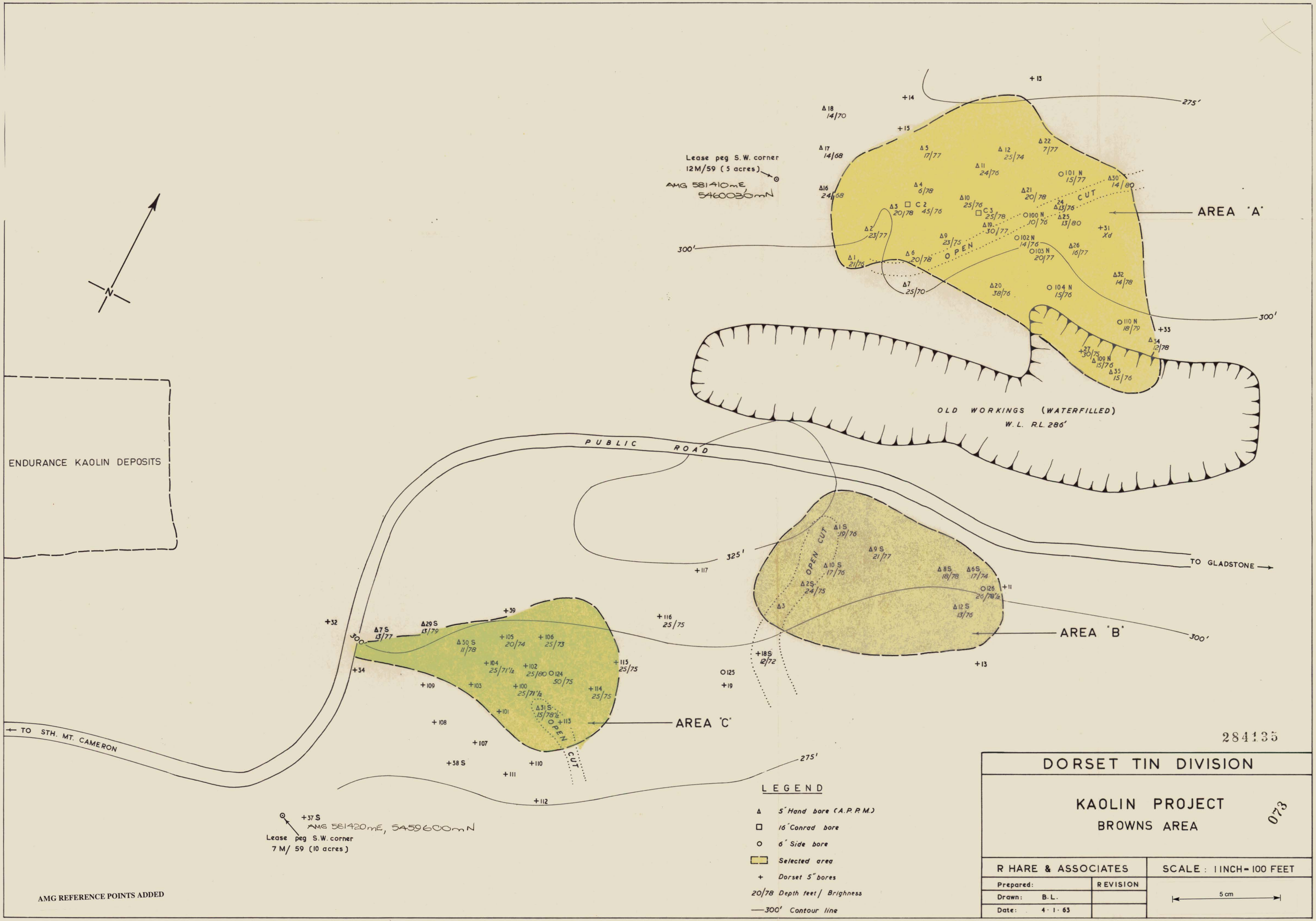
PROGRAMME	1963 QUARTER ENDING				1964 QUARTER ENDING				1965 QUARTER ENDING				CAPITAL	REMARKS
	March 31	June 30	Sept. 30	Dec. 31	March 31	June 30	Sept. 30	Dec. 31	March 31	June 30	Sept. 30	Dec. 31		
FINANCING													—	
PLAN DESIGN	2000	2000											4,000	
TENDERS CALLED													—	
SITE PREPARATION		2000											2,000	
WATER SUPPLY		1000											1,000	
POWER SUPPLY		1000											1,000	
ROADS		1000			PRODUCTIVE OPERATIONS								1,000	
PLANT & ERECTION			20,000	50,000	22,000								92,000	
BUILDINGS				4 000									4,000	
HOUSING			11,000										11,000	
MINING EQUIPMENT				5500	5000								10,500	By arrangement Dorset Tin
CONTINGENCIES & STORES					15,000								15,000	
ACQUISITION OF TITLES					8,500								8,500	
WORKING CAPITAL					10.000								10,000	
EXPENDITURE	2000	7000	31,000	59,500	60,500								£160,000	
QUARTER ENDING	31.3.63	30.6.63	30.9.63	31.12.63	31.3.63									

002

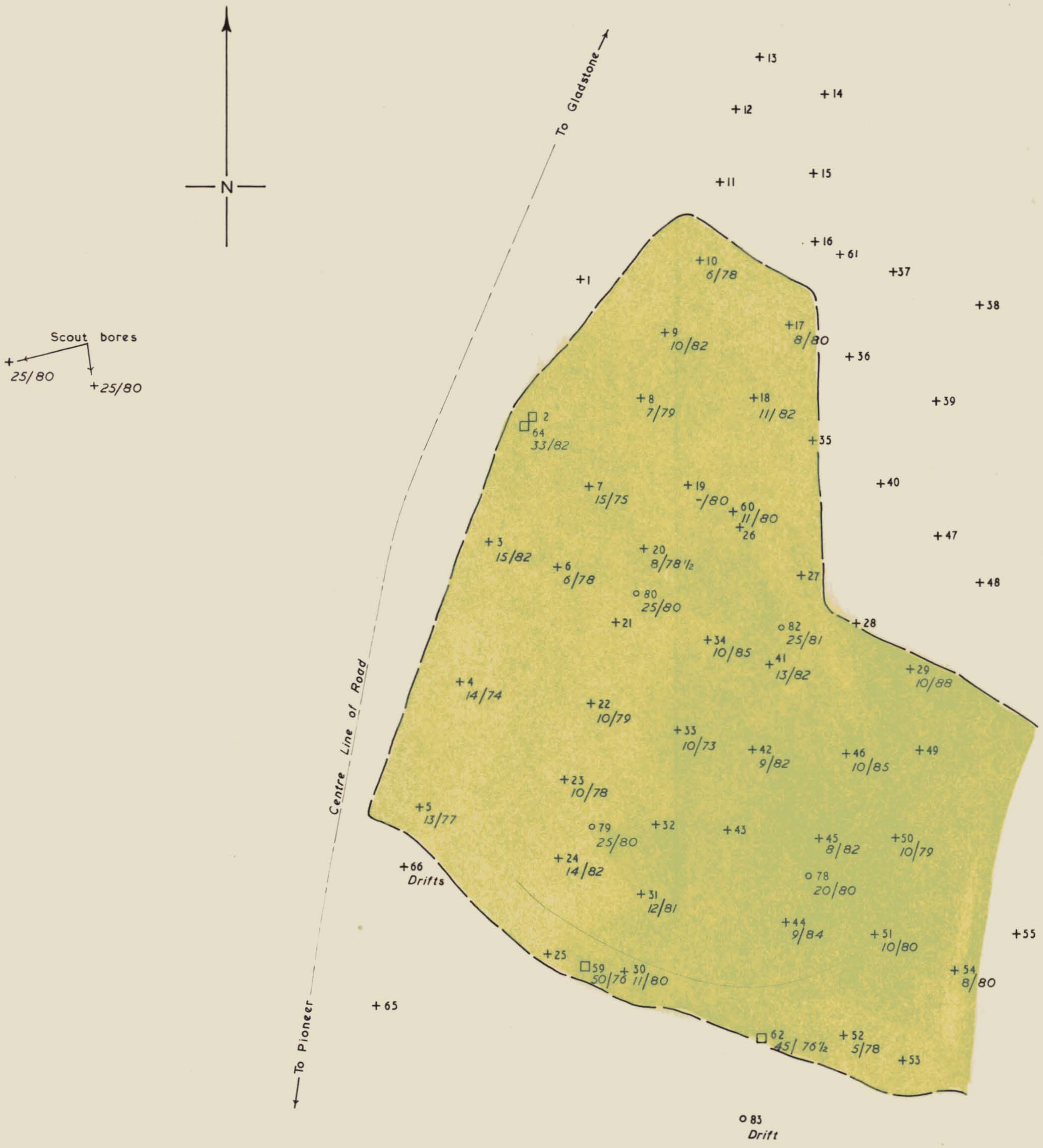
Note — Housing — £11 000 takeover of Houses from Dorset Dredge

—·— Indicates commencement of productive operation
▬▬ Indicates period of time for each segregation

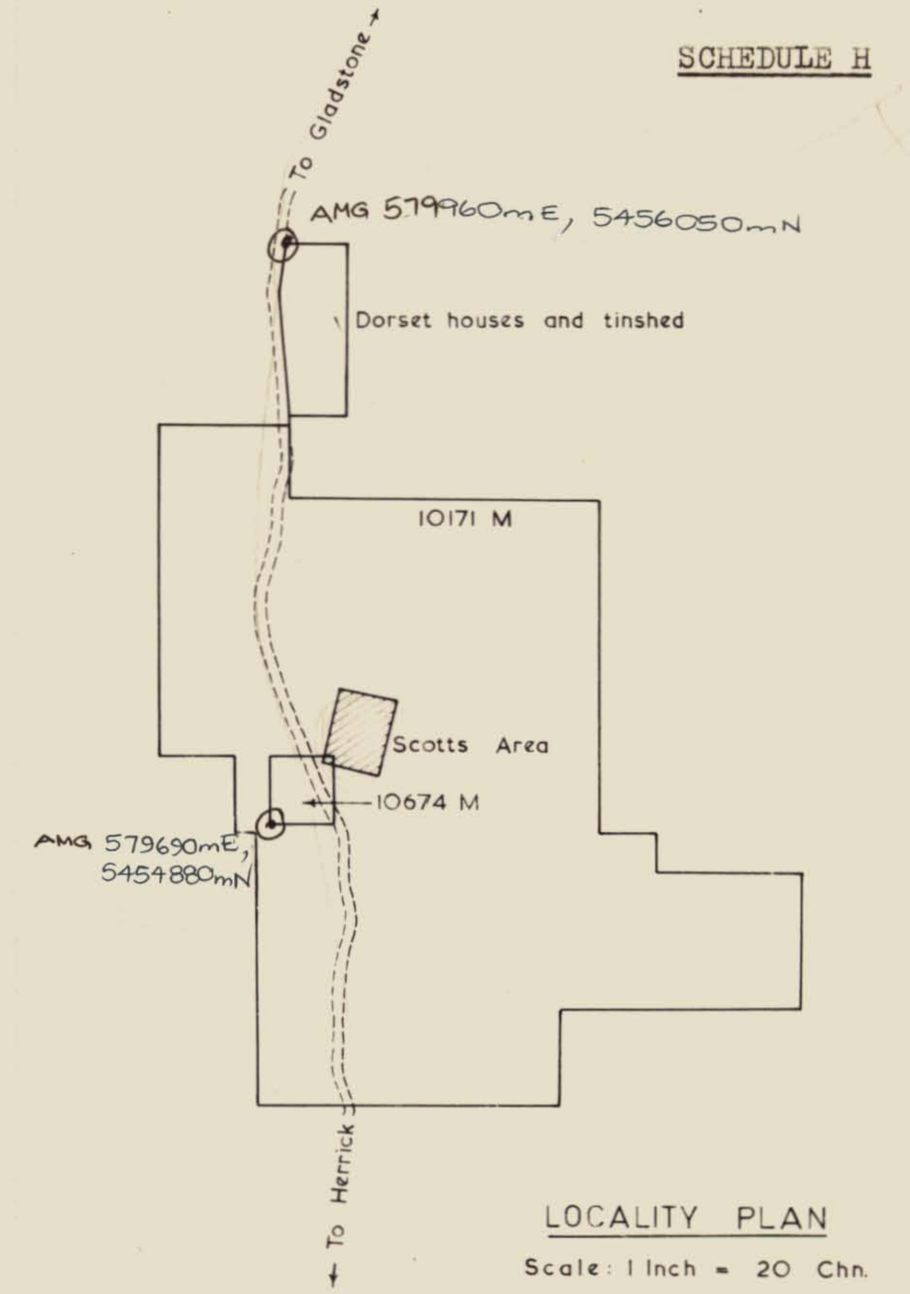
H. Keith [signature]
M.A.I.M.M.


DORSET TIN DIVISION
KAOLIN PROJECT
BROWNS AREA
R HARE & ASSOCIATES
SCALE : 1 INCH = 100 FEET
Prepared:
Drawn: B.L.
Date: 4 · 1 · 63
REVISION
5 cm
073
284135
LEGEND
5" Hand bore (A.P.P.M.)
16" Conrad bore
6" Side bore
Selected area
Dorset 5" bores
20/78 Depth feet / Brighness
300' Contour line
AREA 'A'
AREA 'B'
AREA 'C'
OPEN CUT
OLD WORKINGS (WATERFILLED)
W. L. R.L. 286'
PUBLIC ROAD
TO GLADSTONE
TO STH. MT. CAMERON
ENDURANCE KAOLIN DEPOSITS
N
Lease peg S.W. corner
12M/59 (5 acres)
AMG 581410mE
5460030mN
AMG 581420mE, 5459600mN
Lease peg S.W. corner
7 M/ 59 (10 acres)
275'
300'
325'
AMG REFERENCE POINTS ADDED

SCHEDULE H



AMG REFERENCE POINTS ADDED



LOCALITY PLAN

Scale : 1 Inch = 20 Chn.

L E G E N D

+ 5" Hand bore

□ 6" Side bore

o 16" Conrad bore

Selected area

14/82 Depth sampled feet / Brightness

DORSET TIN DIVISION

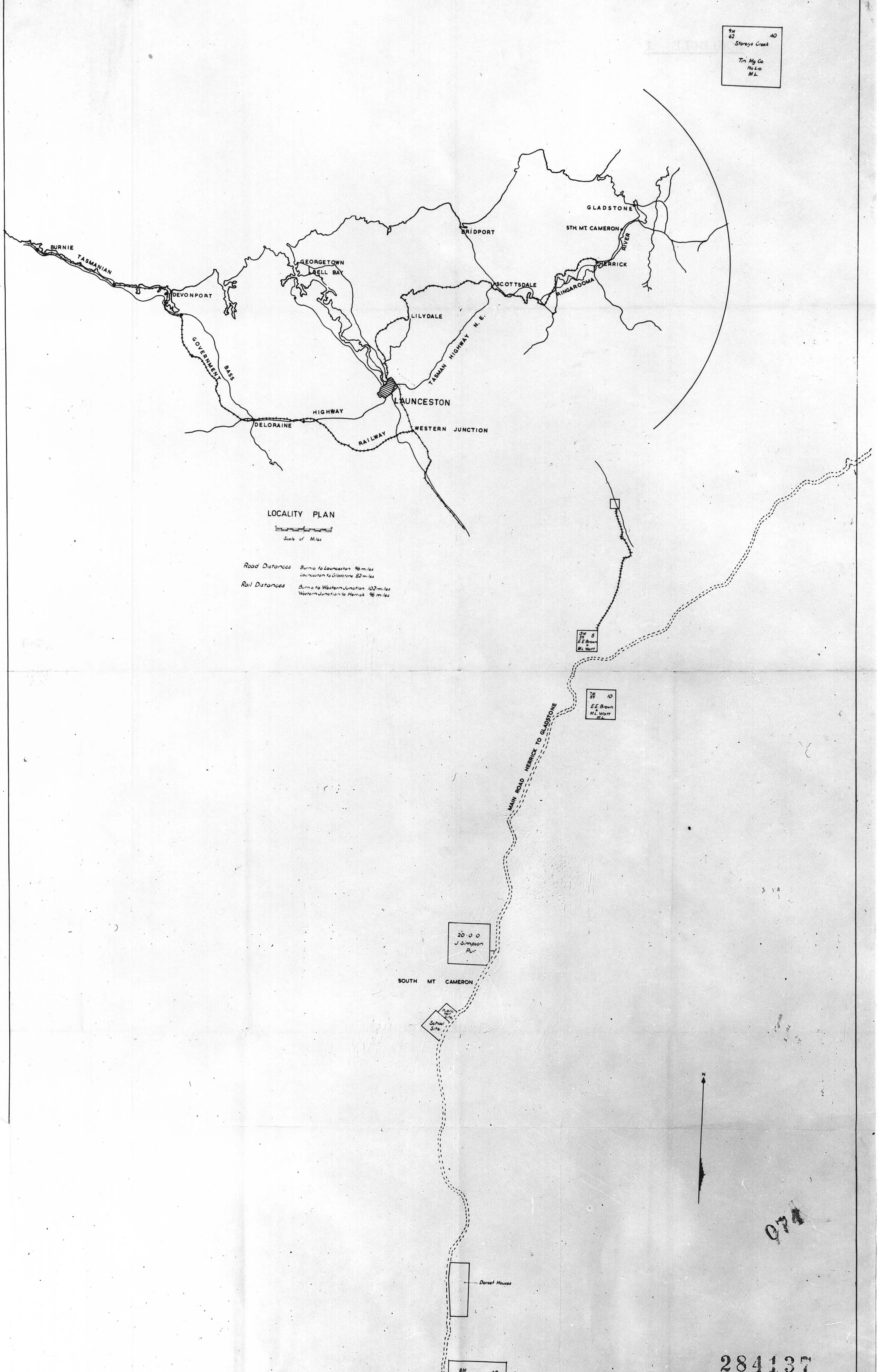
284136

KAOLIN PROJECT

001

SCOTTS AREA

R. HARE & ASSOCIATES		SCALE : 1 INCH = 100 FEET
Prepared :	REVISION	5 cm
Drawn : B. L.		
Date : 2 · 1 · 63		



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PAUL A PHELPS 1 YORK ST	AUTHORISED SURVEYOR LAUNCESTON	11/62

CLAY PROJECTS FOR DORSET TIN

SCALE	SHEET NO	DATE	
~~10 CHNS~~ 1"	1 OF 1	7th March 1963	Paul A Phelps

5 cm